DEVELOPMENT OF LONGWALL MINING SYSTEM FOR STEEPLY INCLINED COAL SEAM AND ITS COUNTERMEASURES IN MONGOLIA

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July 2023

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by

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ABSTRACT

Mongolia has extensive geological coal resources, which are estimated to be approximately 162 billion tons. Total coal production is averaging between 55Mt and 60Mt with 35Mt annually being exported to China. Due to unfavorable geological and economic conditions, especially steep coal seams, many open pit coal mines in the South Gobi regions are intended to transition from open pit to underground. It is inevitable and essential to present proper guidelines on the design and countermeasures for the steep coal underground mining method in transition areas. According to this background, developing an underground mining system, particularly the longwall mining method, for steep coal seam extraction is considered an effective means of enabling rapid transition and securing coal production in open pit coal mines in some regions closer to China. This is an urgent issue that should be addressed through this study. Therefore, in this study, the design and maintenance measures for the optimal gateroads, underground excavations and longwall panels in steep coal seams are investigated by means of numerical analysis.

Chapter 1 introduces the current situation of the coal industry in Mongolia and the rest of the world as well as applicable mining methods and their principles and criteria. Moreover, the advantages and disadvantages of the longwall mining method and a brief introduction of support systems for gateroads and longwall face have been presented. And inevitably, research objectives, expected contribution to the coal mining field and thesis outlines were included.

Chapter 2 aims to present some essential conditions of analysis, such as in-situ stress conditions and possible lateral stress ratios that can be applied in numerical simulation and rock mass quality assessment. According to borehole data and other geological reports of the Narynsukhait coal mine, the rock mass rating system has been used to evaluate rock mass strength in certain quantities. Based on tentative assessment work, RMR and GSI values were calculated as 45 and 40, respectively. These values show that rock mass can be classified as fair rock and class III. Also, some information on the research field has been referenced to in this section.

Chapter 3 investigates the stability of gateroads and effectiveness of passive and active type supports ahead of longwall panel retreat. In terms of longwall face direction, “along the strike” was proposed and gateroads and panels are driven horizontally, which can tackle many problems related to stability issues. Various seam dips, mining depths, in-situ stress conditions and rock deterioration approaches have been considered in analysis and the effects have been discussed in each section. The effect of several seam dips is negligible, while mining depths significantly affect the displacement and fracture zone. Moreover, the rock mass condition has dramatically worsened in the case of lateral stress ratio increases from 0.5 to 2.0. It shows that the higher stress ratio and horizontal stress regime result in high displacement and large fractures around underground openings. Although passive type support, such as a steel arch, is more effective than active type support, like a rock bolt, steel arch support has adverse impacts in which it resists the airflow through the openings and elevates the accumulation of methane gas and spontaneous coal combustion. Additionally, installation is relatively expensive and complex.

Chapter 4 discusses the effects of the various geotechnical and geological conditions and face support characteristics in three different phases; ahead of the panel retreat, half of the panel retreats, and the entire panel retreats. Similarly, seam dips do not substantially influence gateroad stability, while the mining depths and lateral stress ratios are significant. Otherwise, due to the mechanical properties of the intact rock mass used in laboratory experiments, the analysis results seem relatively stable and safe. Therefore, in order to make the numerical analysis conditions similar to actual site characteristics, rock mass deterioration by reducing the GSI value by 25% and 50% has been utilized, resulting in dramatically worsened mining conditions. Moreover, the fully mechanized shield support and choke shield support are not appropriate for steeply dipping longwall coal mining because of their massive weight and technical difficulties associated with stabilizing in an inclined face. Thus, hydraulic prop, a common example of manual support, was employed in face excavation. Even though initial conditions of stress, depth and seam dip were varied, the beam axial stress and z-displacement of prop support installed in the face with certain spacing have been distributed systematically along the face direction. However, after deteriorated rock mass was applied in analysis, the condition of the lower end of the longwall face drastically worsened, and beam axial stress increased substantially.

Chapter 5 discusses the relationship between the orientation of gateroad and major horizontal stress. In a variety of historical data, when the gate entries intersect the major horizontal stress direction, a high frequency of serious rock failure and damage has been observed. So, designing an adequate gateroad layout that involves the orientations and high horizontal stress direction can be a key factor in improving safety issues associated with ground control and stability, as well as enhancing the economic diversity of underground coal mining projects, especially in terms of longwall mining. Three different phases were considered; the k ratio is 1, the gateroad intersects and is parallel to the major horizontal stress direction. In that case, high horizontal stress intersecting with the gateroad dominantly results in serious deformation and large fracture zone propagation in the surrounding rock mass, particularly on the roof and floor. For the Narynsukhait coal mine, it is more appropriate that the longwall face direction can be in-between “along the strike” and “down the seam dip”. By applying this design of face direction, there will be numerous advantages that can be followed upon. Although the gateroad becomes inclined, the inclination of the longwall can be reduced and provide an opportunity to mitigate the adverse impacts of the orientation of the gateroad and high horizontal stress.

Chapter 6 concludes the results of the research.

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Тэртээх тэнгэрээс хүүгээ хараад баярлаж суугаа аавaараа "Хүү минь мундаг шүү" гэж бахархуулахыг хүссэн хүсэл бас ижий, аав хоёртоо өгсөн амлалтаа өөрийн хичээл зүтгэлээрээ хичээнгүйлэн биелүүлсэндээ үнэхээр баяртай байна.
Аавaa, ээжээ хүү нь та хоёртоо хязгааргүй их хайртай шүү...

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CHAPTER ONE. **INTRODUCTION**

Coal is a vital source of power generation and steel-making production worldwide. Massive coal demand will remain increased for the next few decades until humans evolve the high-advanced green energy technologies or find more alternative and replaceable fuel sources. The efforts to reduce CO_2 emissions and diminish the negative environmental impacts result in a decline in solid fuel use. Coal consumption is decreased quickly because many countries and governments struggle not to use coal as the primary energy source. Coal theoretically accumulates in two similar geological environments in terms of its geomorphology and aerial characteristics; on the bottom of the sea and the lake. Most of the coal deposits had accumulated on the bottom of the lake in the midland, particularly in Mongolia. Regional and local tectonic influence is relatively high in coal in western Mongolia. It results in steep, vertical dip angles and intensively faulted, jointed, and folded structures in coal seam measures. Companies usually pay attention to extracting coal in favorable geological and geotechnical conditions, including relatively gentle and horizontal coal seams, which are feasible for the economy and easy to be mined with current mining technologies and techniques. Moreover, ground control can be significantly simple. However, abundant coal resources in some unfavorable geological conditions, including steep and vertical coal seams in highly faulted and folded structures and high in-situ stress regimes, must be considered a great contribution to the country's consistent economic development. Due to weak economic development and a poor infrastructure network in the high potentiality of coal regions, the appropriate guidelines for steep and vertical coal seam deposits nearby the southern border of Mongolia have to be introduced. Realizing the static and dynamic stress conditions and uncertainties associated with ground behavior are key components for the guideline and develop steep coal mining projects.

1.1. Trend of coal production

The high-priority demand for green and eco-friendly energy sources, expanding the current renewable energy technology development, and a massive up in clean power sources recently resulted in a significant decline in coal production and consumption for the last few years. One instance, the principal analyst of coal research Viktor Tanevski, noted that the outlook for capital investment in coal (ex. China) is for a near 20% fall in 2022 with a substantial reduction in thermal coal (Source: *Wood Mackenzie*).

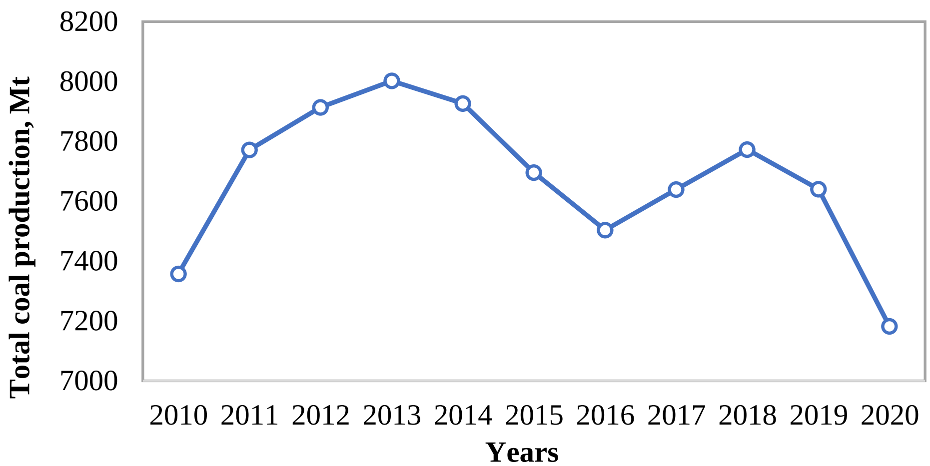


Figure 1.1. Total coal production in the world (Source: www.iea.org)

Meanwhile, many high-developed and developing countries tend to reduce their coal consumption for energy production and tackle CO_2 emissions from solid fuel-based technologies regarding environmental protection and global climate change obligations in front of the world. Numerous coal investors and financial institutions face inevitable challenges whether they stop or continue their investments in coal mining projects. In Figure 1, the statistic of the world's coal production for the last decade has a 10% fall that will keep decreasing, according to the reports released by some prominent coal research institutions. However, we must carefully consider China's coal market fluctuation and energy policy regarding the Mongolian coal sector. China's coal imports have ranged from 200 Mt to 300 Mt in recent years, regarding the statistical data announced by www.statista.com. The lowest coal import was registered in 2010 as 163.1 Mt. It has drastically risen from 2010 to 2013, and the coal import reached 327.02 Mt, the highest in history. Since then, China's coal import and market have remained close to 304.1 MT in 2020, as shown in Figure 2. Mongolia reached a peak amount of coal export of 36.8 Mt in 2019, and all coal products were only exported to China by transporting heavy trucks through ground roads. However, this statistic has dramatically declined since 2021 due to the COVID-19 pandemic spreading throughout the whole world. The coal sector is the major industry that occupies a large portion of the state budget income, and it accounted for over 45% of the total exporting products' income in Mongolia for the last decade.

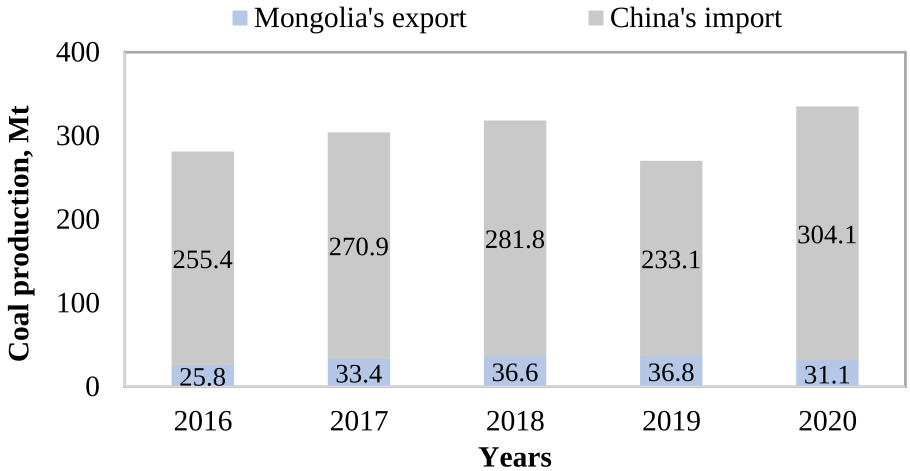


Figure 1.2. Comparison of Mongolian coal export and China’s import
(Source: www.mmhi.gov.mn)

On the other hand, increasing coal export also negatively affects the environment and wild nature near open-pit coal mines and coal transportation roads in some provinces where almost 90% of the coal is extracted, elevating the conflict between local people and the government policy. Therefore, underground mining is another option to support extending active coal mine lifetime near the borders in the future. This method’s advantages can also be answers against serious environmental negative impacts that have to be reduced. Furthermore, the economic pit limits of open-pit coal mines will be reached as soon as if production intensity is still high. We inevitably have to search for other effective, highly efficient and eco-friendly solutions to continue the project lifetime until they mine out the possible mineable coal resources and reserves as feasible with applicable mining methods economically and technologically.

1.2. Literature review

There are several mining methods that we can apply for dip coal seams to mine. However, the two most common types of underground mining methods for coal are summarized in this section.

1.2.1. Underground coal mining methods

Two typical underground coal mining methods have usually been applied to extract coal that cannot be mined out by surface mining methods; those are room and pillar mining and longwall mining. These two mining methods have their efficient advantages and disadvantages. However, some modified versions of these two mining methods have evolved during their development periods, such as high reach single-pass longwall, longwall top coal caving and multi-slice longwall for thicker seams, room and pillar with blasting gallery as well as types of combining with caving mining methods, underground coal gasification, and hydraulic mining can be preferably mentioned. Coal is produced from multiple open entries or rooms in a regular grid arrangement for the room and pillar mining method. The remnants of coal in the room act as pillars to support the immediate roof, which also controls the global ground response of the rock medium (*Brady & Brown, 2007*). One of the remarkable advantages of this method is that the remaining coal pillars as support can reduce surface subsidence risk significantly and make it easy to access previously mined rooms regardless of technical and technological limitations.

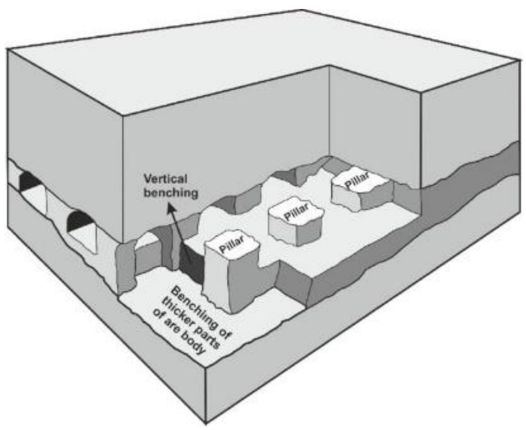


Figure 1.3. Elements of a room and Pillar mining method (*Hamrin, 1980*)

The considerable increases in longwall productivity are due in large part to the rapid uptake of computer-based technologies for automation and monitoring; improved reliability and performance of longwall mining equipment; and the adoption of plant management and loss control principles, leading to both increased rates of production and reduced labor requirements (*J.M. Galvin, 2016*).

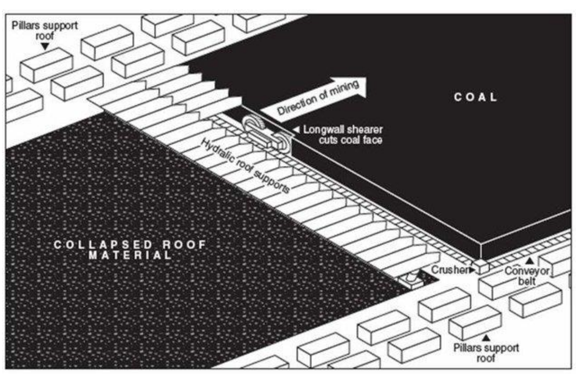


Figure 1.4. Typical layout of longwall mining method

Therefore, the longwall mining method is still a primary underground mining method for coal, and it will continue consistently in the future from an engineering point of view. Serious ground control difficulties can usually be associated with supporting and maintaining gateroads, cut-throughs, and pillar ribs while the panel retreats under the induced stress regimes. Thus, the stress redistribution and stability of the longwall face and gateroads are studied in this research with different seam dips and mining depths increasing up to 700m in three cases.

1.2.2. Background of the longwall mining method

Longwall mining has a large productive efficiency, good coal recovery rate, less remnant of coal, high-efficient economically and is a safer underground mining method for coal extraction. A typical longwall mining configuration schematic is illustrated in Figure 1.5. In contrast, large initial investment, capital costs and requiring longer time for equipment installation and transferring in the face are critical disadvantages of this method. Depending on the direction of the panel excavation, there are two types of basic longwall mining as follows:

- Longwall on the advance
- Longwall on the retreat

Longwall on the retreat is the most common among these two methods. It involves driving one, two and three gateroads down both flanks of a panel to its extremity and then connecting these two sets of gateroads (*J.M. Galvin, 2016*).

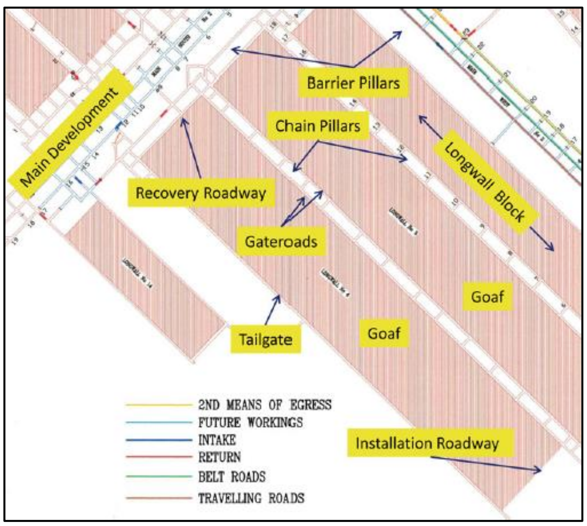


Figure 1.5. A mine layout and terminology associated with longwall mining (*J.M. Galvin, 2016*)

Longwall mining starts with developing inclined accesses or vertical shafts to reach the coal seam. These tunnels are referred to as the main roadway, and gateroads are developed to the end of the longwall panel alongside and connected by excavating cross-cuts or cut-throughs. Subsequently, the coal seam retreats that the mining direction goes from back towards front or barrier pillar that is the portion of coal remained as support to maintain the stability of the main roadway throughout mining life. The longwall face is supported by an array of shield supports including a prop. The legs of the shield support or prop support moved forward with extension and shrunk by hydraulic force to follow the coal shearer drum while the face retreats. For low-strength coal or weak coal measures, in such cases, an extremely large fracture zone occurs around gate entries that require structural supports or strata countermeasures to maintain safe roadway serviceability for a long period. *Brady and Brown (2005)* categorized the stability issues related to ground control of longwall mining as follows:

- *Stability of main roadway and gateroads:* to provide access to the face for equipment, materials, coal transportation and ventilation.
- *Chain or rib pillar (barrier) stability:* to protect the gateroads of the unmined panel from the effect of the ongoing panel excavation.
- *Longwall faces stability:* maintain a safe working condition and ensure a smooth continuous panel retreat operation.
- Backfilling at gob area: to reduce the abutment pressure in the longwall face and gob which tends to result in large aerial surface subsidence and positively affect safety.

According to several educational books and engineering research articles, the geotechnical issues and the stability problems associated with major elements of longwall mining can be divided by author's points in order of developing sequence as assumed in this thesis:

- Development of the main roadway and gateroad
- Dimension of barrier and chain pillar
- Longwall panel design that includes a caving mechanism
- Selection of the appropriate support system
- Surface subsidence

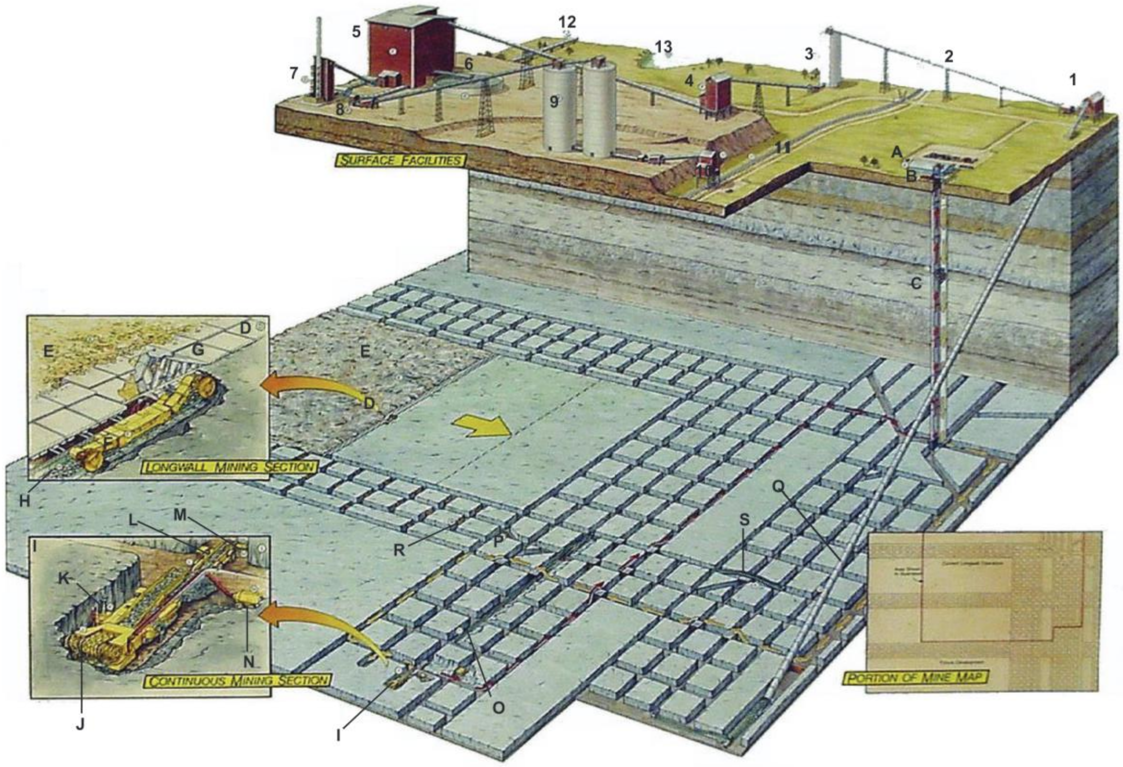


Figure 1.6. Overall view of the longwall coal mining

On the other hand, (*Syd S. Peng, 2008*) defined that major ground control factors in longwall mining design list the following issues:

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

Ground control design became the priority part of mine design whenever a new longwall mine or a new panel in an ongoing mine was planned, thereby ground control is set up as an essential factor in the success or failure of that mining project (*Syd S. Peng, 2020*). Although the entire design of longwall mining for steep coal seams is a severe task for optimizing the interaction between each phase of mining developments in detail, this study usually aims to underline the stability issues related to gateroad and longwall face and the selection of appropriate support system as well as to deepen the perception of the stress concentration clearly during or after the panel excavation.

1.2.3. Vertical stress distribution

Conceptually, a softer coal seam is sandwiched between the relatively stronger roof and floor rocks and loaded by the weight of overburden. Stress is uniformly distributed in the coal seam under such conditions. When gateroads are developed, the equilibrium conditions are destroyed due to the presence of openings (*Syd S. Peng, 2020*). Circumstances can be explained as static stress regimes converting into dynamic stress conditions after openings are set. Vertical stress concentration is schematically illustrated in Figures 1.7 and 1.8.

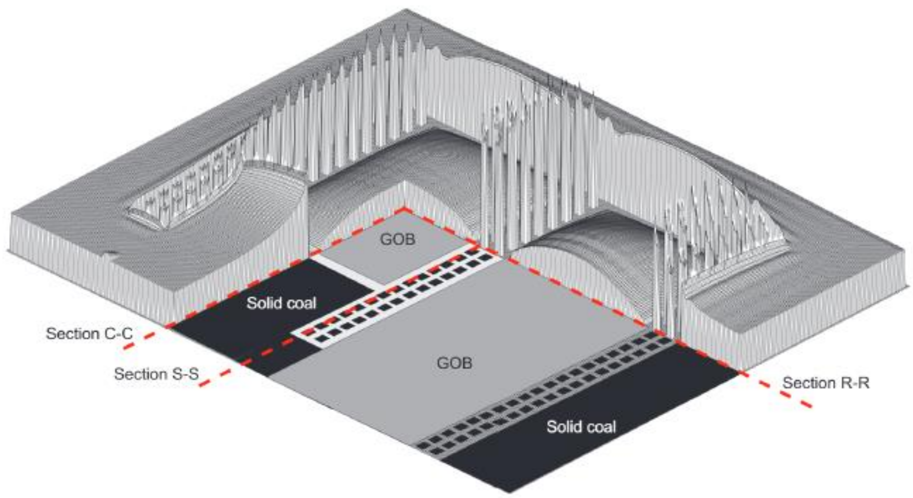


Figure 1.7. Three-dimensional view of vertical stress distribution

Vertical stress concentration in a coal seam exponentially decreases when it takes some distance from the edge of the gateroad or face. As illustrated in the Figures above, the highest vertical stresses occur adjacent to the excavation area on the rib side of openings. Strict interactions between the abutment stress and dimension of pillars, panels, gateroad and main roadway can be realized that the boundary of the pillar and panel width should be out from the high abutment stress zone. It can be understood that the gateroad adjacent to the next unmined panel should be situated far, and the chain pillars have to be taken as large as they can adequately reduce the high impact of the induced stress generated from the previously mined panel. Based on these circumstances, ground control management can be predictable when a geotechnical engineering study is made accurately prior to the beginning of the mining development.

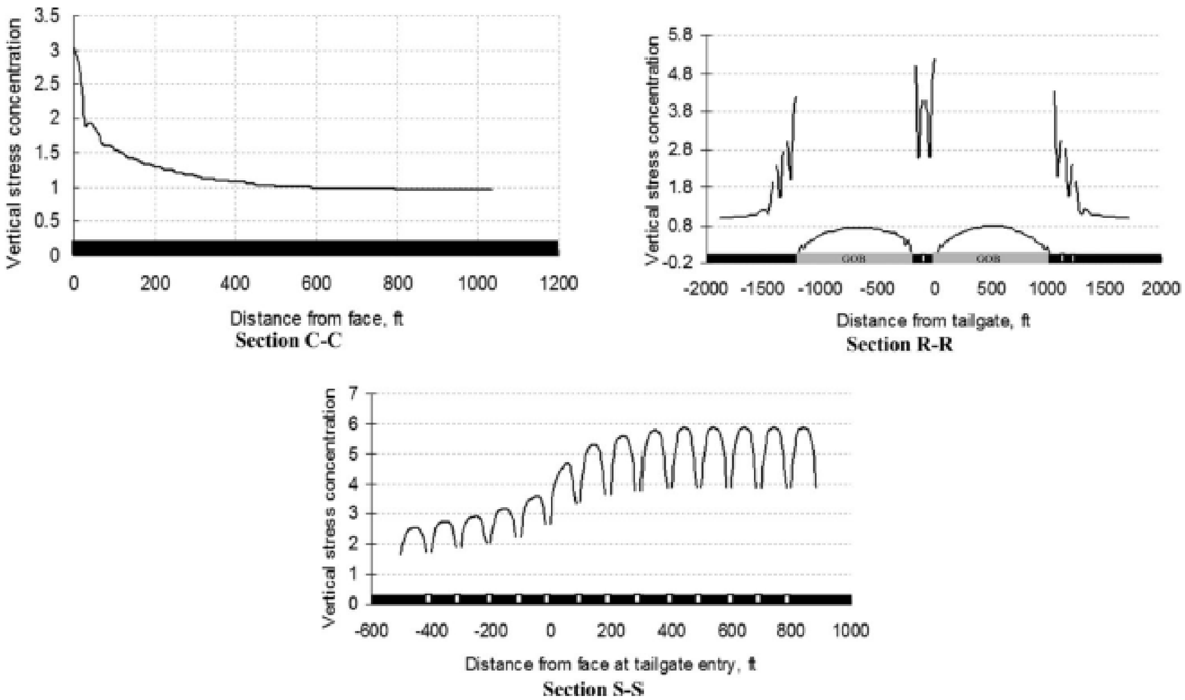


Figure 1.8. Vertical stress distribution at sections
C-C, R-R and S-S (ft ≈ 0.3m)

1.2.4. Gateroad layout

Single-, two-, three- and four-entry systems are developed depending on the technical and economic merits of each system as well as strata conditions. Each entry system has their specific advantages and disadvantages. For example, single-entry system can reduce the overall disturbance of virgin coal area and eliminate chain pillars, cross-cuts and intersections, in turn increase coal extraction ratio. Moreover, it improves initial development rate and is less time/energy-consuming and good control for surface subsidence. However, other uncertain questions and disadvantages related to safety, especially possible escapeway for personnel when dealing with serious rock falls in the gateroad, gateroad stability, enough cross-sectional area of gateroad for large techniques and facilities as well as ventilation problems that whether ventilation can be good enough to convey necessary air quantity and dust/methane control. As referred by (*Syd S. Peng, 2020*), the two-entry system has been practiced in only three states in the USA due to specific ground conditions that are deep and overburden consists of thick sandstone strata that tend to hangover and induce very large abutment pressures in and around longwall panels. In order to reduce the hazards such as coal bumps or sudden outbursts, the two-entry system using a yield pillar has been proposed effectively.

Also, this system contains few numbers of chain pillars compared to three- and four-entry systems. In addition, it can be more feasible to answer the uncertain questions associated with a single-entry system. But this system has widely been applied in rest of the world. Nevertheless, the three-entry system is the most popular, and it accounts for 81% of all longwalls, while the four-entry system accounts for 14% (*MSHA – Mine Safety and Health Administration*) in the USA. Gateroads are usually rectangular shaped and easily driven by the traditional continuous miner or high-advanced miner bolter equipment, as shown in Figure 1.9.

a. Traditional continuous miner b. Miner bolter

Figure 1.9. Typical techniques for gateroad development

The high efficiency of this miner bolter is that this machine is designed to do cutting and bolting simultaneously. Therefore, an advanced miner bolter significantly improves the advance rate of gateroads, and bolts are installed immediately or soon after cutting before the surrounding rock's convergence in the roadway.

1.2.5 Chain pillar design

In the two-entry system, there is only one row of chain pillars between the panels, while in the three-entry system, there are two rows of chain pillars, respectively. Most pillars are square or rectangular in shape, but some are diamond-shaped to facilitate the labor movement in Figure 1.10. Moreover, pillar size is equal or unequal; if the pillar length is much larger than the width, the footage and the number of cut-throughs relatively reduce (*Syd S. Peng, 2020*). Also, there are two pillar types: yield pillar and stiff pillar. Chain pillars in the gateroads are left behind without being mined, although pillar mining concurrent with longwall retreating mining had been done routinely in the past (*Hendon, 1998*). In recent times, numerical modeling and analyzing approaches have been very useful in determining pillar design and size. Although there are numerous aspects of pillar dimension, it is obvious that larger pillar results in low coal recovery. On the other, productivity becomes less economically compared to the appropriate or small dimension of the pillar.

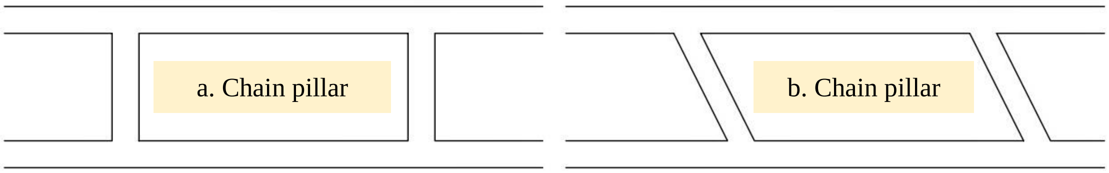


Figure 1.10. Two types of chain pillar layout
(a. Rectangular and b. Diamond-shaped)

1.2.6. Supports in entry and roadway

Several types of supports and reinforcements have been widely used to tackle convergence in tunnels and underground openings and are designed according to geological and geotechnical complexity. However, these countermeasures can be divided into two main types in terms of their structures (*Galvin, 2016*) as follows:

- Support (including standing support, props, timber chocks, cementitious chocks, steel arches and sets, and pillars)
- Tendon support and reinforcement (wooden dowels, rock bolts and cable bolts with/without grouting)

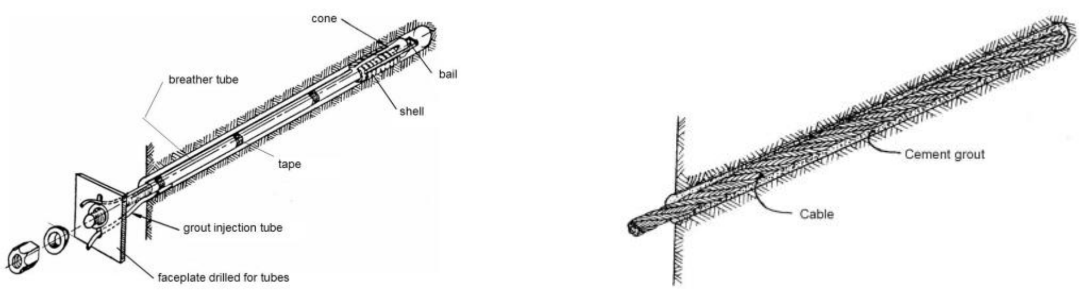
a. Prop support

b. Timber chocks

c. Cementitious chocks d. Steel arches

Figure 1.11. Standing support

Besides, roof support in longwall gateroads can be divided into two main categories with respect to driving equipment: primary and secondary or supplementary supports. The primary support is mainly rock/cable bolts with different spacing and lengths. On the other hand, the secondary supports are installed additionally long after the continuous miner's cutting process and are usually longer rock/cable bolts in order to increase the surrounding rock strength and bearing capacity. However, due to ground conditions and surrounding rock strength, combining several supports and reinforcement is much more reliable and safer than applying single-type support. Numerous examples and engineering practices have been experienced with respect to the selection of support systems in the entry and main roadway.



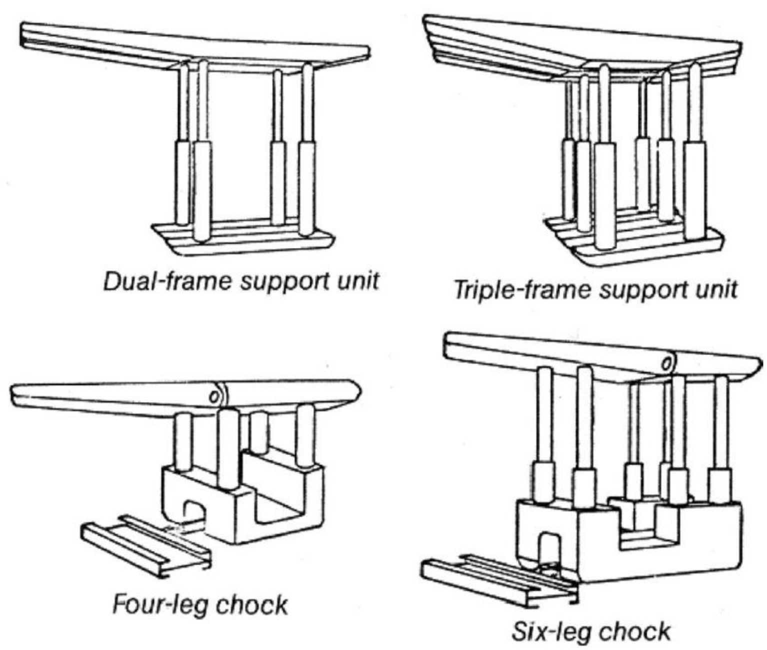
a. Rock bolt with grouting b. Cable bolt with grouting

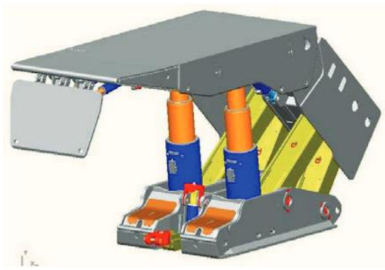
Figure 1.12. Tendon support and reinforcement

A range of important factors that should be taken into consideration as critical points that can maintain stress relief includes the orientation of mine layout, sequence of mining, sequence of secondary extraction, dimension of pillars and barrier pillar, excavation height, formation of stress regime, support and reinforcement type, pattern, and spacing density.

1.2.7. Face supports

For the main element of longwall mining, face stability is a crucial issue, and therefore, it must be taken into consideration, and some types of face support have been introduced. Although longwall mining history dates back to the beginning of the nineteenth century, powered support, for example, shield support, was not introduced until the mid-twentieth century. Since shield support was introduced in longwall mining, numerous nations and mining engineers have rapidly invented remarkable innovations and achievements that have been practiced in underground longwall coal mines worldwide. Shield support mechanization can be categorized as manual, semi-mechanized and fully-mechanized. The computer-based automation rate for all components of face excavation has already moved forward at a high-advanced level and has become safer and extraordinarily effective. Most longwall mining employs self-advancing powered support at face excavation in horizontal and gentle dip coal seams. However, it is relatively difficult that heavy-weight powered shield support to be employed directly at steep and very steep coal seams due to technical problems associated with the high risk of tilting, sliding and gliding along the steep floor. So, hydraulic prop support can be more effective and facilitate many miners' in-field problems related to transportation and installation. Therefore, some detailed explanations of hydraulic prop support and its operational features are summarized in the "Longwall mining" book by *Syd S. Peng (2020)*.





Shield support

Chock shield support

Figure 1.13. Face supports

A hydraulic prop or cylinder, or ram is the power source that converts fluid power into mechanical power. There are single- and double-acting and single-, double-, and triple-stage telescopic hydraulic cylinders. The leg size ranges from 125-480mm in diameter.

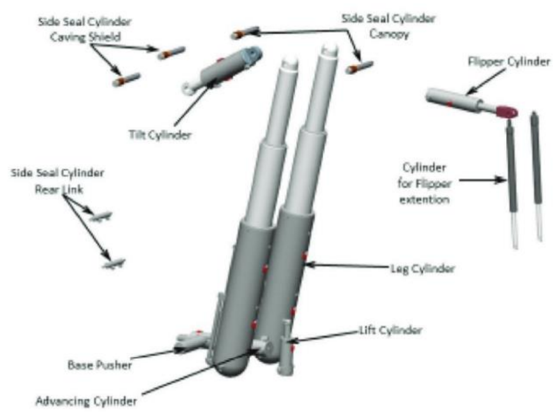


Figure 1.14. Components of hydraulic cylinders used in the shield

The extension of a single-stage (single-telescopic) hydraulic cylinder is adequate, but the collapsed height limits its operation range, making it unsuitable for coal mining because coal seams vary considerably from panel to panel or even in a panel. The major advantage of a two-stage (double-telescopic) cylinder is its small collapsed height. In order to further increase the ratio of extended to collapsed height, the hydraulic legs in a shield are mounted in an inclined position. The larger the inclination from the vertical, the larger is the height adjustment ratio (Figure 1.14) Prevention of fluid leakage over the service life of the prop or cylinder, or ram is extremely important. Among those, the piston seal is the most critical because it has to withstand high pressure and the effect of additional friction and force. Both the top and bottom of the prop are spherical in shape for easy connection to the canopy and base plate, respectively, and robust load transmission from the canopy to the leg and leg to the base plate. The top-stage working pressure is greater than that of the lower stage due to the difference in piston areas. Thus, the top-stage piston is the most susceptible to leakage.

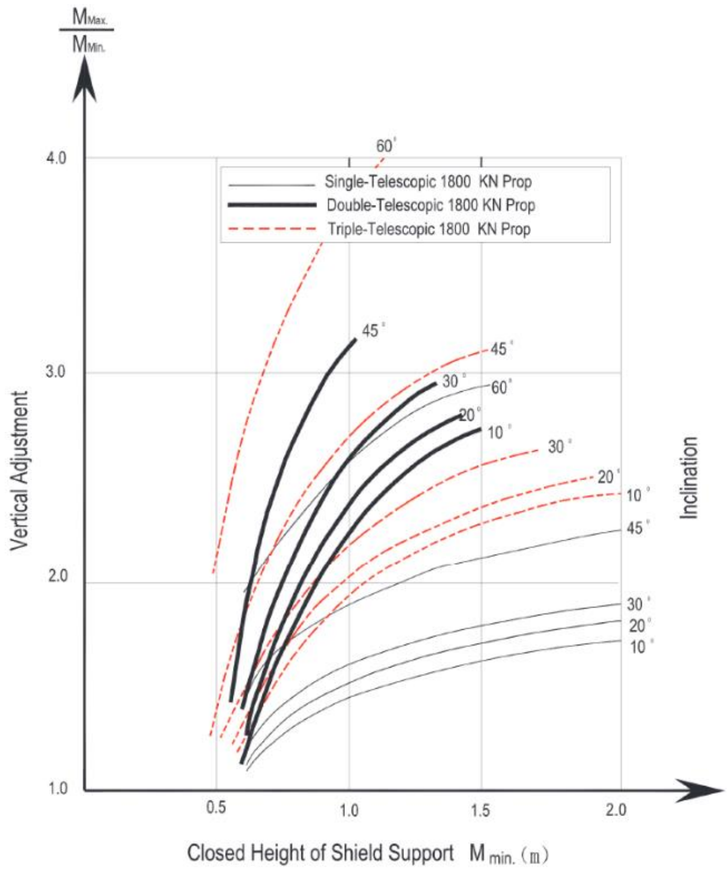


Figure 1.15. Height adjustment of hydraulic legs varies with the number of telescopes and angle of inclination

1.2.8. Surface subsidence issue

During longwall mining, when the gob exceeds a certain size, overburden strata movement reaches the surface and forms a low area of limited size above the gob. This is called the subsidence basin and it changes surface topography. As a result, it also affects surface water and groundwater flows in some areas. This is a serious issue for the local residents living nearby the mining area and when the public gets involved, surface subsidence provokes much emotion and the mining company usually gets in cooperation trouble with local people. Therefore, in order to avoid escalating conflicts between the company and the public, the mine location and selection of the mining system are very important to be considered for this circumstance. As explained in section 1.2, the room-and-pillar mining method is suitable for applications near residential areas.

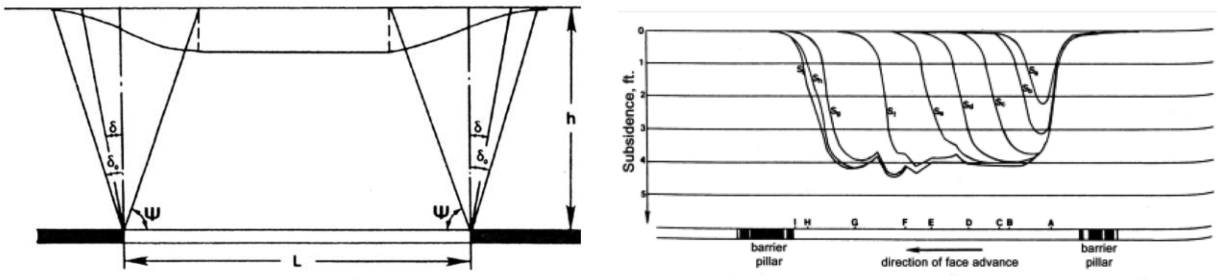


Figure 1.16. Surface subsidence cross and along the panel excavation

Syd S. Peng (2020) determined that under normal conditions, when the face advances for a distance of approximately one-sixth to one-third of the seam depth from the setup room, depending on the seam depth and the physical properties of the overburden strata, the movement of the overburden strata reaches the ground surface, whereupon the surface also begins to deform. Many researchers and engineers have been discussing better comparing the different analyzing methods, such as empirical and numerical. These two methods are the most common in engineering design, research, and even geotechnical engineering. The empirical method is usually based on a large database of numerous case histories. Also, this method has been evolved by comparing the study works of many projects that have similar geological and geotechnical environments. The critical disadvantage of it is that this method is strictly dependent upon field measurements and cannot be applied to the different mining sites which have extraneous geological and mining conditions (*Elashiry, Gomma and Imbaby, 2009; Xu et al., 2013; Pongpanya Deth, 2017*)

In recent times, the numerical simulation method has widely been employed in various engineering research fields, including mining and geotechnical engineering and is highly developed with respect to high-capacity computer-based tools. Many initial or basic concepts of mining site conditions and uncertainties associated with ground control can be predicted using a series of numerical simulations sitting in front of the typical personal computer, without any supercomputers or calculating equipment. This is an exceptional achievement for engineering science and significantly influences all-around improvement and diversity. Numerical analysis can be used in any mining conditions, even if reliable input data such as mechanical properties of rock types, information on primitive stress regime, and geological stratigraphic column are available (Villegas, 2008; Iwanec & Carter, 2016) The finite difference method with continuum modeling software FLAC^{3D} and the finite element method is high-advanced software for geotechnical engineering research. So, this software had usually been used to do a series of numerical analyses, an important part of this thesis work.

1.3. Applicable mining methods for steep coal seams

As mentioned in previous section, there are two common underground coal mining methods. However, corresponding the geological conditions such as seam dip angle, thickness, coal quality and physical characteristics of coal, several modified versions of longwall mining method and sub-level caving mining method have been utilized worldwide. Unfortunately, the task of finding the best method that satisfy all the specific requirements for efficient mining development is impossible. Therefore, those methods have briefly been introduced with illustrations as follow:

a) Full seam thickness extraction longwall mining with drilling and blasting method

Coal is extracted by drilling and blasting, and the face is supported by hydraulic props and sliding beams. The roof rock strata are fully caved, and this method is more suitable for extracting coal seams that are gently inclined, and the thickness is around 3m or less. (Figure 1.17)

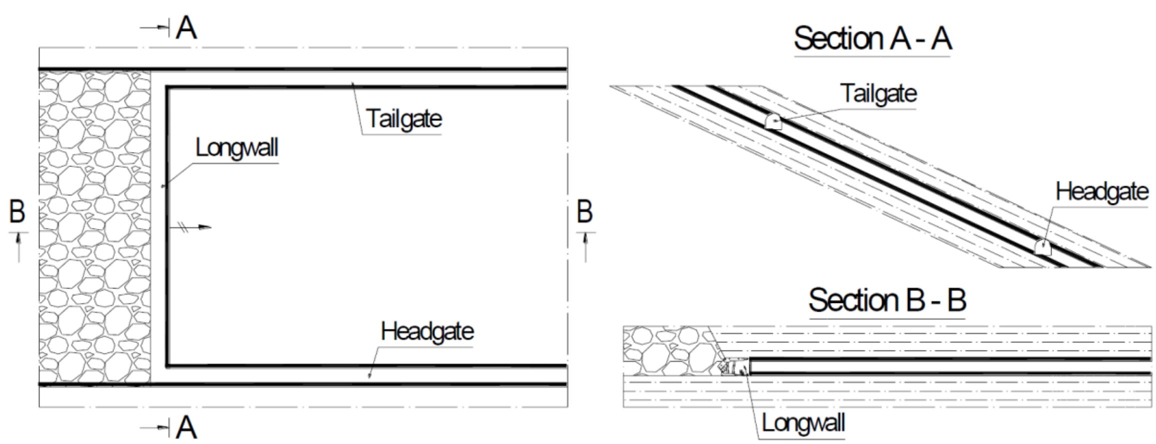


Figure 1.17. Scheme of longwall full seam thickness extraction with drilling and blasting

b) Full seam thickness extraction longwall mining with cutting machine

The general scheme of the coal extraction process is the same as the previous method. A face-cutting machine is utilized for coal extraction instead drilling and blasting method. Many factors can be improved by employing a cutting machine such as panel retreat advancing rate, low energy consumption, cost-effectiveness, and relatively small adverse impact of vibration than the drilling and blasting method does.

c) Longwall top-coal caving mining

This method is usually applied for thick coal seams that cannot be covered by the height of typical face supports. The longwall face can be extracted by a cutting machine or drilling and blasting either while the upper part of the coal is caved gravitationally and recovered from the backside of support. (Figure 1.18)

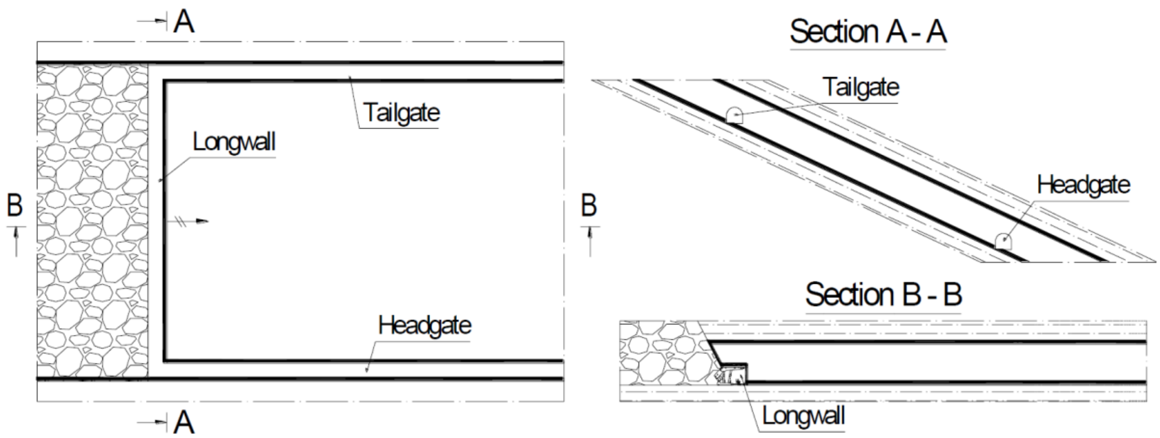


Figure 1.18. Scheme of longwall top-coal caving method

d) Self-advancing shield support in longwall mining face

Self-advancing face support was a large step for the longwall mining evolvement process. This system includes a coal-cutting machine and self-advancing shield support that has two belt conveyers located on both sides, the front and back of the support. Generally, this system is applicable for the gently dipping coal seam whose angle can be up to 30˚. Nevertheless, if it is applied in steep longwall coal mining, several measures should be taken, and operational efficiency could be decreased significantly. Due to two technical reasons that one is that face extraction advancing rate become slower because of hard rock suddenly appears in the range of longwall face and another is that low ratio of coal recovery from back of the support, the spontaneous coal combustion issue has to be taken into serious consideration. (Figure 1.19)

e) Multi-slice longwall mining

This method is predominantly applied in thick or extra-thick coal seams. A coal seam is divided into a number of slices vertically. The slices can be extracted concurrently with certain distance between the longwall faces. (Figure 1.20)

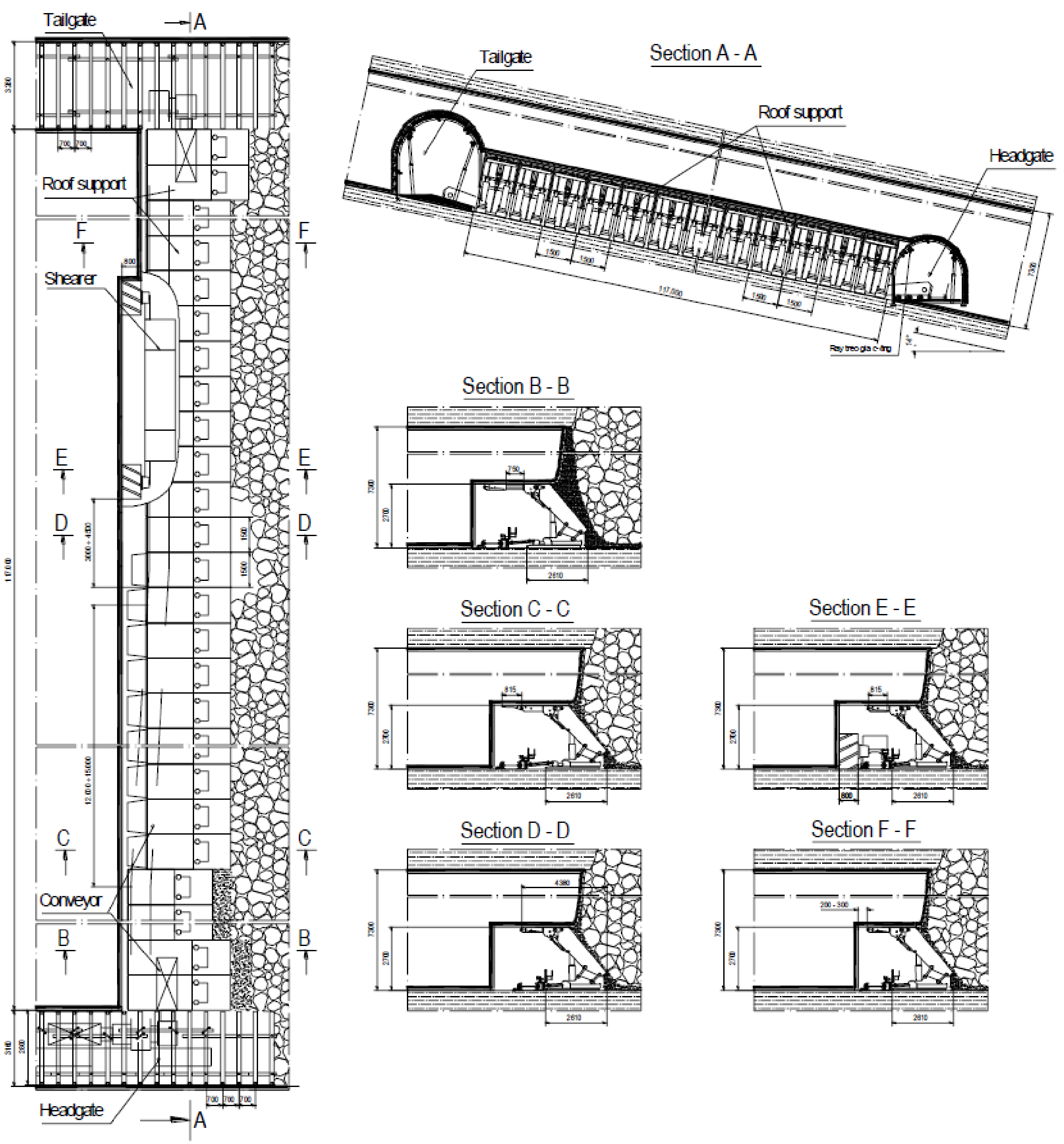


Figure 1.19 General scheme of fully-mechanized longwall mining
with self-advancing face support

f) <u>Sub-level stopping mining method for steep and vertical coal seam.</u>

This method is suitable for steep and vertical coal seams that cannot be mined by the longwall mining method. There is another considerable factor which is coal seam thickness that should be thick. A coal seam is divided into vertical blocks along the strike. Then roadway is driven near the bottom of the block, and the coal falls down gradationally after the drilling and blasting operation is done. The gate-entry can be supported by square timber chocks and hydraulic support systems. The most difficult operational problem is controlling the coal dilution and low recovery ratio because of a large amount of residual coal in gob. (Figure 1.21)

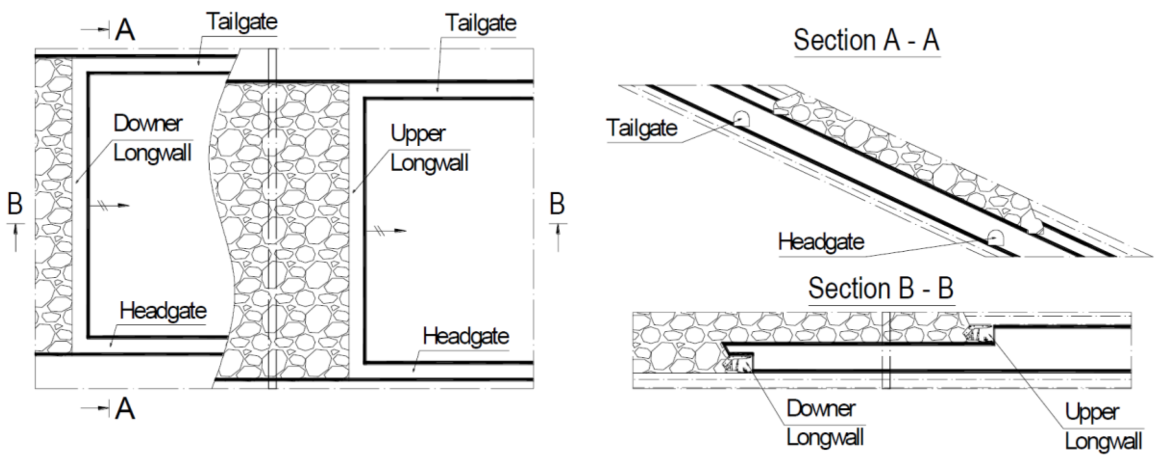


Figure 1.20. Scheme of multi-slice longwall mining method

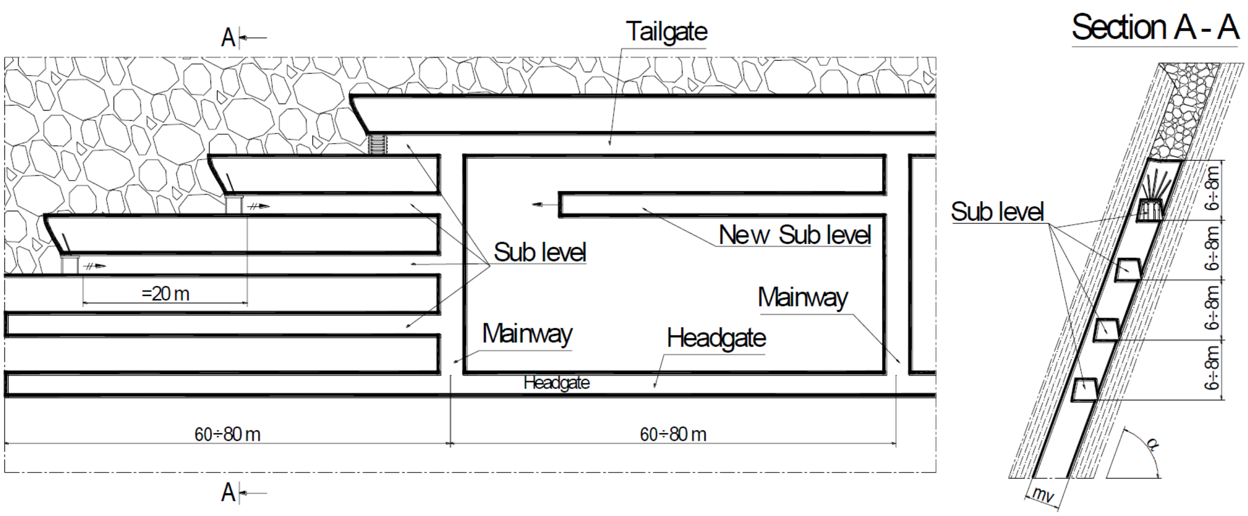


Figure 1.21. Scheme of sub-level stopping mining method

1.4. Research objectives

Longwall mining is more appropriate for gently inclined and horizontal coal seam when a dip angle is less than approximately 30 degrees under favorable geological and geotechnical conditions that include coal seams are flat, fairly uniform and the roof strata is easy to cave (*Syd S. Peng, 2020*). Nevertheless, due to technical and technological limitations, a need for well-experienced labor for steep coal mining, and a world coal market trend remain inconsistent, high-advanced, newly innovated and eco-friendly solutions must be invented. In horizontal or gentle longwall coal mining, ground control progresses are reliable and uncertainties can accurately be predicted compared to steep and vertical underground coal mining. Particularly, economic factors affecting mining operations must be considered significantly for this type of mining operations that are located in very poor developed infrastructure territories. However, Mongolia has abundant steep coal resources and reserves, especially those are discovered close to the southern border between Mongolia and China. It is an exceptional situation because transportation costs could be smaller for the large volume of coal transportation through hundreds of kilometers. Longwall mining guidelines and practices were often developed with historical data and practices from hard-rock-bearing coal mining areas with competent rock mass. Modern techniques combined with well-educated and experienced labor can easily control difficulties and risks associated with rock engineering principles. The purpose of this research proposes the following very important objectives:

1. To study the effect of various geological and geotechnical conditions, including seam dips and mining depths, in the steep coal seam mine based on numerous data and information obtained from exploration reports and field investigations.
2. To study the stability of gateroad and the effectiveness of different support systems.
3. To investigate the influences of inclined panel extraction on the ground behavior, stress conditions and stability by applying several mining conditions.
4. To recommend the appropriate guideline for steep coal seam extraction and inferences to improve the other technical high-risk consequences of the ventilation system, spontaneous coal combustion and severe in-situ stress regimes.

In order to find the approaches for those objectives, the Narynsukhait coal mine in Mongolia was selected as the research site, and input parameters for numerical analysis and geological data were obtained from the exploration work report and feasibility study for the long-term. The stability of the gateroad and the influence of different stress conditions have mainly been considered. This study has been done based on a series of numerical analyses using FLAC^{3D} and RSData software. The author firmly focused on involving many important aspects of rock engineering in steep longwall coal mining from the geotechnical point of view.

1.5. The contribution of the research

Huge hope is dedicated to that this study can contribute to enhancing engineering knowledge and broaden the notions of the underground longwall mining for steep coal seam in the following ways from diverse engineering points of view:

- Understanding the importance of the initial and correct data collection process and realizing the specific geological and geotechnical features of underground steep coal seam mining is a higher priority in the rock engineering research field. Also, steep coal seam mining at greater depth is a new and challenging task for the coal mining sector in Mongolia. Although we have several historical experiences of underground coal mining in the past, engineering research for this type of mining is currently insufficient.
- The applicable underground mining methods for steep coal seam and their exclusive advantages/disadvantages should be studied and introduced in a wide range for those active/inactive coal mines that aim to continue their project lifetime and keep gaining economic profits for a longer time. For this reason, it believes that this research can be a reasonable beginning of further studies for great achievement.
- As long as geology is very complex and diverse, the geotechnical condition is also complicated. Based on a series of numerical analysis, results and discussions, this study is able to introduce some insights and principles related to rock engineering principles for understanding the uncertainties associated with mining operations and rock mass characteristics of steep coal seam mining.
- Moreover, it can be a fundamental for the creation of appropriate guidelines and engineering solutions for underground steep coal seam mining at greater depth under various mining conditions. By doing it, it will give a large beneficial impact on the contribution for economic development in Mongolia as the mining sector is the biggest industrial role in terms of gross domestic product and income of export.
- Finally, the last but not least wish of the author is to contribute educational knowledge for mining engineers and support their efforts that have been committed to enhancing the capacity of the mining sector.

1.6. Thesis outlines

The thesis is composed of six chapters, and all are listed with a brief description of each as follows:

Chapter 1 presents the current situations of coal production trend in the world and Mongolia. Also, it introduces the literature review of applicable underground coal mining methods for the steep coal seam seam, support systems and other relevant issues. In the end, research objectives and the contributions are proposed.

Chapter 2 introduces the background of the selected research site, the Narynsukhait coal mine, and possible in-situ stress conditions that can be applied in the numerical analysis. Moreover, tentative rock mass strength evaluation in the research site has been made using rock mass rating (RMR) system and geological strength index (GSI) assessment approaches. Drill core data and information from the exploration report were used for a rock mass quality assessment.

Chapter 3 discusses the stability of gateroad in steep longwall coal mining with various geological and geotechnical mining conditions such as seam dips, mining depths, stress regimes and rock mass deterioration. Rock mass deterioration has made using the GSI value reduction and the effectiveness of different support systems such as active and passive support types in roadway will be studied.

Chapter 4 discusses the stability of gateroad and the effects of longwall panel excavation as well as face support characteristics by applying several mining conditions, for example seam dips, depths, lateral stress ratios and deteriorated rock. Manual prop support was proposed to employ in longwall mining face. Ground respond and instability in different mining phases combining other parametric variance will be studied in detail.

Chapter 5 discusses the effects of orientation of high horizontal stress on gateroad stability. Different major horizontal stress directions are applied axially in the numerical simulation. Moreover, intermediate stress conditions were also involved to study the influence the fracture development and deformation.

Chapter 6 concludes the results of the research.

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CHAPTER TWO. **THE POTENTIALITY OF STEEP COAL SEAM UNDERGROUND MINING IN MONGOLIA**

2.1. Background

The Mongolian coal mining industry has been facing certain challenges and difficulties related to the weak infrastructure network throughout the country. Regardless of abundant coal resources in the western and northern parts of the county, this important reason puts us on a mission to change our view on the mining projects near the southern border between Mongolia and China. Besides that, a large amount of high-quality coal reserves to export is usually in unfavorable and complicated geological conditions in those regions. By 2020, approximately 33.0Bt of coal resources and reserves had been estimated for over 120 coal deposits in Mongolia. This statistical data shows that the Mongolian coal mining industry can be developed in a large scale for export and domestic use based on abundant coal resources and reserves. Since the beginning of the twenty-first century, a large quantity of coal has usually been mined by surface mining method and exported to our southern neighbor, the world's biggest coal user and producer, which is China.

Figure 2.1. The location of the biggest coal deposits in the high tectonic activity region.

Around 30 Mt of raw and washed coal has been exported to China annually for several years. It includes high-quality thermal coal, semi-soft coking coal, hard coking coal, and a small amount of anthracite, as published in the Ministry of Mining and Heavy Industry report. Approximately 98% of the coal mine is surface mines, and only 2% is a small-scaled underground coal mine in which the room and pillar mining method has applied.

An incomplete statistical record and data show that around 18% of Mongolia's total estimated coal resource explored by geological exploration is steep coal seam. The seam dip angle is greater than 19 degrees, regarding the classification of the coal deposits recommended by the Ministry of Mining and Heavy Industry. Most high-quality bituminous and sub-bituminous coal deposits are located in highly faulted and folded areas where the mountainous region has been extremely dominant and induced by local and regional tectonic activities.

Coal seam classification by seam dip and thickness in Mongolia:

By coal seam dip:	*By seam thickness:*
<3˚ - Horizontal	<0.7m – Extra-thin
4˚-18˚ - Gently inclined	0.7-1.2m – Thin
4˚-18˚ - Inclined	1.2m-3.5m – Moderate thick
4˚-18˚ -Steeply inclined	3.5m-15.0m – Thick
>56˚ - Vertical	>15.0m – Extra-thick

Seam dip and thickness range from 8˚-80˚, the average is around 43˚,
and 1.7m-60.2m, the average is around 30.7m, respectively.

2.2. In-situ stress regime

Rock engineering is a relatively young and modern scientific field. Particularly, this is entirely new in Mongolia, and rock or geotechnical engineering has intensively been taken into consideration in the mining industry for the last few years. Oyu Tolgoi, the fourth largest copper-molybdenum-gold underground coal mining project, is one of the reasons to introduce geotechnical engineering research to mining engineers at an advanced level. On the other hand, geotechnical engineering research is a minor and missing topic in which only some empirical analysis has been done to study the rock engineering field in Mongolia. Therefore, there needs to be more information or data about the in-situ conditions and lateral stress ratio in the Narynsukhait coal mine, even for other mines.

Thus, some research articles for in-situ stress characteristics conducted by Chinese engineers and researchers for years are utilized in numerical analyses creating different primitive stress regimes—generally, in-situ stress increase with mining depths. The increase rate in horizontal stress is greater than in vertical stress in shallow mining depth, and this phenomenon is similar and proven in many practices and site investigations worldwide. With increasing depth, horizontal stresses decrease, and large folds and faults often result in the orientation of the maximum horizontal stress and magnitude. H. Kang et al. (2010) noted a systematic scheme for stress field distribution in terms of different mining depths.

Shallow coal mines (<400m): $\sigma_H > \sigma_h > \sigma_v$

Moderate deep coal mines (400m-600m): $\sigma_H > \sigma_v > \sigma_h$

From moderate to deep coal mines (>600m): $\sigma_v > \sigma_H > \sigma_h$

The ratio of the maximum horizontal stresses to vertical stress is usually between 0.5 and 2.0 in coal districts in China and the Inner Mongolian coal districts. In-situ stress measurements were conducted using hydraulic fracturing at 204 test sites in 49 underground coal mines.

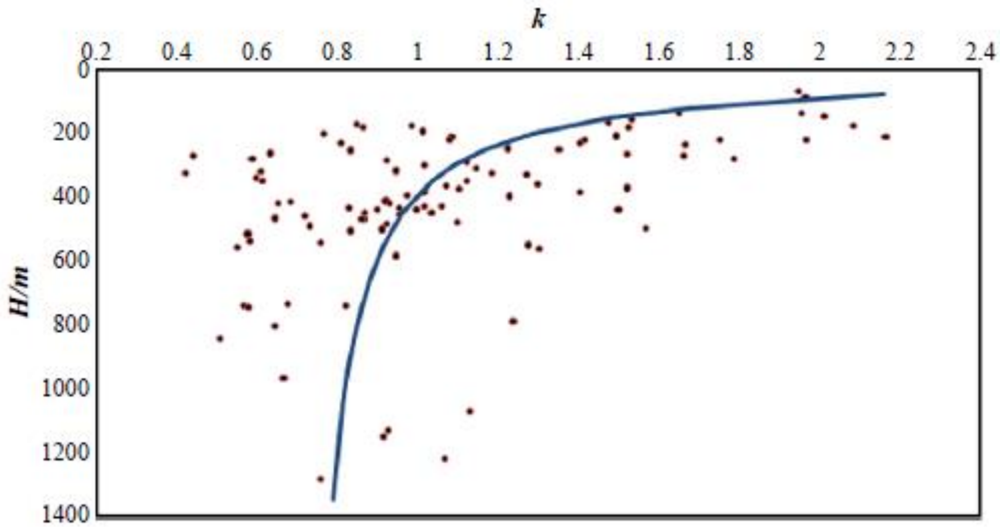

Figure 2.2. The ratio of average horizontal principal stress to vertical stress vs. depth

The results of the research can be concluded by H. Kang et al. (2010) that:

- Overall, in-situ stresses increase with mining depth in underground coal mines. In shallow mines, the increase in horizontal stresses is greater than that of vertical stress and gradually decreases with depth increase. As well as, large folds and faults often result in variations in the orientations and magnitudes of horizontal principle stresses.
- As explained above, the three typical stress fields dominated the field investigation results. For deep coal mines with a strong influence on geological structures, the $\sigma_H > \sigma_v > \sigma_h$ type can also occur.

- The lateral stress ratio tends to decrease and approaches 0.5 with increasing depth. According to the test results, the lateral stress ratio increases to 2.0 in coal mines above 400m depth and decreases to 0.5 in moderate and deep coal mines with soft rock strata. However, the lateral stress ratio is predominantly between 0.5 and 2.0.

The variation curve and phenomena of lateral stress ratio are very similar to that from Brown and Hoek in general trend. It can be reasonable that the general trend of in-situ stress conditions and lateral stress ratio can be applied in Mongolian coal mining practices. The lateral stress increasing from 0.5 to 2.0 will be utilized to understand the effects of steep coal seam mining at different depths ranging from 300m to 700m.

2.3. Rock mass classification

Several rock strength classification systems can be utilized associated with the purpose of application. Those classifications are categorized as qualitative and quantitative based on the mode of characterization (*Dyson Moses, 2018*). The type of qualitative scheme includes the Geological strength index (GSI) and Rock load while (Q). Rock quality designation (RQD), Rock mass rating (RMR), and Rock structure rating (RSR) are quantitative schemes (*Abbas & Konietzky, 2017*). In this research, the Geological strength index (GSI), initially proposed by Evert Hoek in 1994, is a widely accepted scheme for describing rock mass characterization in the field fast and simply for numerous applications and has been employed to determine GSI value accurately based on the estimation of Rock mass rating system (RMR) introduced by *Bieniawski* in *1989*. GSI and RMR systems are depicted in Figure 2.3 and Table 2.1, respectively. However, *Thomas, Ayemin and Nielsen* (*2015*) proposed that mostly, information on the rock mass conditions may be accessible only from the drill core, from which GSI cannot be assigned directly. In order to mitigate those constraints, the empirical relationship between GSI and other rock mass classification systems has been developed, particularly the RMR system was preferred to characterize the rock mass because of its simplicity and versatility. RMR value is calculated based on the formula that Singh & Goel (2011) established, as shown below.

$$GSI = RMR_{89} - 5 \tag{2.1}$$

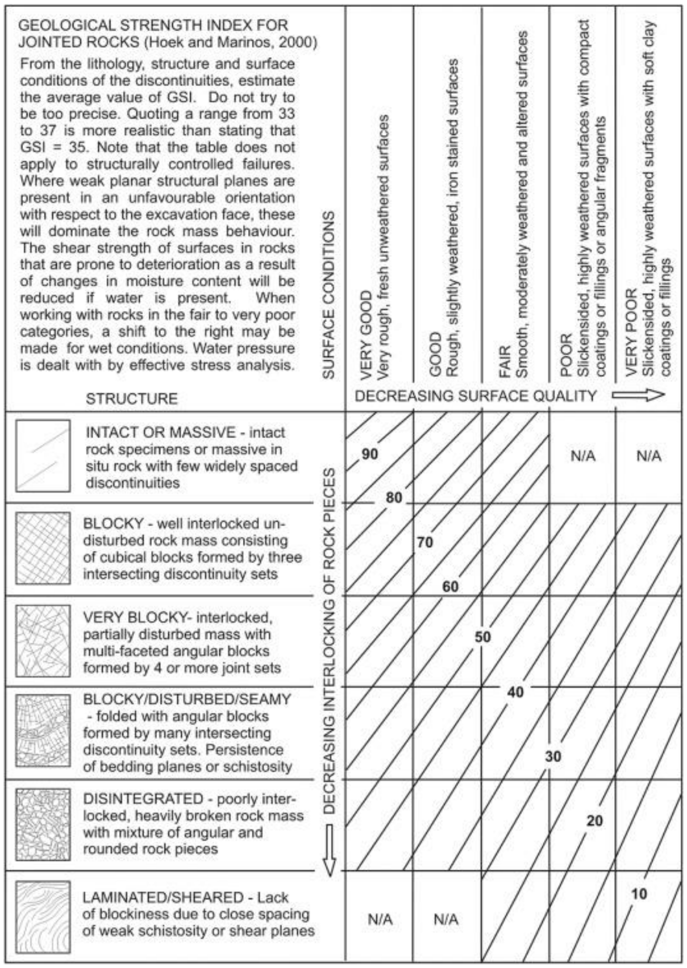

Figure 2.3. The basic GSI chart for visual geologic characterization of rock masses (*P. Mairnos & E. Hoek, 2000*)

Table 2.1. Rock mass rating system (*Z. T. Bieniawski, 1997*)

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS									
	Parameter		Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	2% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensidd surfaces or Gouge 5 mm thick or Separaton 1 -5 mm Continuou	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

Using the drill core data, uniaxial compressive strength values of intact rock, conditions of discontinuities and report of the groundwater research made through the exploration stages, approximate values of RMR and GSI have been calculated regarding rock mass classification schemes.

2.3.1. Tentative estimation of RMR and GSI value for rock mass in the Narynsukhait coal mine

Figure 2.4. Rock sample tray of drill core

Table 2.2. Tentative calculation of RMR value for the rock mass in NS coal mine

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS									
	Parameter		Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	2% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensidd surfaces or Gouge 5 mm thick or Separaton 1 -5 mm Continuou	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

Table 2.3. Rock mass quality classes of the RMR classification system

Rock mass description	Very good	Good	Fair	Poor	Very poor
Ranking	100-81	80-61	60-41	40-21	<20
Class	I	II	III	IV	V

For the Narynsukhait coal mine, according to the drill core data and detailed information in the exploration report, the following parameters were selected. As referred to: the average uniaxial compressive strength is 58.7MPa, the drill core quality (RQD) is around 65%, the spacing of discontinuities ranges from 60 to 200mm, the condition of discontinuities is classified as slickenside surfaces or gouge 5mm thick or separation 1-5mm continuous, and the groundwater condition is partially wet. Thus, the total rock mass rating (RMR) value was approximately 45, and the GSI value can be calculated as 40. Also, the rock mass in the Narynsukhait coal mine can preliminarily be categorized as class III and fair rock.

2.4. Background of the Narynsukhait coal mine

The Narynsukhait coal mine is the second largest coal mine and a research site selected to develop the guideline for extraction of steep longwall coal mining and is located in Umnugobi province, Mongolia. The mining operation commenced in 2008 and is distanced 296km from the capital city of the Umnogobi province, 849km from Ulaanbaatar, the capital city of Mongolia and 60km-70km from the Shivee-Khuren port on the southern border between Mongolia and China.

It is connected to the border by a heavy technological concrete road, and coal is transported by a heavy truck which has high transportation costs as a disadvantage. Some key parameters of NS open-pit coal mine are listed as follows:

- Resource and reserve:
  - Measured + Indicated + Inferred – 576.5Mt
  - Mineable reserve – 498.2Mt
- Method – An open-pit mining method
- Final open-pit depth – up to 250m
- Annual production – 8.0-12.0Mt
- Coal type – Bituminous coal and anthracite
- Coal quality – Semi-soft coking coal and high-quality thermal coal.

2.4.1. Geology and rock properties

The Narynsukhait coal deposit is situated in the Orgikhbulag formation, a major coal-bearing formation divided into two members; Narynsukhait member (J_{2NS}) and MAK member (J_{2MAK}). Moreover, it belongs to the Jurassic in terms of geological aging which ranges from 145 million to

200 million years ago. The Orgikhbulag formation consists of dark to light siltstone, coal, muddy shale and coarse/fine-grained conglomerates, mudstone and sandstone that have predominantly occurred among the surrounding rocks. The geological structure involves two distinct synclines whose coal properties differ slightly. Thus, the morphology and distribution of coal seams in those two synclines were explained separately in an exploration report, as shown in the lower part of Figure 2.6. It is around 12km in length along the strike direction and 1.5-2.1km in width along the dip direction.

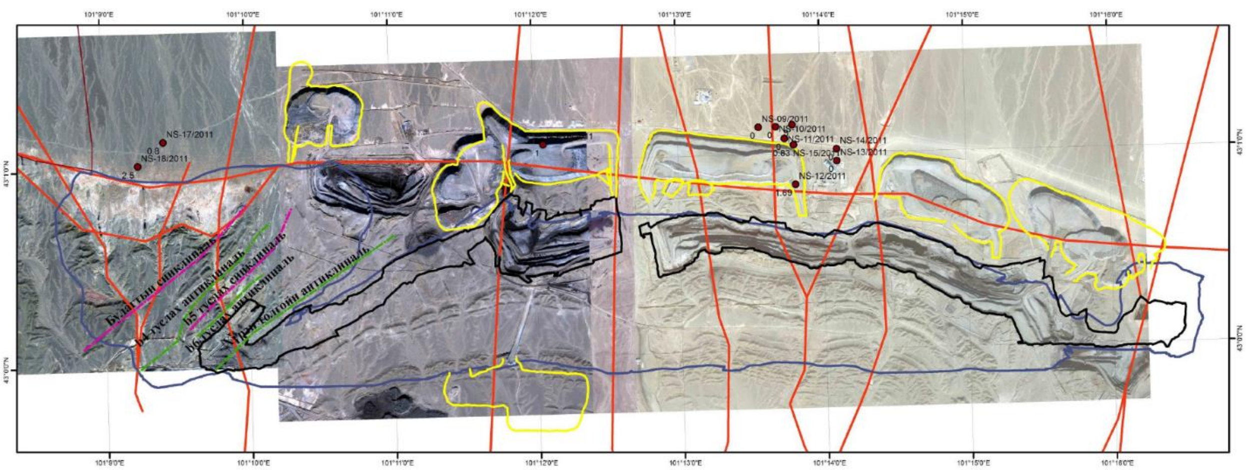

Figure 2.5. The Narynsukhait open-pit mine

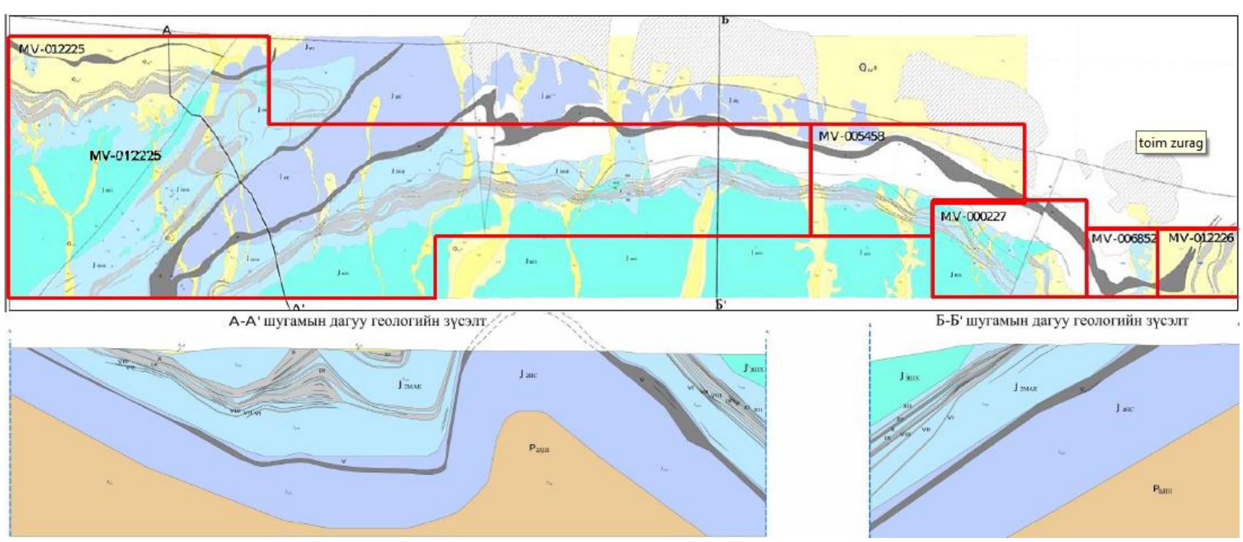

Figure 2.6. Geological map and cross-sections

There are 12 coal seams, seam I-IV are infeasible for economic purposes, seam V-XII are feasible to be extracted economically and technically, and the thickest seam is seam V, whose true thickness varies up to 60m. The view of the open pit, geological map and cross-sections are depicted in Figures 2.5 and 2.7, respectively. Coal types are bituminous and anthracite. High-quality thermal coal, semi-soft coking coal and a small amount of anthracite have been mined and exported to China via the Shivee-Khuren southern port.

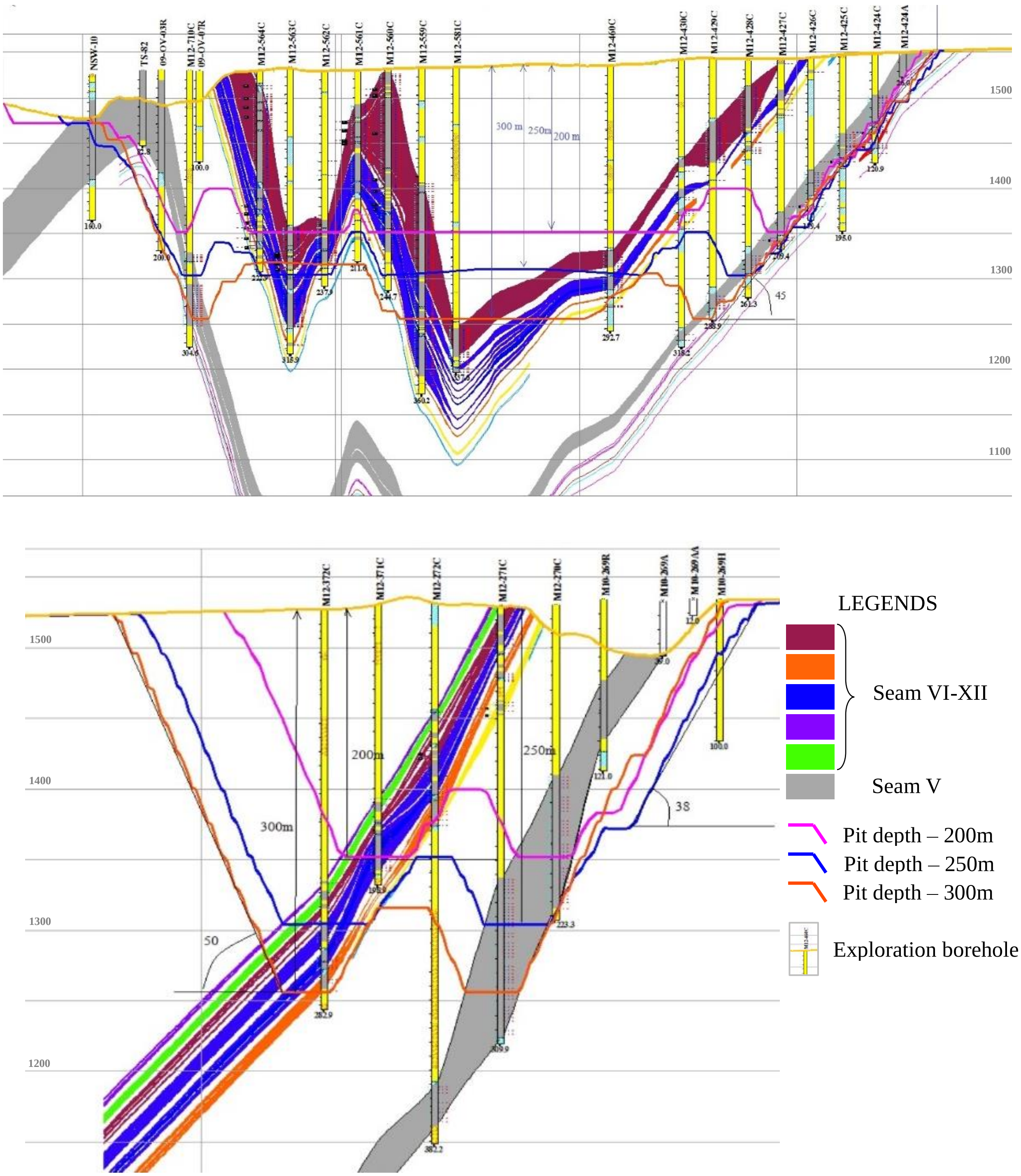

Figure 2.7. Cross sections of the Narynsukhait open-pit mine

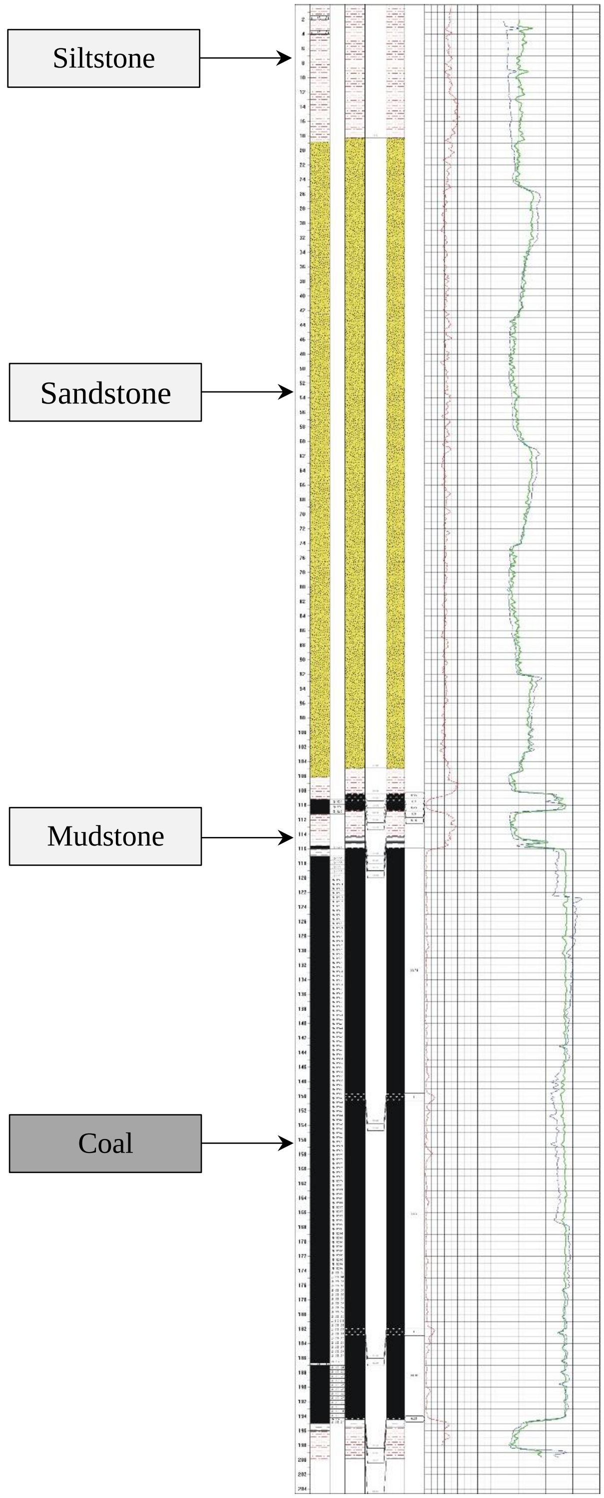

Figure 2.8. Stratigraphic column of drill core

Most faults are aligned with the dip direction, and one major fault evolved along the strike during the tectonic process. *H. Kang et al. (2010)* also summarized that the stress variations caused by complex geological structures might also result in some data scatter, and regional and local tectonic fault structures significantly influence the orientation of horizontal stresses. Measuring and defining the orientation of horizontal stresses and magnitudes are important in mining due to the complexity of geology, rock strata structures and distinct disparity in rock properties in each field. Therefore, an adequate study for stress regimes has to be conducted ahead of commencing underground coal mining projects; to begin this type of research is a priority issue in Mongolia. In order to study the interaction between stress orientation and the stability of the underground openings and supports, further study will significantly consider the different orientations of the maximum horizontal stresses and magnitudes in numerical analyses with various variants. Figure 2.9. shows the relationship between mining depths and the uniaxial compressive strength (UCS) of several rock types in the Narynsukhait coal mine. These results were obtained from laboratory experiments of rock samples collected from the drill core. Rock mass in the Narynsukhait coal mine can be classified as medium strong and strong based on the practical estimates of rock strength by Hoek and Brown (1997).

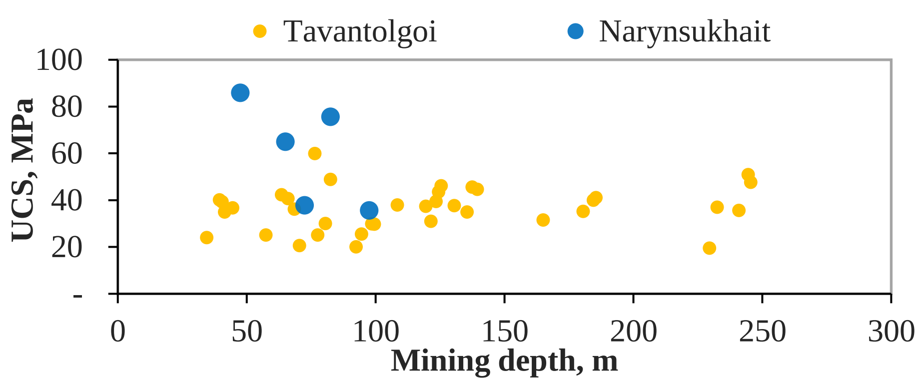

Figure 2.9. Comparison of the uniaxial compressive strength of rocks in Tavantolgoi and Narynsukhait coal mines, Mongolia

2.5. Summary

This chapter discussed geological and geotechnical conditions, including in-situ stress regimes and tentative rock mass classification made based on the drill core samples, the potentiality of steep coal seam mining, and introduced the basic background of the Narynsukhait coal mine as the selected main research spot. It is an ongoing mining project and the second largest open-pit coal mine in Mongolia, producing around 10.0Mt raw and washed coal and all are exported to China every year since the project commenced in 2011. This statistic shows that this mine significantly contributes to the economic development and income of coal export in the state budget. Due to this exclusive reason, steep coal seam mining has to be developed in territories that are near the southern border and infrastructure is well developed. Although any research on in-situ stress investigation and measurement has not been conducted yet in Mongolia, the essential trend of the primitive stress regimes can be determined by comparing several engineering studies that have been carried out to investigate in-situ field stress regimes in underground coal mines at a wide range of depths ranging from hundreds to thousand meters in several provinces in China and Inner Mongolia. Thus, different in-situ conditions and lateral stress ratios corresponding with mining depths have been considered and employed in later numerical analyses. Since rock masses have inherent characteristics of the different field conditions, rock masses in the NS coal mine can be classified accurately with qualitative and quantitative assessment schemes, as RMR and GSI values have been determined at 45 and 40, respectively. Moreover, coal seams in the NS coal mine were classified as steep and extra-thick coal seam according to the criterion for coal deposits recommended by Ministry of Mining and Heavy Industry in terms of geo-morphology.

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CHAPTER THREE. **THE STABILITY OF THE GATEROAD AND THE EFFECTIVENESS OF SUPPORTS**

3.1. Background

As explained in Chapter 1, the longwall mining method is considered the most effective and highly productive economically and technically for extracting coal in the rest of the world. Nevertheless, the difficulties and uncertainties related to ground behavior and applying the longwall mining technology in the steep coal seam mine play main roles in concerns associated with safety and consistent mining operations for the entire mining life as well as the complexity of the geology, especially the pervasively varying coal seam dips and thicknesses. Inevitably, various technical processes and appropriate patterns of the longwall mining design have been considered to afford certain features of distinct coal layers and rock strata. For these reasons, this chapter introduces the numerical model, certain elements of longwall mining, and input parameters, including the mechanical properties of rock and coal. In order to study the stability of longwall mining in steep coal seam, the numerical analysis model has involved different coal seam dips, mining depths, various stress directions and magnitudes and support systems with spacing variations. These numerical analyses were carried out by finite difference method utilizing FLAC^{3D} software, a highly advanced three-dimensional continuum modeling software for mining and civil engineering applications, for solving complex geotechnical problems of soil, rock, concrete, structural ground support, and groundwater flow. Moreover, rock mass deterioration calculated with RSData software has been applied to pay attention to the diversity of this research from the geotechnical and engineering point of view.

3.2. The face directions for the steep coal seam longwall mining

The selection of the longwall face direction is one of the important operational concerns. Compared to the horizontal, nearly horizontal or gently inclined seams, a longwall face in steep coal seams can retreat following directions in actual practices as illustrated in Figure 3.1:

- Along the strike
- Down the seam dip
- Up the seam dip

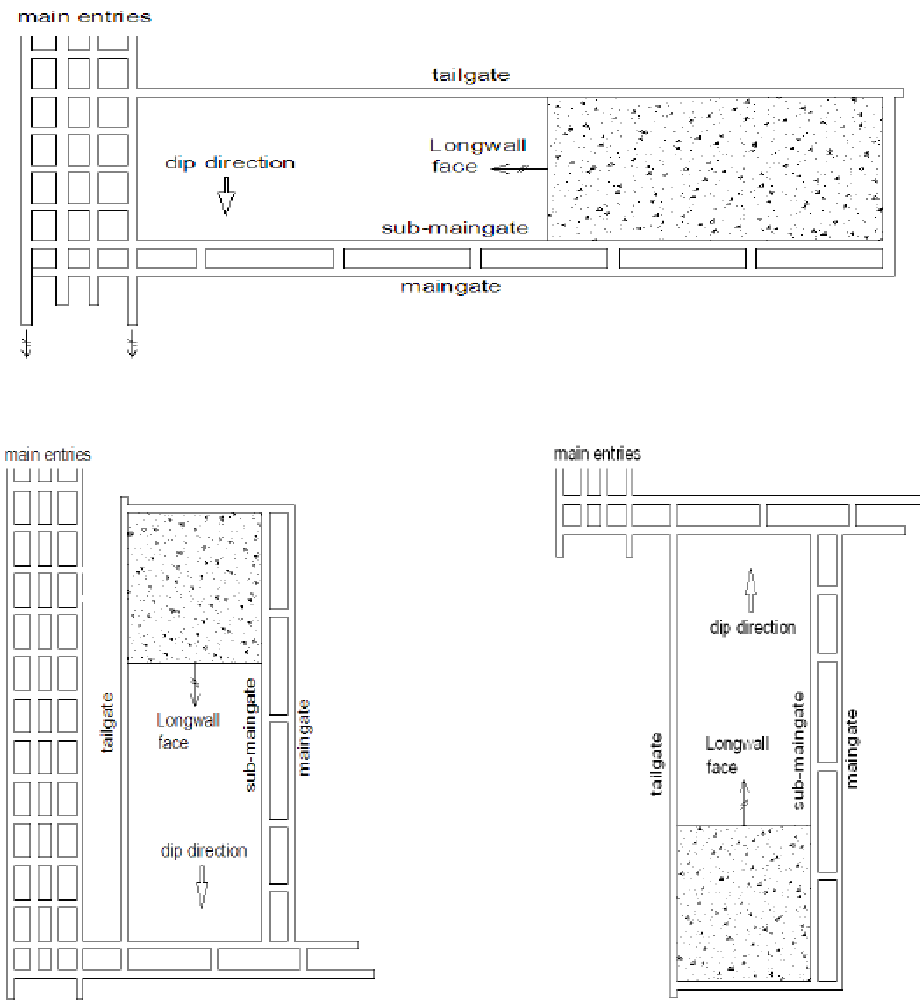


Figure 3.1. The common directions of longwall panel retreat

These three directions have been specifically applied due to the ground control issues, operational efficiency and economical profitability. For example, for extracting inclined coal seams by multi-slice longwall mining method with backfilling in Poland, the face is usually oriented in an up-dip direction to allow better drainage of the hydraulic backfill (*Palarski, 1999*). Here is another example that was practiced in the USSR (Russia). In order to extract the lower section of the inclined thick coal seams by longwall top-coal caving mining method with steel mats (the 2m-2.5m top section is mined by the conventional longwall method, laying steel mats on the floor during operation), the face was oriented in down-dip direction (*Adam, 1976*) probably to attract more coal from the middle layer between the two slices. Different face directions have certain advantages and disadvantages associated with mining operations, stability issues of underground excavation and economic impacts. Regarding the operational point of view, the face retreating along the strike has many advantages compared to the up-dip or down-dip directions. Herewith those are listed below:

- The main gate and tailgate are horizontal and much easier to be driven, and less time consuming to be developed
- The ventilation during the face excavation and development is simple.
- The length of the longwall panel can be developed much longer.
- Driving the horizontal gateroads can allow underground water to drain on collection points at the main gate that can de-water during the face excavation.
- Transportation for miners and techniques is convenient on horizontal gateroads.

Regarding those important advantages and a number of successful practices of face retreating along the strike in the rest of the world, face excavation in steep coal seams along the strike was applied in further numerical analysis.

3.3. Model description

Corresponding to the main objectives of this research, utilizing three-dimensional numerical model and series of analysis which contain various geological and geotechnical conditions are the key approaches for investigating the stability of the gateroads before or after panel retreats. Besides, the effects of the parametric variations of countermeasures that can be installed as support in gateroads and face to improve surrounding rock masses being weakened have studied. The overall view of the numerical model is shown in Figure 3.2. The model is composed of two different rock types; coal and sandstone. Furthermore, this simple model includes a single steep coal seam, overburden and floor rock consisting of only sandstone, and two longwall panels set in a coal seam along the strike and separated by chain pillars with the lower and upper gateroads. The width of the numerical model ranges from 300m-400m along the x-axis, depending on the mining depths and model geometrical adjustment, and the length is fixed at 500m along the y-axis. After 200m long panel retreats, 150m coal seam will have remained on two ends that are assumed to reduce the boundary effect for the analysis as much as possible. All detailed variable and non-variable input parameters associated related to geological and geotechnical conditions and stress regimes applied in numerical simulation are listed which follow:

Non-variable parameters:

- Coal seam thickness – 3m
- Longwall face height – 3m

- Longwall panel length and width – 200m and 100m
- Gateroad width and height – 5m and 3m

Variable parameters:

- Mining depths – 300m, 500m and 700m
- Seam dips - 30˚, 45˚ and 60˚
- Chain pillar width – 30m (10m and 20m investigated in the NS mine initial case)
- Initial lateral stress ratio – 0.5, 1, 1.5 and 2
- Different high horizontal stresses – $\sigma_x>\sigma_y=\sigma_z$; $\sigma_y>\sigma_x=\sigma_z$
- Different intermediate horizontal stresses – $\sigma_x>\sigma_y>\sigma_z$; $\sigma_y>\sigma_x>\sigma_z$

There is no one constitutive law or failure criterion that is mechanistically rigorous or universally applicable. However, there are several failure criterions that can be applied in civil and mining engineering applications. Among those, the elasto-plastic Mohr-Coulomb failure criterion, one of the most common criterions, is pervasively used and more appropriate for underground excavation, particularly for mining engineering field. The top boundary or the ground surface is only set free to move in all directions, while other boundaries are restrained axially. The mechanical properties of rock and coal are listed in Table 3.1, and some parameters were obtained from laboratory experiments conducted on drill core samples from the NS coal mine. In terms of gob modeling and material properties, the gob material properties suggested by *Thin, Pine & Trueman (1993), Xie, Chen & Wang (1999), Yasiti & Unver (2005)* and gob modeling scheme made by using estimating the caving height of roof strata concept suggested by *Yavuz (2004)*, latterly used by several alumni of Kyushu University, were both proposed. For mining steps, rectangular-shaped horizontal two-gateroads are driven first, and then the lower and upper longwall panels retreat sequentially. Moreover, two distinct cases of gateroads and longwall faces are considered to study the effect of support systems; unsupported and supported. For the study of supported cases, active type supports, which are rock bolt and cable bolt, and passive type support, which is steel arch, have been employed to determine the appropriate version. Rock bolt support was applied to maintain the roof and rib sides, cable bolt was applied for the floor heave problem, and steel arch support was used to compare the results of various supports and how these different supports improve the surrounding rock mass.

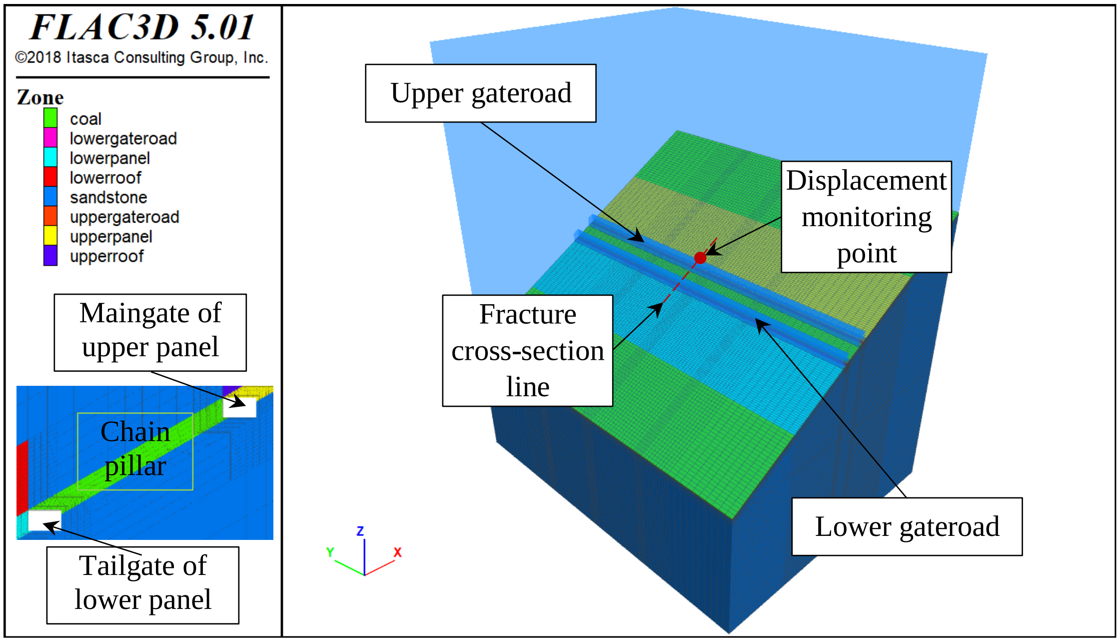


Figure 3.2. General view of analyzing model

Table 3.1. The mechanical properties of rock and coal

№	Parameters	Unit	Sandstone	Coal
1	Bulk modulus	MPa	8,548	4,820
2	Shear modulus	MPa	2,640	2,010
3	Young's modulus	MPa	7,181	-
4	Poisson's ratio	-	0.4	-
5	Tensile strength	MPa	7.0	0.7
6	Friction angle	deg	36	43.1
7	Cohesion	MPa	1.6	1.4
8	Density	kg/m^3	2,498	1,380

The interaction between the orientation of high horizontal stress and gateroad, cut-throughs and longwall face is a very important issue from a geotechnical engineering point of view, even though it can allow diminishing concerns for the safety and economic factors with respect to mining operations. The mechanical properties of rock and coal are mainly obtained from laboratory experiments through geological exploration work and mining operations. Nevertheless, those rock samples are taken from intact rock mass, which is relatively solid, has fewer discontinuities, and is not weathered and weakened. Due to this circumstance, the rock mass seems to have more strength than rock mass in the actual condition that involves a lot of discontinuities, weathering, faults, folds and crevices. Thus, rock mass deterioration using GSI value reduction that utilized RSData software version 1.007 released by Rocscience in 2022 has been trialed in numerical simulations in further sections. The new and weak mechanical properties of sandstone have been created by reducing the GSI value by 25% and 50%.

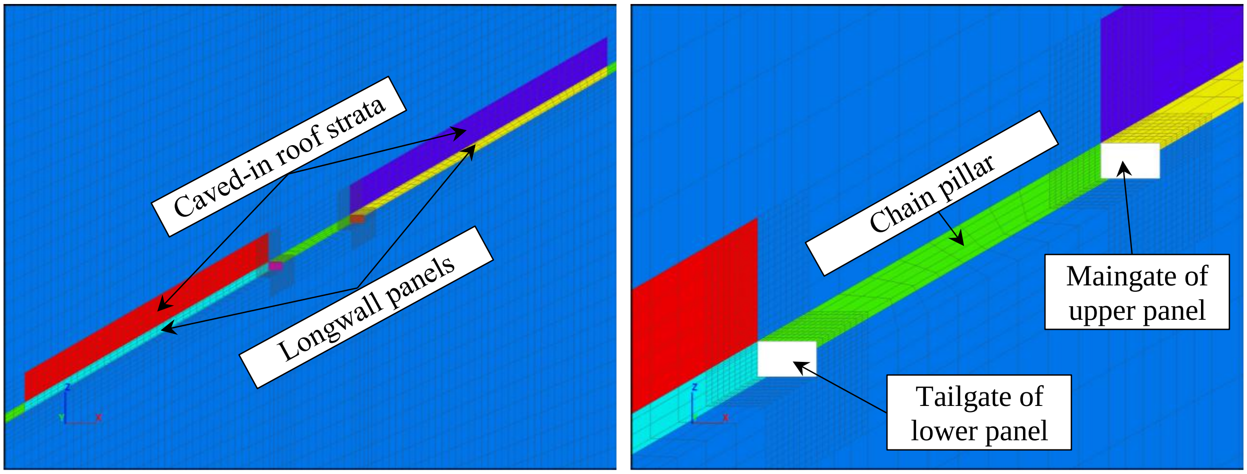


Figure 3.3. Front view of analyzing model

As seen in Figure 3.3, there are two rectangular-shaped gateroads; one is a tailgate of the lower panel, and another is the maingate of the upper panel. Also, the chain pillar, steep coal seam and caving roof rock that caved-in height were calculated in advance are presented. The determination of stable or unstable conditions will be made based on parametric analysis and numerical results, such as displacement, fracture zone distribution and dimension. Besides that, the safety factor can be used to specify the influence of dip angles on the stability variations of underground excavations, including the gateroad, longwall face and ground surface related to subsidence issues. The displacement is one of the priority parameters resulting from numerical analysis representing ground conditions and increment of instability and failure state or fracture zone to be considered in addition.

3.4. The effect of coal seam dip

Considering the various coal seam dips, and several geological features, three coal seam dips ranging from 30˚ to 60˚ were selected to be studied, excluding horizontal seams. Basically, there are no horizontal or nearly horizontal coal seams, and the average dip angle is around 43˚ in the NS coal mine. To investigate the effect of seam dips on the displacement, the measurement points are fixed on the center of the roof every 50m along the upper gateroad, which is named displacement along the gateroad. The vertical displacement values of the roof with three different seam dips range from 3.2mm to 5.44mm at maximum when other initial conditions of analysis are set constantly as the lateral stress ratio is 1 and mining depth is 300m. The highest displacement values shown in Figure 3.4 have been collected from the brink of the roof along the gateroad. This figure also shows that the highest displacement occurs near the beginning of the gateroad advance, and a very slight displacement variation occurs with the seam dip increasing.

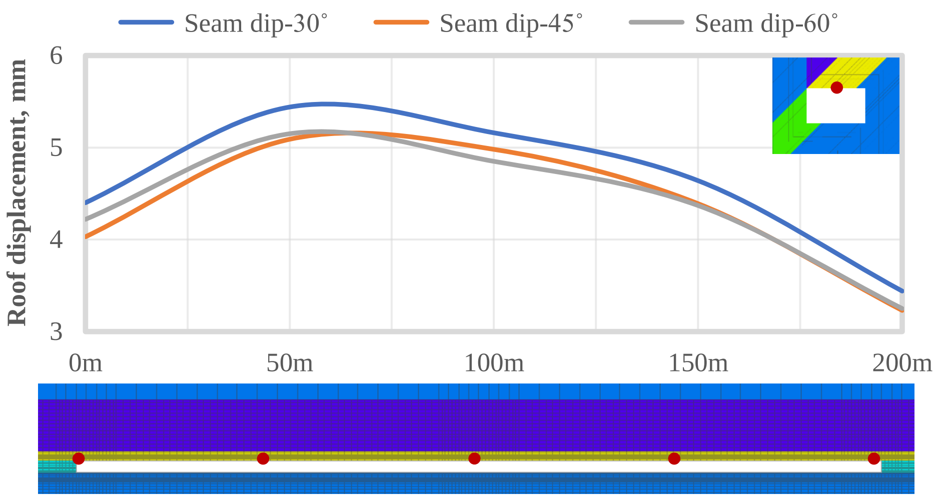


Figure 3.4. The relationship between roof displacement and coal seam dips along the gateroad length

For the fracture zone development around the gateroad, it is seen that once the longwall face mining direction is along the strike, horizontal gate entries are less affected by various seam dips. Small changes in fracture zone propagation have been observed near the corners and rib sides. The failure zone size around the gateroad in the result of the seam dip-30˚ is relatively larger than the results of seam dip-45˚ and seam dip-60˚. One reason for the larger shear failure zone occurring in the low seam dip model is that shear failure is easily propagated along the boundary between the coal and rock on the upper left corner of the gateroad which is near to the roadway surface. Another reason for the larger shear failure that occurred on the right rib of the gateroad is related to the mechanical properties of sandstone. The shear failure, particularly shear-p, dominantly prevailed. Shear failure is common failure mechanism in underground coal mining roadways and is usually induced by high horizontal stress.

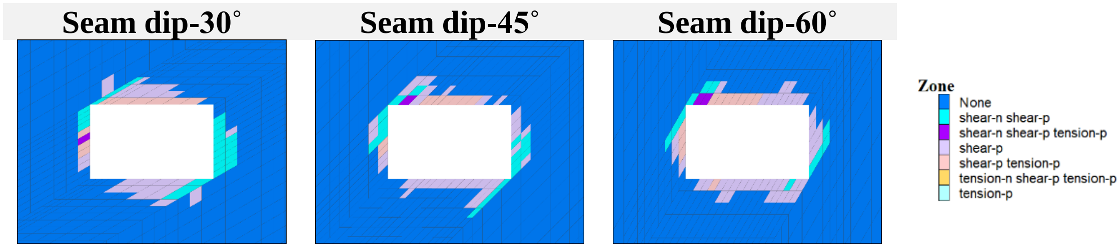


Figure 3.5. The relationship between fracture zone and coal seam dips

If shear failure initiates at the roadway corners and extends deep into the roof, roof falls caused by shear failure in large scale can usually be occurred. But, sometimes, this can be quite difficult to predict the size and length of shear failure propagation deep into the roof rock, which is not visible in actual observations. In FLAC^{3D} simulation, two types of failure mechanisms are indicated by plasticity state plot: shear and tensile failure. The plot indicates whether stresses within a zone are currently on the yield surface (the zone is at active failure now, **-n**) or the zone has failed earlier in the model run, but now the stresses fall below the yield surface (the zone has failed in the past, **-p**). Thus, shear-p indicates that shear failure occurred in the gateroad have failed in the past or earlier in the model run. It can be summarized that the larger dip angle of the coal seam does not result in higher displacement and fracture zone development around the gateroad before the panel retreats. It even does not lead to serious instability and can be assumed stable.

3.5. The effect of mining depth

Compared to the variations of displacement for different seam dips, it is obvious that the mining depth increases from 300m up to 700m with vertical displacement of the roof and floor dramatically increasing, correspondingly. The lateral stress ratio is 1, and the seam dip is 45°, as shown in Figure 3.6. Regarding the specific features of longwall coal mining, floor heave is another serious concern that usually occurs if the mine deals with soft rock strata on the floor and in case of high horizontal stress regime prevails. Therefore, the floor displacement should not remain out of engineering consideration.

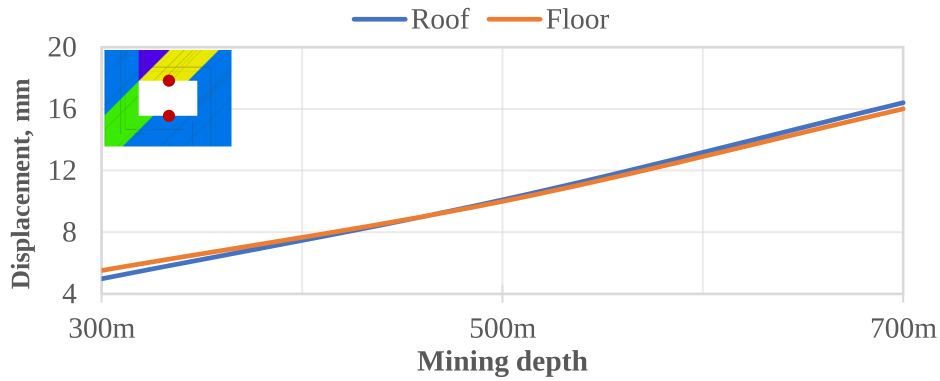


Figure 3.6. Vertical displacements at different mining depths

When mining depths are 300m, 500m and 700m, the roof and floor displacement values are 4.98mm, 10.1mm, 16.4mm and 5.52mm, 10.0mm, and 16.0mm, respectively. Although the relationship between the displacement and mining depth is very clear, there is no significant difference between roof and floor displacement. Figure 3.7 shows that the greater depth causes the larger fracture propagation around the gateroad. It is realized that all fracture zone forms the arch pattern, and the failure thickness broadens mainly on the roof and floor strata. These phenomena can lead the roof rock collapse and considerable floor heave. The rib sides have worsened, and the fracture on the ribs is enlarged. However, the fracture zone occurred smaller on the roadway's lower left and upper right corners than on other corners. Due to noticed instability and rock falls of the roof and rib, combining different types of support or augmenting primary support needs to be carefully considered at greater depths, especially when mining is being transited from shallow to deep underground. In the numerous underground coal mining practices which have competent and strong rock mass, rock bolt or a combination of rock bolt and cable bolt supports have widely been proposed in the present and past considering some economic aspects. As mentioned in many practices and experiences, it can be recommended that the thickness of shear failure propagated into the rock mass should be shorter than 2/3 of the rock bolt length. If the shear fracture extends above the length of the rock bolt anchorage, a massive roof fall may suddenly occur and damage equipment and personnel. However, a similar circumstance has appeared as the significant influence of different seam dips was very small in the analysis of deep underground.

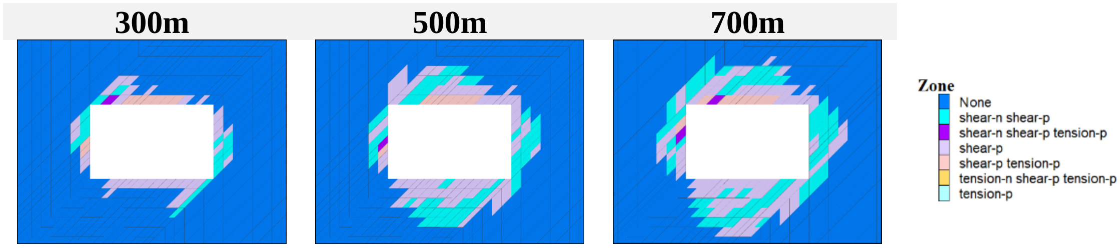


Figure 3.7. The relationship between the fracture zone and mining depth

3.6. The effect of lateral stress ratio

As explained in Chapter II, unfortunately, no in-situ stress measurement and actual field investigation research to determine primitive stress regime have not been conducted in the NS coal mine, even thoughout Mongolian territory. However, the general trend of stress regimes proposed by numerous researchers is quite similar in the rest of the world. Among those remarkable studies, *H. Kang et al. (2010)* introduced and conducted numerous field measurements for in-situ stress regimes throughout China and Inner Mongolia. The result was relatively similar and comparable to the trend introduced by Brown and Hoek. According to this situation, the different lateral stress ratios have been applied in the analysis, and their significant impacts on the stability of gateroad and longwall face excavation will be studied to delineate in detail. In three-dimensional modeling analysis, lateral stress ratios 0.5, 1, 1.5 and 2 have been set. When the lateral stress ratio varies, horizontal stresses in the two axes of x and y change in the same magnitude concurrently. It is noted that intermediate horizontal stress was not considered in this simulation. In order to study the effect of the various stress conditions on the gateroad stability, the following schemes are applied in numerical analysis.

- k=0.5 $\sigma_x = \sigma_y < 2\sigma_z$
- k=1.0 $\sigma_x = \sigma_y = \sigma_z$
- k=1.5 $1.5\sigma_x = 1.5\sigma_y > \sigma_z$
- k=2.0 $2\sigma_x = 2\sigma_y > \sigma_z$

The numerical results show some typical ground response principles with respect to applied different stress conditions. A larger stress ratio or high horizontal stresses apparently worsens roof and floor rock conditions. For example, when the lateral stress ratio increases, the vertical displacement in the gateroad also increases. Nevertheless, depending on the seam dip, the low inclination of the coal seam can cause increasing roof displacement when the lateral stress ratio becomes higher, as shown in Figure 3.9. The vertical displacement value of seam dip-45˚ and seam dip-60˚ were 7.83mm and 8.31mm, respectively, while that of the seam dip-60˚ was 11.2mm, increased by around 35%. Another circumstance drawn attention was that when the lateral stress ratio increases, the direction of the displacement vectors rotates anti-clockwise, which was observed in the high displacement area that has been moved from the center line of the gateroad to the sides, depicted in Figure 3.8.

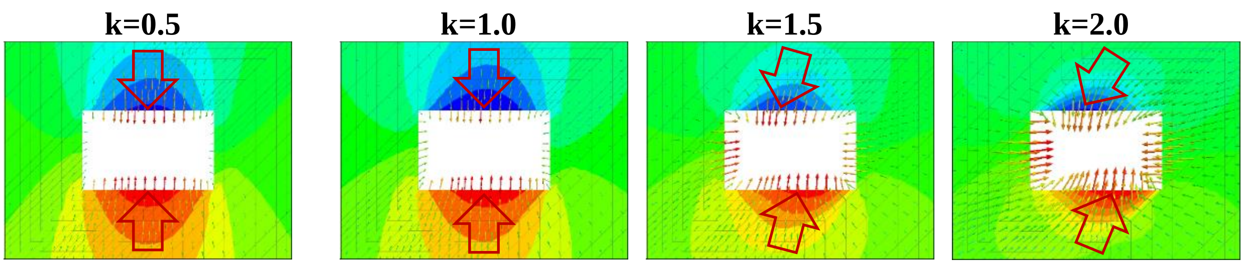


Figure 3.8. The distribution of displacement vectors

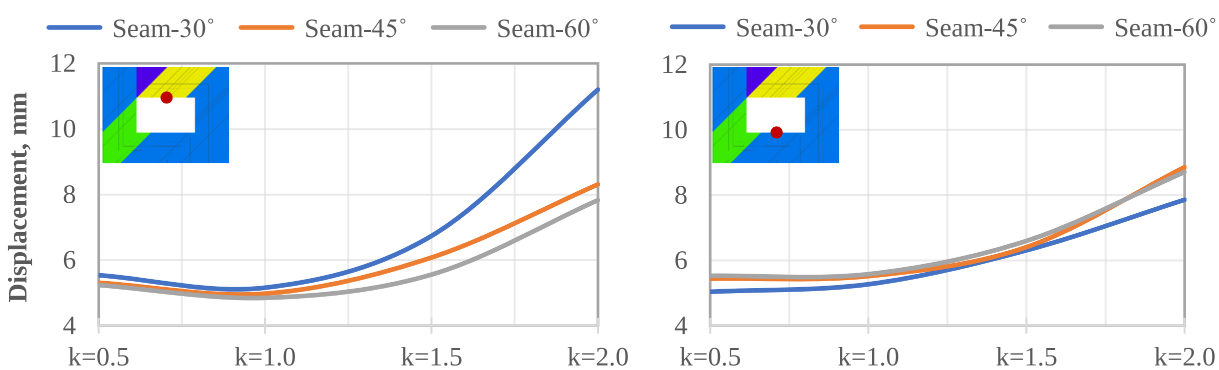


Figure 3.9. The relationship between the roof/floor displacement and k ratio

It can be realized that regardless of the higher lateral stress ratio, displacement in steep coal seams has not drastically increased, and the ground control problems can be relieved significantly compared to gently dipping or horizontal coal seams. Moreover, the diagonalized opposite corners may be assumed differently for the design of the support system. A small difference in floor displacement was observed under various stress conditions.

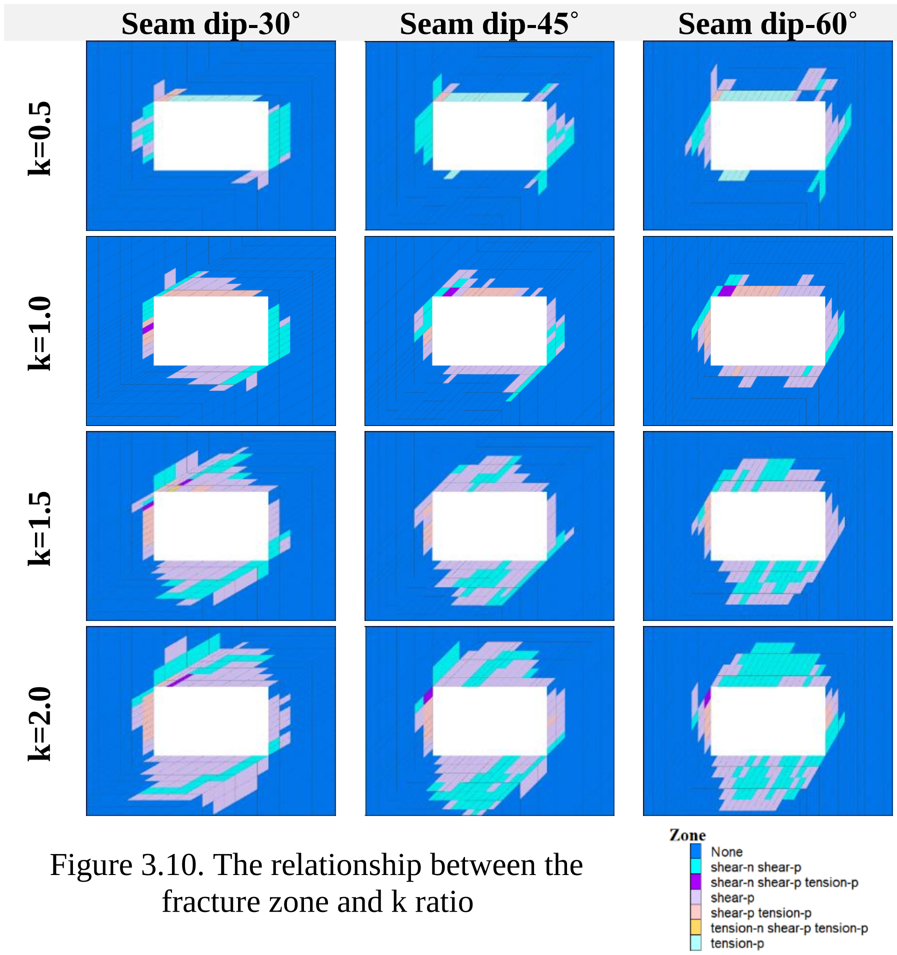


Figure 3.10. The relationship between the fracture zone and k ratio

When the lateral stress ratios are 0.5 to 1, the failure zone on the roof and floor is relatively smaller than the failure zone on the rib sides. Once the lateral stress ratio increases to 1.5 and 2.0, the failure zone size on the roof and floor is apparently enlarged and more severe than that of the rib sides. It notes that a higher stress ratio or high horizontal stress condition deteriorates the surrounding rock mass, particularly for the roof and floor. The correlation between displacement and fracture zone development around the gateroad was unclear. However, it can be summarized that a higher lateral stress ratio results in a substantial increase in fracture zone propagation around the gateroad, especially on the roof and floor. Otherwise, a drastic increase in displacement of roof and a gradual increase in displacement of floor have occurred.

3.7. The effect of the rock mass deterioration by using GSI value reduction

As mentioned in the model description and initial conditions, the mechanical properties of rock (sandstone) have deteriorated by reducing the tentative Geological strength index value the author calculated based on the drill core data and field observation. The GSI value of the NS coal mine was 40, and it will be reduced by 25% and 50% using RSData software version 1.007. In order to create deteriorated mechanical properties of rock, the generalized Hoek-Brown criterion was employed, and after inserting the initial parameters into the software, renewed parameters were produced automatically correlating with major/minor and shear/normal stress conditions. The deteriorated mechanical properties of sandstone are listed with the initial variant in Table 3.2.

Table 3.2. The deteriorated rock mass properties

№	Parameters	unit	Sandstone		
			GSI-40	GSI-30 Reduced by 25%	GSI-20 Reduced by 50%
1	Bulk modulus	MPa	8,548	1,414	960
2	Shear modulus	MPa	2,640	436	296
3	Young’s modulus	MPa	7,181	1,188	806
4	Poisson’s ratio	-	0.4	0.36	0.36
5	Tensile strength	MPa	7.0	0.014	0.006
6	Friction angle	deg	36	37.04	31.57
7	Cohesion	MPa	1.6	0.85	0.63
8	Density	kg/m^3	2,498	-	-

The vertical displacement of the roof ranges from 15.2mm to 29.0mm when the rock mass deterioration rate is 25%. If the deterioration rate increases to 50%, the displacement values grow from 24.7mm to 69.2mm with the stress ratio increasing, as depicted in Figure 3.11. The difference in displacement between the deterioration rate of 50% and 25% is drastic once the stress ratio rises to 2.0. In this case, the vertical displacement value has increased by 2.4 times. On the other hand, the difference between the deterioration rate of 25% and non-deteriorated rock mass is increased steadily in a similar quantity until the stress ratio reaches 2.0. There is another problem If the GSI value is reduced by 50% and the stress ratio reaches 2.0, serious floor heave inevitably occurs in the roadway. Because floor displacement is increased dramatically to 168.1mm in the numerical result. These circumstances show that weak rock mass properties cause dramatic changes in roof displacement and significantly influence the ground conditions. Besides that, this sensitive correlation cannot be ignored, and the reliability of the rock properties data, rock sampling methods, the processes of laboratory experiments, and the standards to follow compulsory for obtaining initial parameters must be carefully carried out. Although the whole rock mass in the numerical model was deteriorated by reducing the GSI value, the area of weak or highly jointed/faulted rock mass is also significant. Unfortunately, it is almost impossible to create a specific area of weak or highly jointed rock mass in the numerical model due to insufficient field data. According to the general trend between the mining depth and rock mass strength, rock strength can be improved, and rock-mass structure is so tight at greater depth that the rock-mass behavior approaches that of intact rock. In this case, GSI is no longer meaningful to be applied. If weak or highly jointed/faulted rock mass is encountered during the roadway advance, the augmenting number of support systems and length or decreasing support spacing can be considered as well as passive-type support such as steel arch, shotcrete and concrete lines. There is no ultimate correlation between the GSI evaluation and the selection of support systems. In such cases it is inappropriate to use active-type supports such as rock bolts and cable bolts because of the unreliability of the anchorage process in weak rock regarding smaller GSI values.

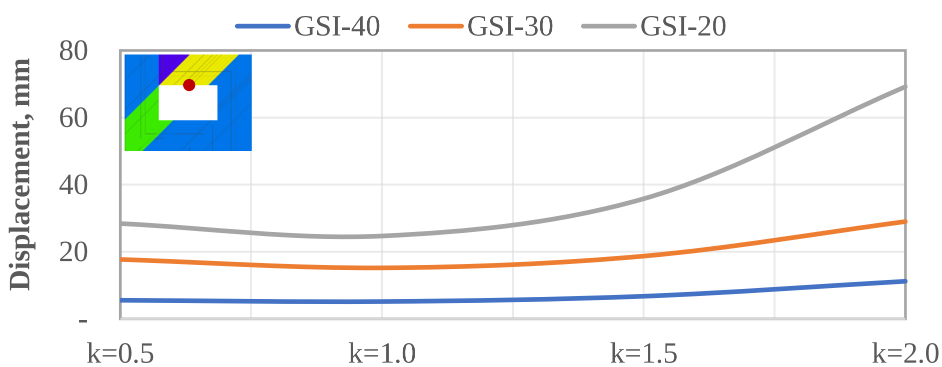


Figure 3.11. Roof displacements with different deteriorated rocks

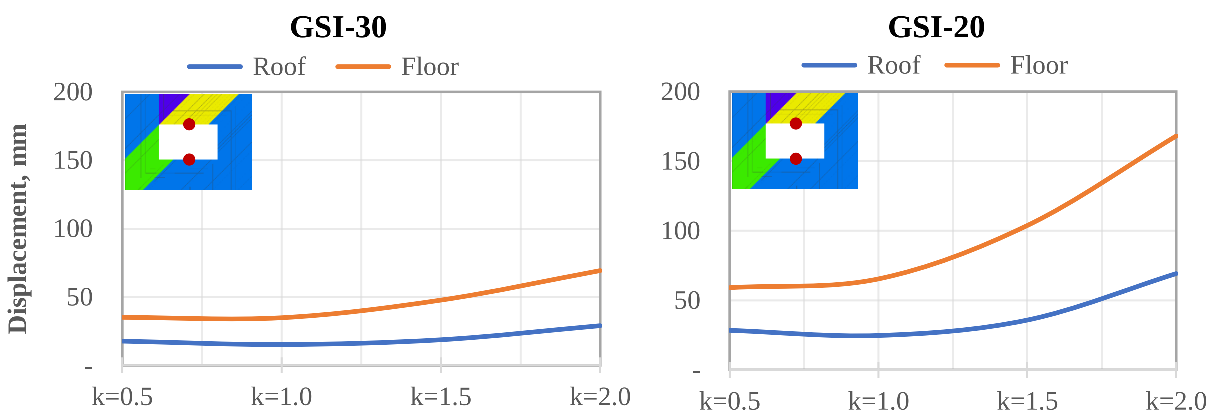


Figure 3.12. The relationship between the roof/floor displacement and deteriorated rock

Furthermore, it requires a well-experienced workforce. Generally, some researchers assume that due to strong rock properties obtained from laboratory experiments conducted on an intact rock sample, rock mass deterioration has to be considered a mandatory task, especially in numerical analysis, because intact rock mass properties cannot represent the heterogeneous rock masses and geological complexity in the actual field. For the fracture zone propagation, the high-stress ratio resulted in a larger fracture zone around the gateroad. Although the stress ratio is small, the fracture has propagated deeply into the roof along the coal seam in a case deterioration rate is 50%. However, mesh geometry could be why rock fracturing was broadened longer. Also, a stress ratio of 2.0 enlarged the greater fracture zone into the floor rock and coal as, similar to the stress ratio of 0.5. It can be summarized that monitoring and contiguous investigation for the weak rock mass properties and their locations that might be dealt with underground openings can be a very important part of predicting real-field situations for geotechnical engineering research in actual mining operations.

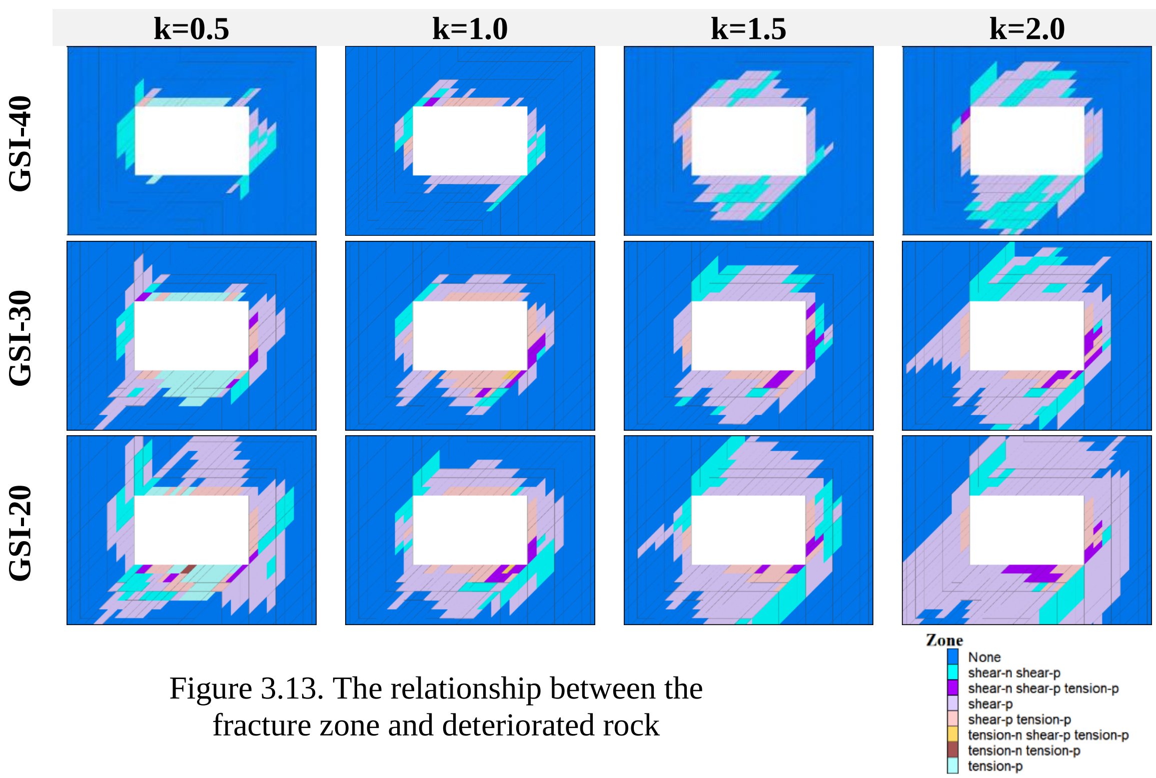
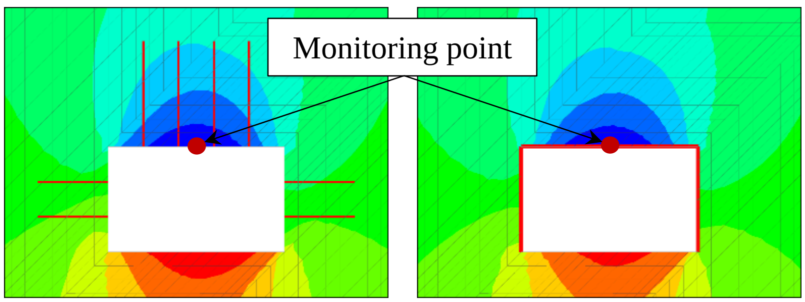


Figure 3.13. The relationship between the fracture zone and deteriorated rock

3.8. The effectiveness of support systems

There are many support systems and designs recommended by researchers for underground mining. In this study, two types of support systems were utilized to improve the stability of the gateroads, especially on the roof and rib sides. One is active type support, rock bolt with grouting, and the other is passive type support, steel arch. These supports will be installed in the gateroad with certain patterns and spacing. Each type of supports has own capacity, efficiency and economic aspect. For example, a friction rock bolt is beneficial to enhance the rock mass strength by tightening the loose rock blocks and confinement near the excavation surface. Moreover, it helps to prevent the rock fall that can cause the unsafe working conditions as well as damage to the equipment and miners. In contrast, steel arch support does not interact with rock mass as rock bolt does. Basically, this type of support can only resist the load acting on the inward movement of the rock. Another advantage of a steel arch is that it retains interlocking of rock mass and allows it to block the pathway of underground water flow that can ingress into the opening through the joints and discontinuities. The properties of friction rock bolt and steel arch support are introduced in Table 3.3 and 3.4. Design of rock bolt and steel are illustrated in Figure 3.14 below.



Rock bolt support Steel arch support

Figure 3.14. View of supports and measurement point in the gateroad

Table 3.3. Rock bolt properties

№	Parameters	Unit	Value
1	Length (Roof)	m	3
	Length (Rib)	m	2
2	Diameter	mm	46
3	Tensile strength	kN	178
4	Young’s modulus	GPa	200
5	Poisson’s ratio	-	0.25
6	Yield strength	MPa	588

Table 3.4. Steel arch properties

№	Parameters	Unit	Value
1	Dimension	mm	95×115
2	Cross-section area	cm^2	36.51
4	Young’s modulus	GPa	200
5	Poisson’s ratio	-	0.3
6	Yield strength	MPa	551

The model included a coal seam dip of 45˚, mining depth of 300m and stress ratio of 1. Friction rock bolts with 1m row spacing and distance between the rock bolts is 1m set in analysis. The rock bolt length into the roof and rib sides is calculated as practically equal to 2/3 of the fracture zone thickness. For example, if fracture zone thickness is 1m, then rock bolt length should be 1.5m or longer. Steel arch is installed with a spacing of 1m. In the many practices of underground coal mining in the world, rock bolt with or without grouting has been widely used as a primary support, cable bolt and shotcrete are secondary supports in roadway depending on certain engineering requirements. For example, rock bolt and cable bolt supports are not effective due to the inadequate anchorage process in the weak rock in southeast Asian countries (*Sasaoka et al., 2014*). Thus, steel arches or high-efficiency standing type of supports need to be installed to provide better safety and proper maintenance for the gateroad stability. Furthermore, three cases of gateroad support are considered in this study as follow:

- Unsupported
- Supported with rock bolt (space of rows and bolts are 1m)
- Supported with steel arch (space of steel arches is 1m)

Cost is another main concern for the selection of support systems in underground mining. *Tri Karian (2016)* introduced some market prices of several supports in the study. For example, rock bolt is 15.8 USD/unit, cable bolt is around 9.8 USD/unit, H-beam is 1000 USD/unit, and shotcrete is 1570 USD/m^3. Due to high cost and price, many mines have been preferring to use tendon supports, although steel arch support can provide high capacity and reliability. The following parametric and qualitative analysis results are introduced to understand the efficiency of support systems and how different types of support improve the surrounding rock mass with different stress conditions shown in Figure 3.15.

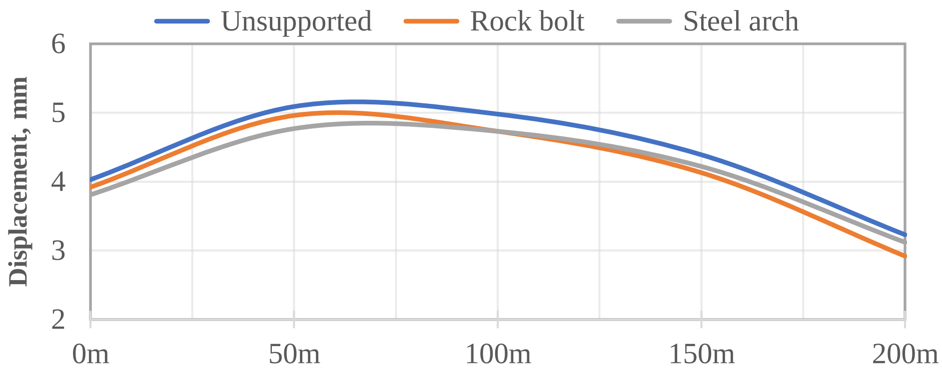


Figure 3.15. The relationship between the roof displacement and unsupported/supported phases along the gateroad

The difference in roof displacement between the cases of unsupported and supported with rock bolts and steel arch is not clear either. The figure above shows that the first 75m of gateroad advance has slightly improved with a supporting steel arch. In contrast, the rock bolt was more effective after 75m of gateroad until the end of the 200m long roadway. Probably single-type support is not sufficient to enhance the gateroad stability. However, the gateroad is not the main roadway and is unnecessary for long-term serviceability through mining life. It is only once used for specific purposes such as transportation for coal, materials, miners, water drainage and ventilation until the adjacent panels are entirely extracted. Besides, the variations of the fracture zone propagation around gateroad with/without supports is shown in Figure 3.16.

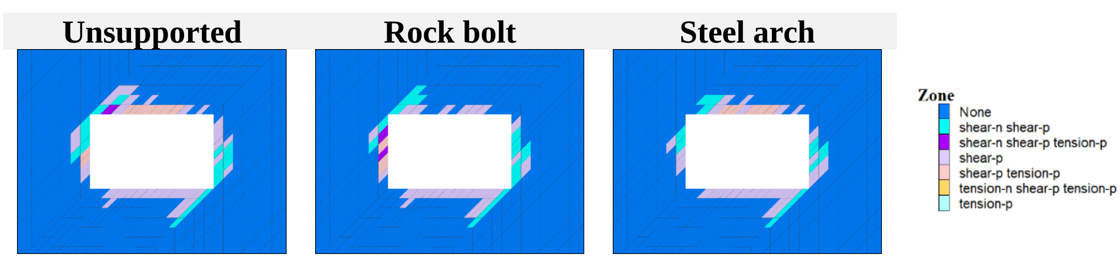


Figure 3.16. Interaction of support systems on the fracture zone

Shear failures usually prevailed in the surrounding rock with different supports. It is observed that there is no serious or large tension failure occurred in numerical results. Moreover, rock bolt support mainly improves the roof and rib side nearby installed and does not affect the fracture propagation on the floor. Whereas steel arch is slightly more effective than the rock bolt for reducing the fracture zone on the roof and rib as well as corners.

3.8.1. The effect of lateral stress ratios on the support effectiveness

Herewith, how the different supports influence the stability of surrounding rock mass behavior when the lateral stress ratio ranges from 0.5 to 2.0. Aforementioned above, the difference between the unsupported and supported cases was very small and it was complicated to clearly define the efficiency for the supports. Probably, those results do not show the vivid variation of results because of strong rock and mechanical properties. It could be supposed to displace in very small quantities. Vice versa, the bigger fracture zone could be formed for the strong or fair rock masses.

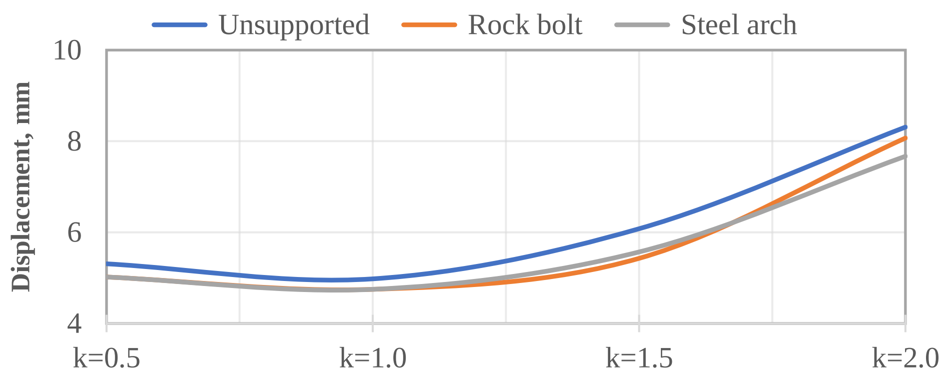
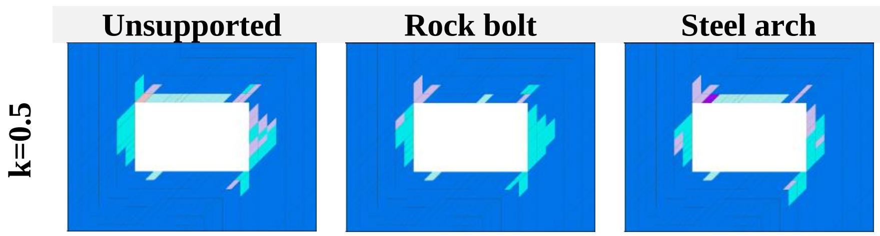


Figure 3.17. The relationship between the roof displacement and unsupported/supported phases

The highest variation of displacement value is occurred when stress ratio reached 2.0 and is decreased by only 8% as depicted in Figure 3.17. The rock bolt and steel arch slightly reduced the fracture zone around the gateroad when stress ratios are varied. The high horizontal stress is still the dominant key aspect resulting in the larger fracture zone propagation and higher displacement, even though two different types of support are installed in the roadway. Moreover, the study of an appropriate support design for the floor heave is needed to be carried out indispensably.



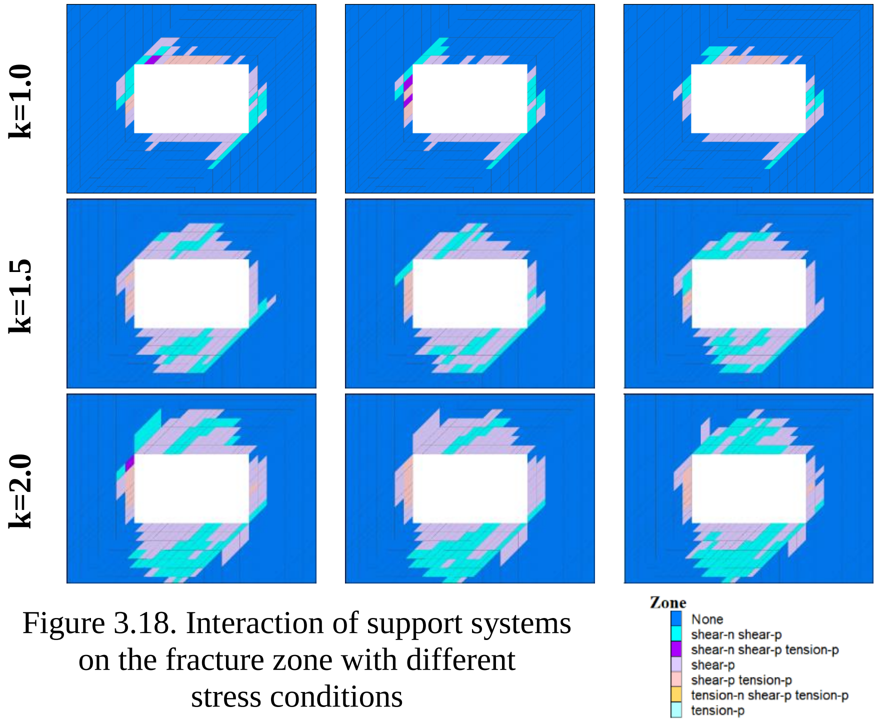


Figure 3.18. Interaction of support systems on the fracture zone with different stress conditions

Some parameters, for example, the beam axial stress of the steel arch and the z-component force of the rock bolt, have been illustrated in Figures 3.19 and 3.20. The locations of the higher values are circled by a red dashed line on the figures. Here, the different loading distribution in steep coal seams has been noticed compared to horizontal coal seam mining. The maximum load acted on the rib side and near corners.

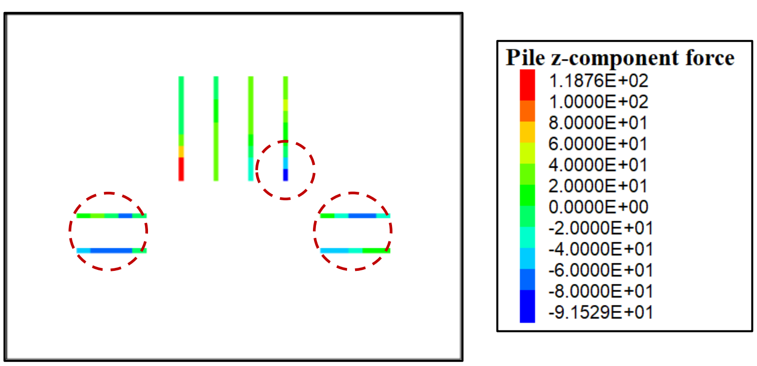


Figure 3.19. The z-component force of rock bolt support

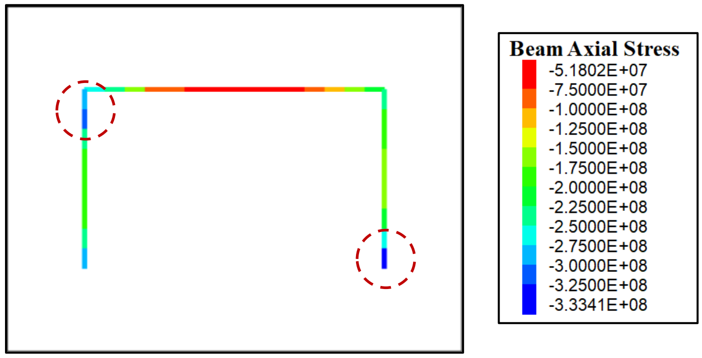


Figure 3.20. The beam axial stress of steel arch support

3.8.2. The effect of rock mass deterioration on the support effectiveness

The roof displacement of the gateroad supported by steel arches has decreased significantly compared to unsupported and supported by rock bolts. The highest roof displacements of rock deteriorated by 25% and 50%, and without support are 15.2mm and 24.7mm. After the roadway is supported by rock bolts and steel arches, the roof displacements were decreased to 13.3mm and 11.3mm in case of 25% deterioration as well as 20.9mm and 14.8mm in case of 50% deterioration. The significant change in displacement decreased by around 42% when the steel arch was applied. It can be said that the rock deterioration process can show the variation of numerical results more vividly and clearly.

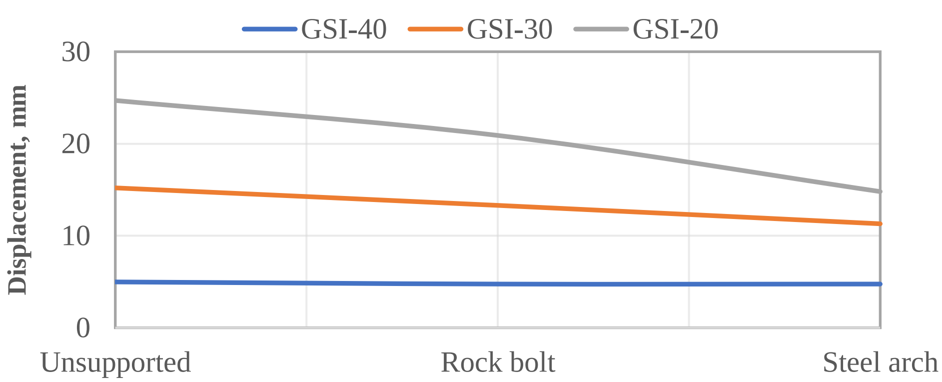
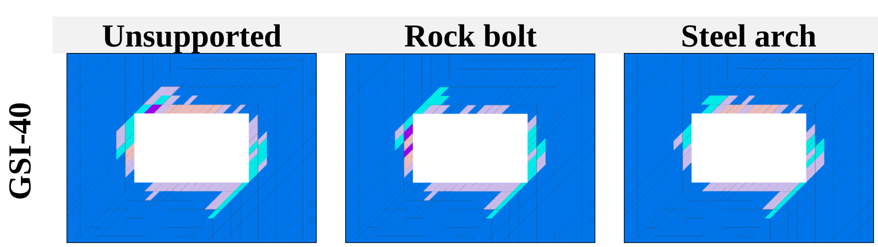


Figure 3.21. The relationship between the roof displacement and unsupported/supported phases along the gateroad



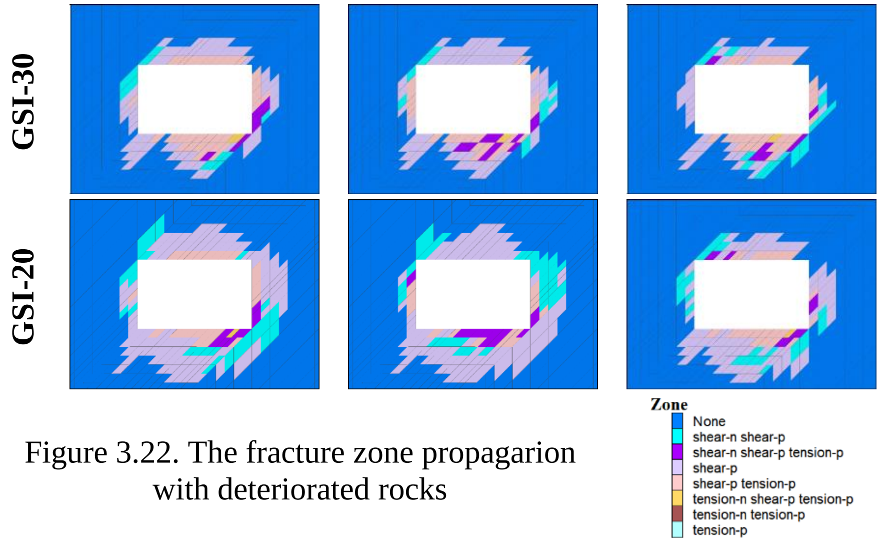


Figure 3.22. The fracture zone propagarion with deteriorated rocks

In terms of the fracture zone around the roadway, the only observable reduction of fracture was noticed when steel arch support was installed. The effect of rock bolt that influence the fracture is a small and shear failure prevails in the simulation results.

3.9. Summary

Ahead of starting the longwall panel excavation for steep coal seam, the specific features for the longwall mining design and layout of the roadway, gateroad, longwall face/panel and pillars have to be designated completely. In this research, three common practices of longwall face directions are introduced with respect to the ground characteristics and technological capabilities. However, considering the noticeable advantages and fewer technical difficulties, the longwall face direction of "along the strike" can be preferred compared to the other two options. But, in this case, the potential difficulties associated with face excavation and its support stability can be raised. The main characteristics of this issue will be discussed in the next chapter. The effect of seam dip on the stability of gateroad especially changes in displacement and fracture zone propagation, is negligible, while the relationship between the mining depth and ground behavior of surrounding rock mass in gateroad is strong. When the mining depth increases, the drastic increase and enlargement of displacement parameters and fracture zone have occurred apparently. Regarding the general trend of correlation between in-situ stress conditions and mining depths, lateral stress ratios of 0.5, 1.0, 1.5 and 2.0 have been proposed to investigate the influences. Based on analyzing results, it can summarize that the higher stress ratio results in larger deformation and fracture zone development significantly. On the one hand, the adverse impacts of a higher lateral stress ratio, for example, a stress ratio of 2.0, is much greater than the effects of seam dips and mining depths from shallow to moderate. Moreover, in order to reduce the error difference between empiric/numerical study and actual mining situations, the rock mass deterioration approach by reducing the Geological strength index value was utilized. Because initial data of rock properties obtained from laboratory experiments made on an intact rock sample is doubtful to represent the reality of rock mass in the actual mine site. The stability of the gateroad in the analyzing results was worsened substantially. On the other hand, in the cases of that rock mass is strong and very competent as well as not deteriorated, the effectiveness of different supports was not clear, and results were in very stable condition. Using the rock mass deterioration approach in analysis, it can be realized that the passive type support, steel arch, is more effective in improving gateroad stability than active type support, rock bolt. There was another mentionable circumstance. It was that by remaining thicker coal on the roof due to the coal seam dip becoming steeper, the roof condition had improved considerably. This experience has been widely applied in many underground coal mining practices. But, it also has a negative influence that elevates the risk of spontaneous coal combustion during mining operations. So, the optimization of the ventilation system will be considered the main role in mitigating dangerous situations related to spontaneous coal combustion.

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CHAPTER FOUR. THE EFFECTS OF LONGWALL PANEL EXCAVATION AND PROP SUPPORT CHARACTERISTICS

4.1 Background

Excavation of longwall panel and its operational consistency is essential and most important part of this method. As presented in previous chapters, this chapter will investigate the diverse effects of geological and geotechnical conditions and their influences on the longwall panel retreat by comparing the analyzing results. Moreover, the process of panel excavation and caved-in zone in numerical analysis are crucial issues for the steep coal seam underground mines. The problems that miners/engineers are likely to deal with in the actual situations of longwall panel retreat and the caving mechanism are relatively different from the horizontal and gently inclined coal seam mining. In those cases, a lot of symmetrical characteristics of displacement, fracture zone and stresses are observed, especially in numerical analysis. Furthermore, the support systems which are applicable to be used and maintain the face stability effectively during the mining operations will be discussed. The highly evolved shield and chock shield supports sometimes are not suitable for steep coal seam extraction because of the weight and unavoidable difficulties related to stabilization in the face. Thus, alternative longwall face support options should be considered, and justification for the support system selection needs to be reasonable.

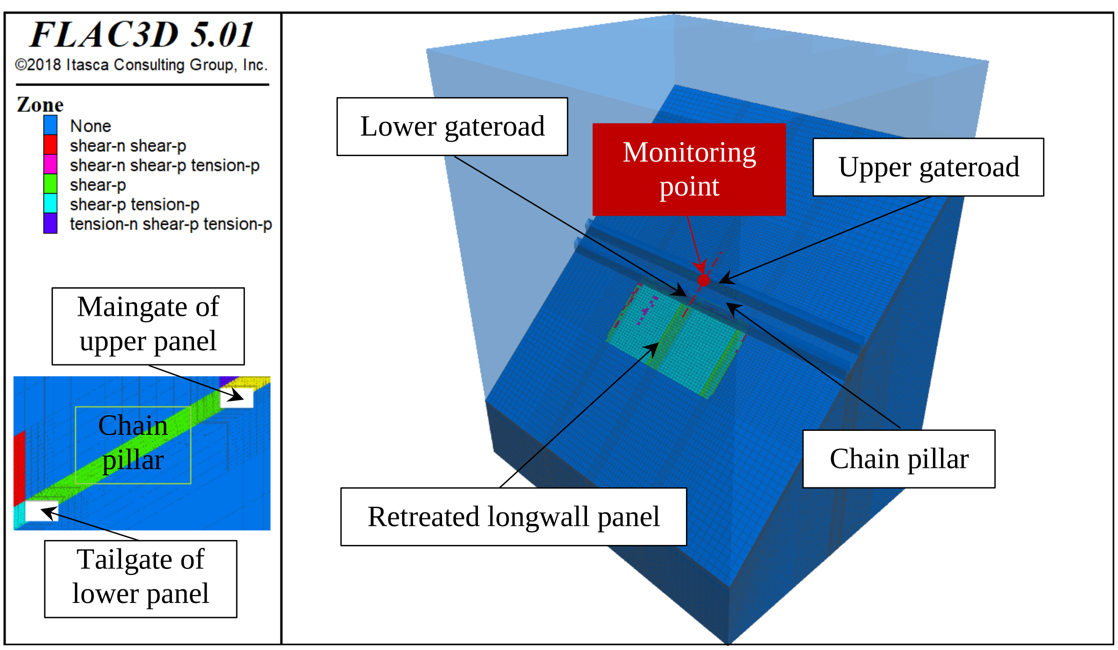


Figure 4.1. View of excavated longwall panel in the model

4.2. Gob modeling and mechanical properties

Passing the longwall face excavation, the immediate roof rock strata above the mined-out area bend and cave into the void behind the longwall face, called caved area or gob. The gob is made of collapsed rock pieces; hence it was modeled as the aggregate of fracture rocks (*Yasitli & Unver, 2005; Deth Pongpanya, 2017*). To simulate comprehensive longwall mining analysis, coal seam and immediate roof whose height is calculated with the mathematical equation for estimating the height of caved roof rock proposed by (*Yavuz, 2004*) are excavated, and then the entire void is filled with very soft and weak broken rock materials immediately. Determination of the actual gob material properties and caving mechanism behind the longwall face under different geological and geotechnical conditions is very complex because of the inaccessibility; there are no unquestionable methods for gob modeling. Regarding longwall mining with horizontal or very gentle coal seams, the approach for caved roof estimation suggested by (*Yavuz, 2004*) was utilized, and the equation is presented below.

$$H_{cave} = \frac{100*h}{c_1*h+c_2} \qquad (2)$$

where; H_{cave} is the height of caved roof rock, h is the height of the longwall face, c_1 and c_2 are the constant coefficients depending on the lithology and compressive strength of the rock. Ahead of selecting the constant coefficients, the uniaxial compressive strength of immediate roof strata should be determined. General view of the gob modeling process in numerical analysis is shown in Figure 4.2. The gob properties used in numerical simulation has taken from research publications written by *(Thin, Pine & Trueman, 1993; Xie, Chen & Wang, 1999; Yasiti & Unver, 2005)*. The process of longwall face excavation and gob making are designed simply in the analysis. The longwall face is excavated at first. Then the upper part of the rock above the previously mined face is removed, and the void is filled with gob material that properties are shown in Table 4.1. These steps will be repeated sequentially until the entire panel is excavated.

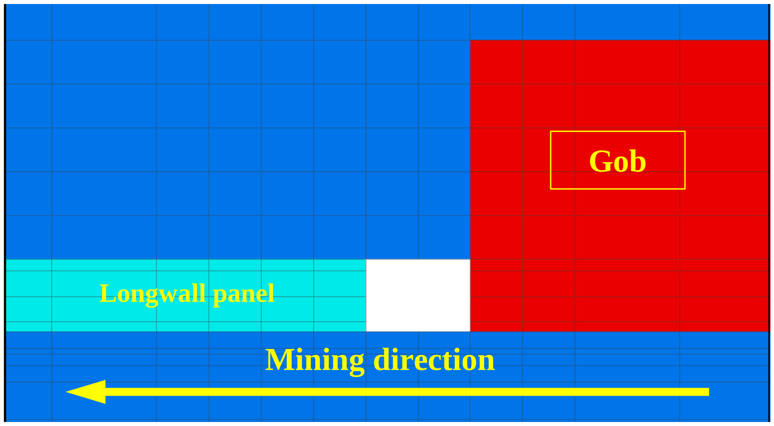


Figure 4.2. Process of the gob formation in numerical analysis

The approach was originally developed in China based on the numerous field data and practical investigations for longwall mining under various conditions, including diverse lithological and geometrical characteristics. Moreover, this equation has been widely used in US and Australian mining practices and designs where rock masses are mainly classified from medium to strong. As discussed in the previous chapter about rock mass strength evaluation, the rock mass in the NS coal mine can be classified as fair rock according to GSI and RMR values. The worst scenario of longwall mining is that the roof rock is strong, and the floor rock is weak. Generally, the proper circumstance of face excavation is that after the longwall face retreat is advanced, the roof rock above the mined face should immediately fall down into the void. It can ease severe ground conditions, especially preventing high loading on the face support due to strong roof rock that is not easily caved.

Table 4.1. The mechanical properties of gob

№	Parameters	Unit	Gob
1	Bulk modulus	MPa	-
2	Shear modulus	MPa	-
3	Young's modulus	MPa	15.0
4	Poisson's ratio	-	0.25
5	Tensile strength	MPa	-
6	Friction angle	deg	25
7	Cohesion	MPa	0.001
8	Density	kg/m^3	1,700

On the other hand, the equation proposed by Yavuz is relatively deficient in creating gob in numerical simulation for steep coal seam underground mining. This approach is more appropriate for horizontal or gently dipping coal seams extraction. Because the caving mechanism is quite different in extraction for steep coal seam compared to horizontal coal, as illustrated in Figures 4.3 and 4.4.

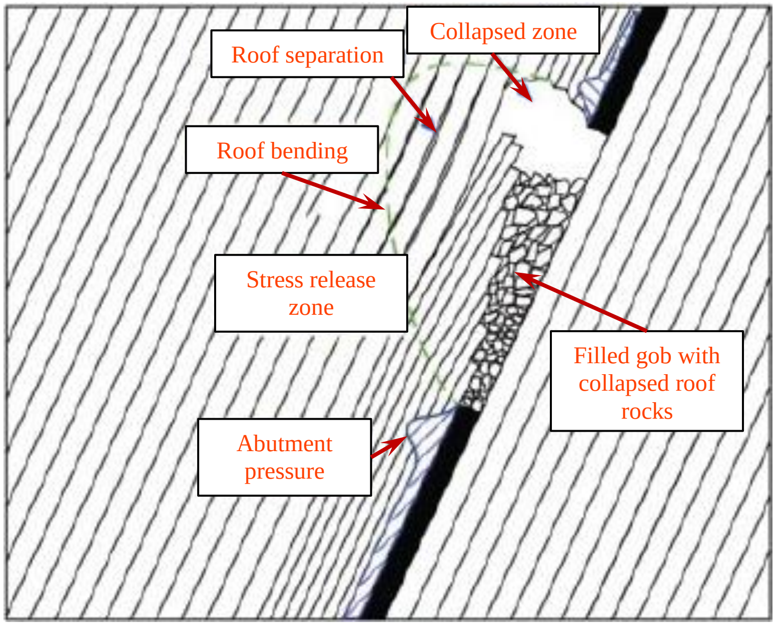


Figure 4.3. The typical concept of the caving mechanism in steep coal seam

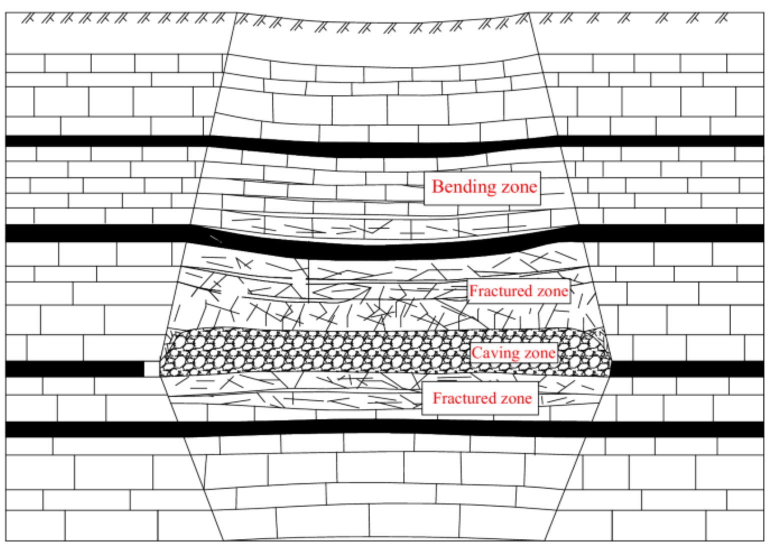


Figure 4.4. The caving mechanism in horizontal coal seam

Unfortunately, there are no standard methods or approaches to specifying the rock-caving mechanism in the gob area of steep coal seam longwall mining. Therefore, some simplification has been made in numerical analysis because the changes in rock mass behavior and ground conditions are the most significant points in the research scope. Absolutely, the influence of caving mechanism in longwall mining is very important.

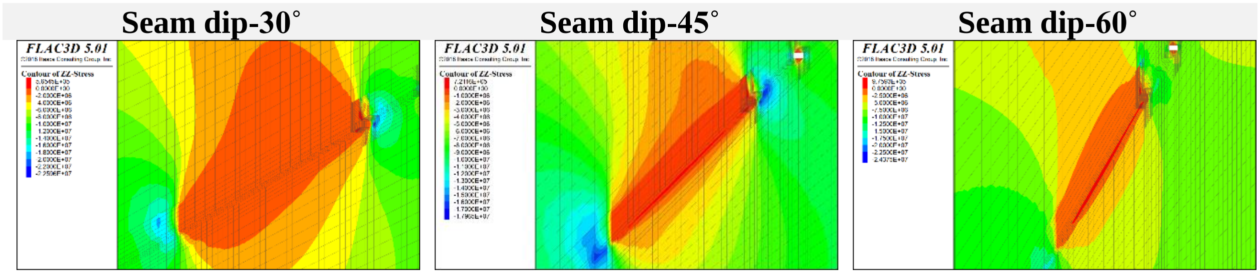


Figure 4.5. The stress distribution around the gob

Although the pattern of the vertical stress cannot indicate the overall scene of the caving mechanism around the gob, some similarities can be observed, as explained in Figure 4.3. It can be realized that Figure 4.5 shows the large potential of larger caving zone that could occur in inclined coal seam mining than in gob of steep coal seam panel excavation. The size of stress-relieved zone contours which is red area in numerical results becomes to be smaller when the coal seam becomes steeper. Besides that, the highest abutment stress, mainly concentrated near the upper end of the panel, is around 22.5MPa when the coal seam dip is 30˚. Depending on the high vertical stress distribution, especially near the upper end of the panel, the abutment stress on the chain pillar can be high, leading to serious damage. In this chapter, three distinct phases of longwall panel excavations have been considered in order to investigate the diverse effects of geological and geotechnical conditions. Also, the way how those factors affect the stability of gateroad, chain pillar and longwall face.

1. Unexcavated – Ahead of panel retreat.
2. Half – The panel retreat reaches the middle of panel.
3. Entire – The entire panel is retreated.

All numerical results that have been obtained from those three cases illustrated in Figure 4.6 will be discussed and interpreted by using the displacement, fracture zone and vertical stress magnitude. Based on the discussions, the adequate notion for steep coal seam longwall mining in several conditions can be summarized and it helps to determine the appropriate design for steep coal seam extraction.

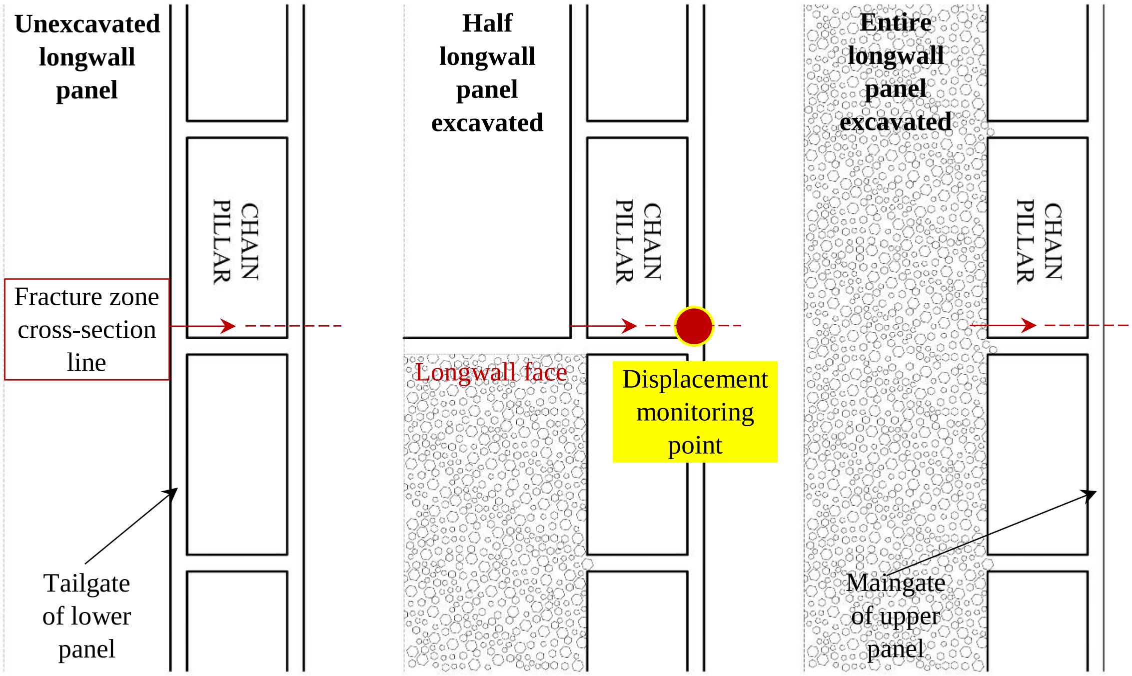


Figure 4.6. Illustrations of three different mining phases

4.3. The effect of coal seam dip

In terms of the different seam dips, the substantial difference in roof displacement taken from measurement point set on the center of the upper gateroad has occurred in seam dip of 60˚, it has been dramatically increased until panel is excavated entirely. The value has been varied from 4.9mm to 14.1mm, around 2.9 times. But, the displacement values of seam dip-30˚ and 45˚ were not increased significanlty compared to the seam dip-60˚.

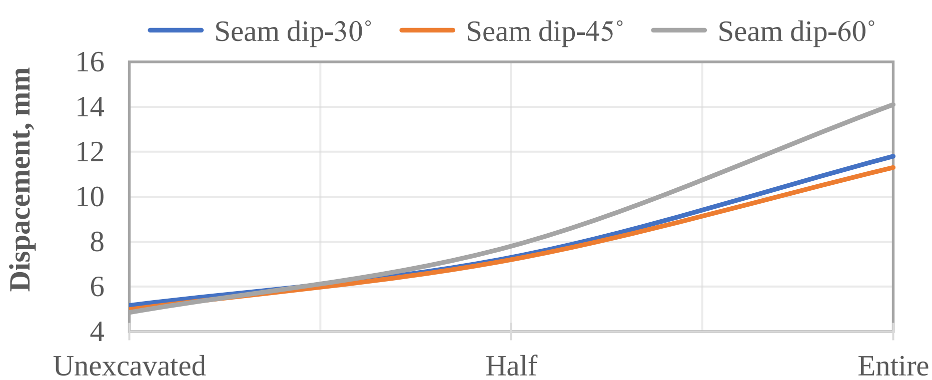


Figure 4.7. The relationship between the roof displacement and seam dip

It is obvious that the excavation of the entire panel should adversely affect fracture zone propagation around the gateroad because of induced stress and stress re-distribution around the mining area. When the lower panel retreat comes to its half as on the same line with the monitoring point already set in center of the maingate presented in Figure 4.6, the sides of the rib and floor consist of sandstone have yielded more than the sides of coal. The floor condition has been worse than on other sides since the first mining phase, even though a large fracture has been gradually generated on the roof when the panel retreat advances on. After the entire panel is excavated, the fracture zone has enlarged and deepened into the floor and rib which consist of sandstone more. The difference of fracture zone propagation and type of fractures can be clearly showed in Figure 4.8. Shear failure is major while tension is occasionally occurred in the results.

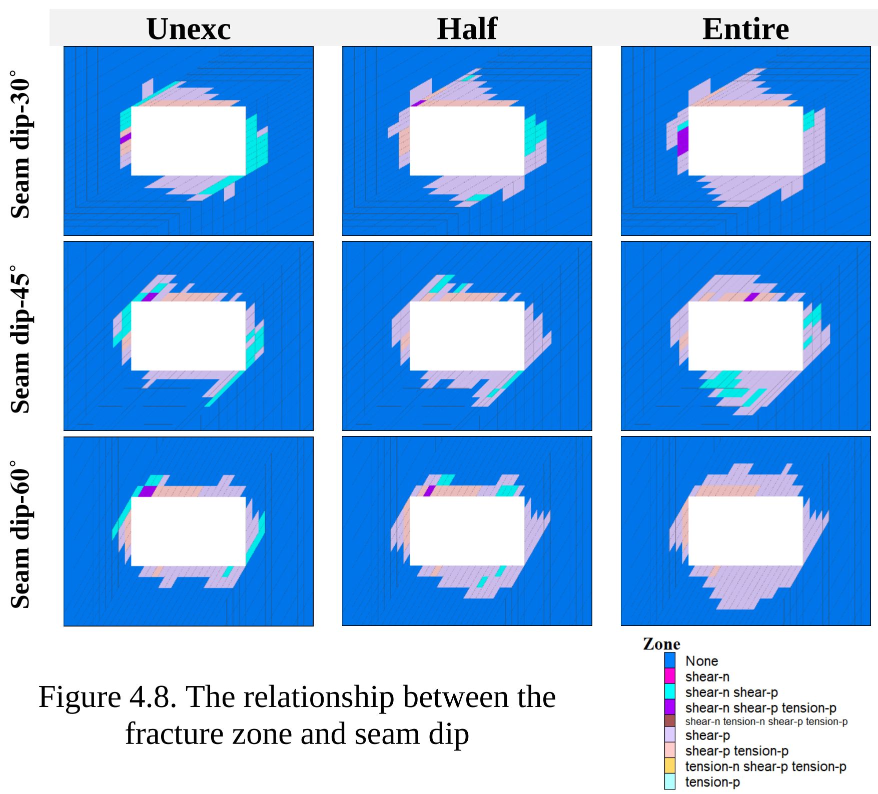


Figure 4.8. The relationship between the fracture zone and seam dip

4.3.1. The induced stress characteristic

A lot of diverse features of steep coal seam underground mining have made it distinct from horizontal coal seam mining. Due to the various seam dip angles, stress magnitude and distribution need to be discussed and must not be remained out of attention. So, the vertical stresses in three cases were measured from a total of 16 points differently spaced and set in 15m into the ribs of the maingate of upper panel and illustrated in Figure 4.9. The purpose of doing this measurement is to determine how the panel retreat influences the stress redistribution nearby the maingate.

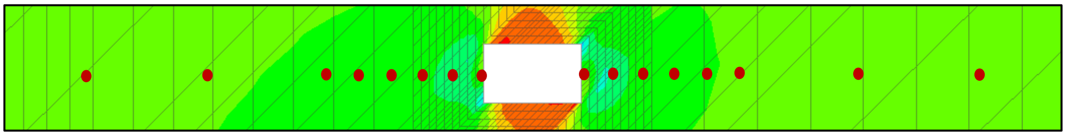


Figure 4.9. Locations of stress monitoring points around maingate

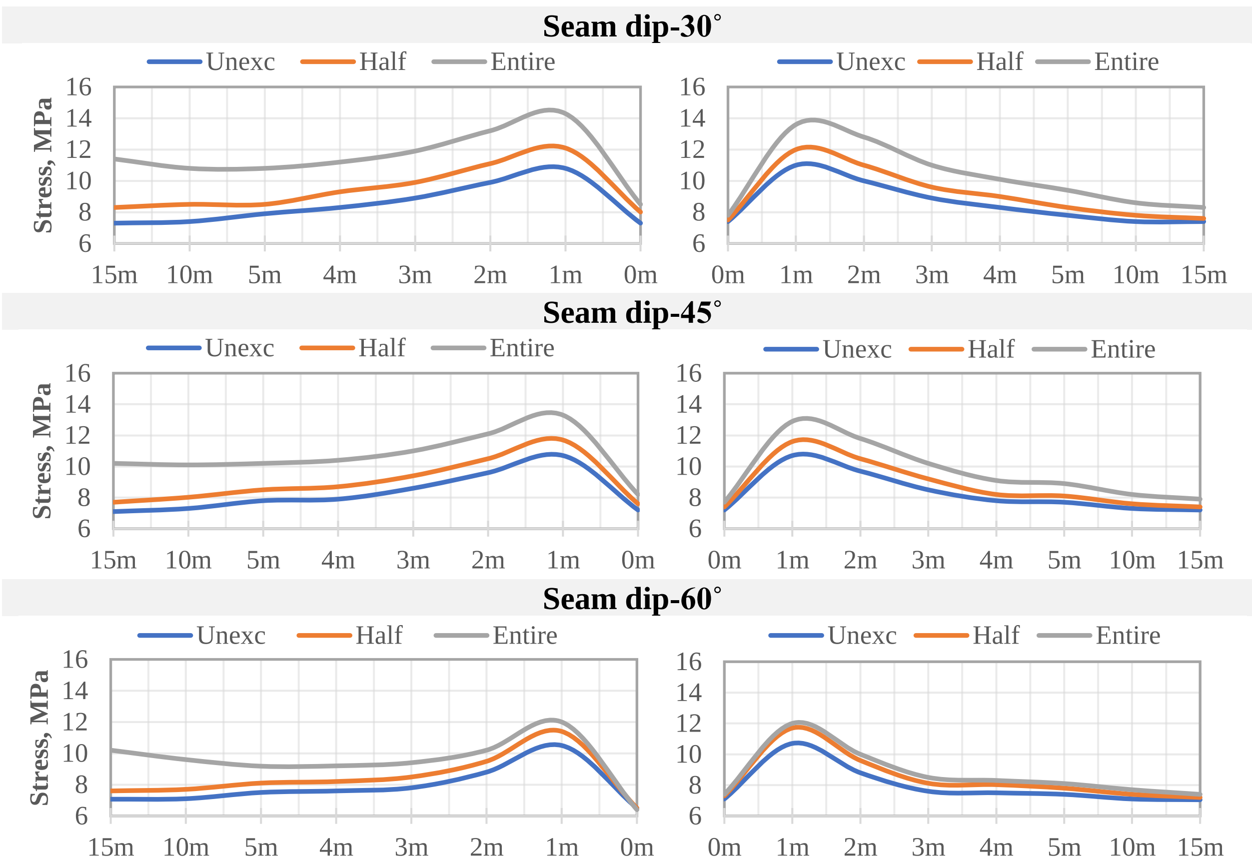


Figure 4.10. Stress distribution on the ribs of upper gateroad

Higher stress concentration usually occurs within around 2m into both ribs. The highest stress value is 14.3MPa in the seam dip-30˚ and the smallest stress is 6.4MPa in the seam dip-60˚ after the longwall panel is entirely excavated. Another interesting circumstance observed during the measurement is that the vertical stress on the side of the panel retreating has gradually been increased only in the case of the panel excavated. It notes that the panel extraction can result in larger induced stress redistribution around upper part of chain pillar's bottom end that will be discussed in next section about the stability of chain pillar. Moreover, the gentle coal seam dip mainly causes of substantial difference of induced stress concentration. It can be proven that the highest vertical stress in the seam dip-30˚ is 14.1MPa, while 11.7MPa in the seam dip-60˚. On the other hand, the stress distribution around tailgate of lower panel is shown in Figure 4.11.

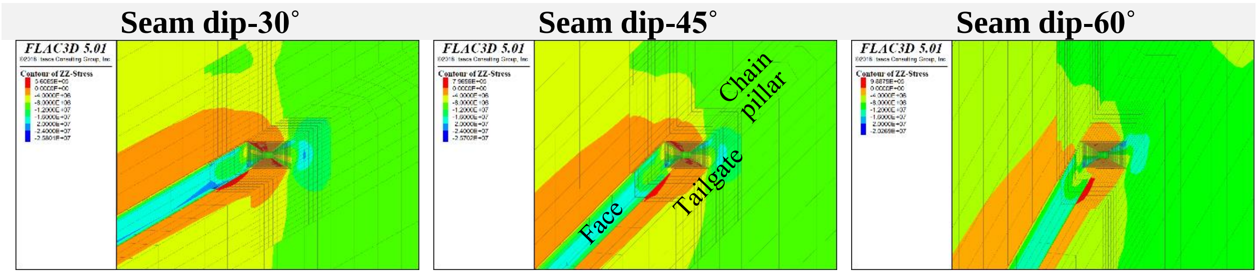


Figure 4.11. Stress distribution around tailgate of lower panel
in the half panel excavated mining phase

Large magnitude of abutment stress is seen on the opposite rib of panel, when panel excavation reached middle of it. The stress values range from 20.2MPa to 25.8MPa and more concentrated nearby lower left and upper right corners in the gateroad. However, these vertical stresses are taken from horizontal measurement lines into the ribs. For the horizontal seam mining, stress distribution along the longwall face is relatively consistent and significantly increases adjacent to the gateroad. To investigate the stress distribution through the longwall face, the vertical stress values have been collected from the measurement points set on the center of the face through the panel advance direction, as illustrated in Figure 4.12. According to Figure 4.12, the highest stress has occurred on the upper part adjacent to the gateroad, which is approximately 19.7 MPa in the seam dip-30˚, while the smallest value is 13.6 MPa in the seam dip-60˚ when the panel advanced to half. Vertical stress magnitude gradually decreased until around 10m into the panel and since then it has been stabilized ranging from 8 MPa to 9 MPa.

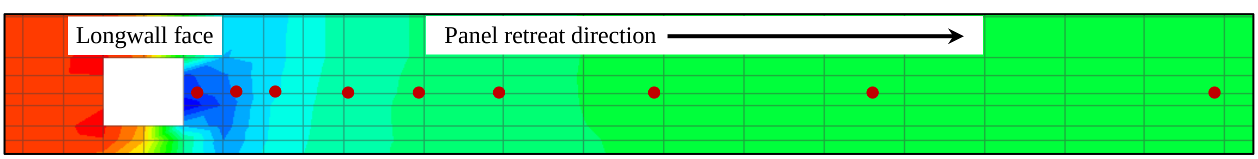


Figure 4.12. Locations of stress monitoring points along the longwall panel

Seam dip-30˚

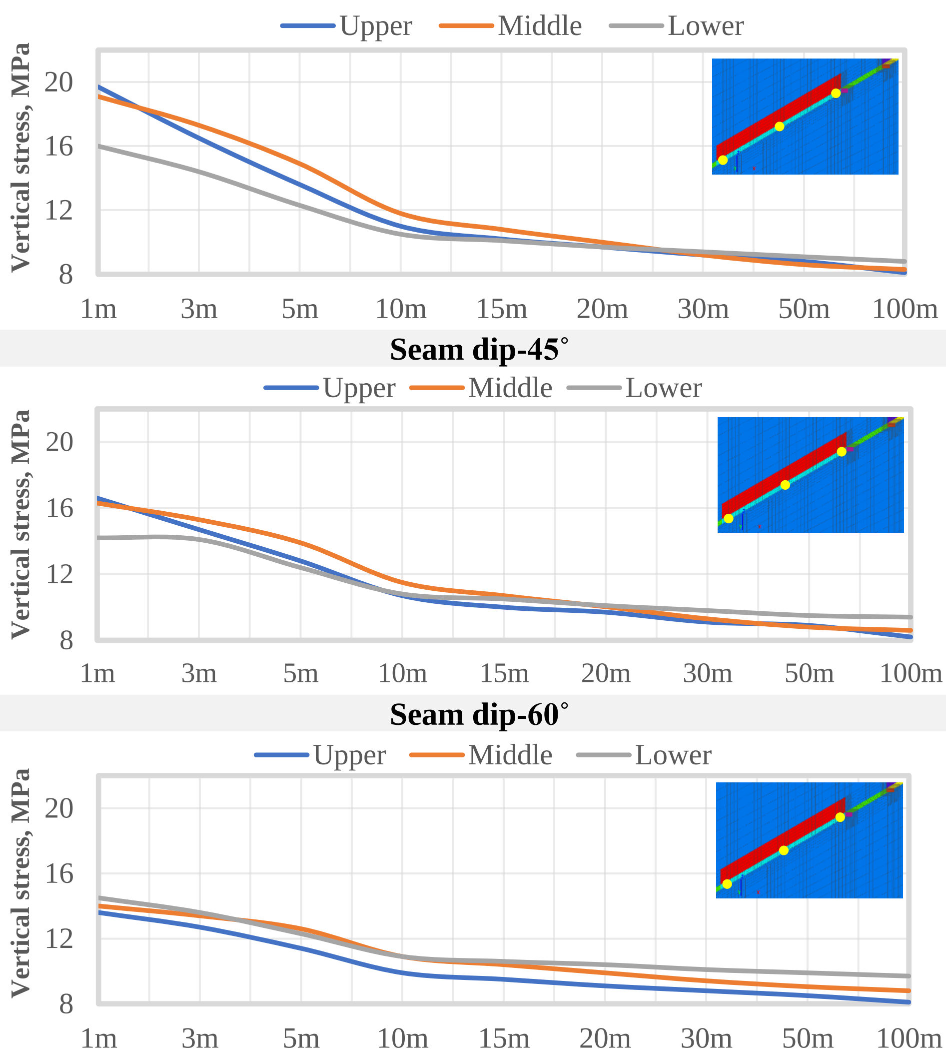


Figure 4.13. Stress distribution along the panel retreat direction

Regarding the high-stress concentration in the longwall face near the gateroad, some additional or secondary support needs to be installed in that area as well as support spacing and support row spacing can be augmented. Besides, the stresses of the upper and middle parts are steadily higher in small quantity for the seam dip-30˚ and seam dip-45˚ than for the seam dip-60˚. In the case of the seam dip-60˚, the stress of the lower part is slightly higher than the other two positions.

4.3.2. The stability of chain pillar

To investigate the adverse impacts of longwall panel excavation on the other openings of longwall mining, the vertical stress characteristics and failure zone development have been illustrated to explain the major principles. As shown in Figure 4.14, a massive fracture zone where shear failure prevails above the excavated longwall panel has not been observed with the different coal seam dips. Due to coal inclinations, the general pattern of failure is not arc-shaped and regular. A shear fracture zone has slightly more been propagated into the roof rock on the bottom end of the excavated panel. But, based on the overall view of fracture zone propagation around gob, an extension of the roof rock fracturing is not large that surface subsidence can be smaller with steeper coal seam extraction.

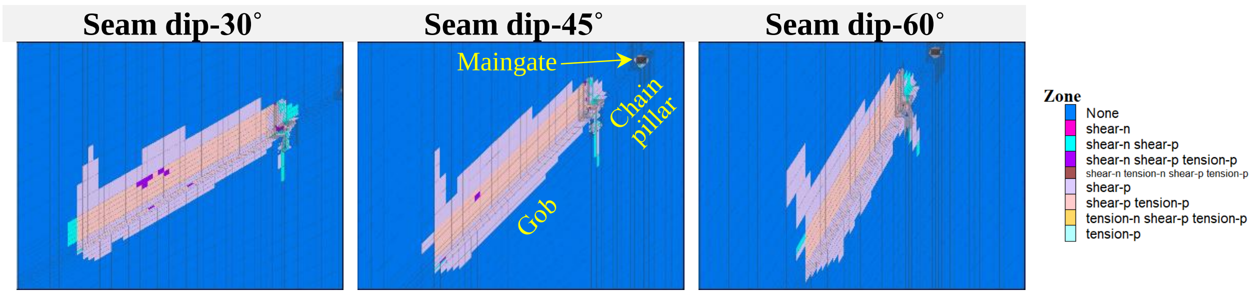


Figure 4.14. Fracture zone propagation around the gob

The chain pillar dimension can be designed according to high abutment stress and it also significantly affect the cut-throughs and gateroad on other sides. There are two main schematic options for gateroad locations within the high abutment stress induced by panel excavation; a) Wide chain pillar in order to locate gateroad away from high abutment stress b) Narrow chain pillar in order to induce controlled pillar yield so that gateroad is then located in a stress relieved zone. In terms of two-entry system designated in this study, the wider pillar should be considered in order to locate maingate of upper panel distance from the high abutment stress is more suitable. To understand induced stress distribution through chain pillar and gateroad, Figure 4.15 is depicted. The highest vertical stress area has moved up from next to the lower gateroad with coal seam becomes steeper.

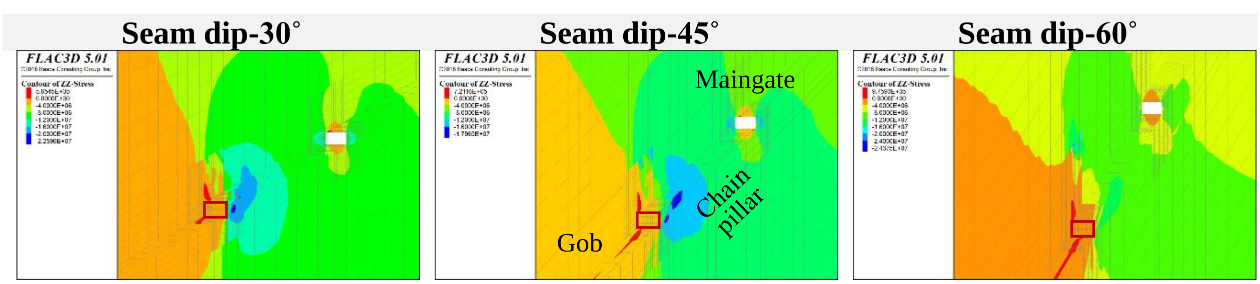


Figure 4.15. Induced stress distribution manner through chain pillar

Based on the numerical results, it can be seen that the gateroads of the seam dip-30˚ and seam dip-45˚ are in a high abutment stress profile. Vice versa, the gateroad of the seam dip-60˚ is in stress relieved zone adjacent to yielded pillar as similar as the second schematic option for locating gateroad. Substantial stress concentration through the chain pillar has gradually decreased with seam dip increasing. Also, Figure 4.16 can clearly show the relationship between the seam dips and stress values collected from measurement point set on the center of the chain pillar as well as three different panel excavation cases are assumed.

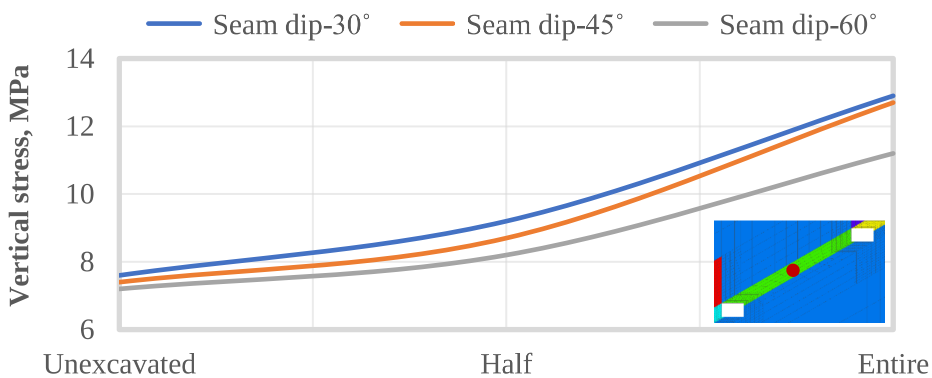


Figure 4.16. Stress on the center of pillar in the mining phases

The changes of stress magnitude are quite small and after the entire panel is retreated, stress is increased from around 7.5MPa to 12.9MPa in maximum for the seam dip-30˚. It can be realized that panel excavation of steeper coal seams does not affect significantly the stress increment in the pillar compared to inclined coal seams. Circumstance of stress increment is relatively similar in case of coal seam dip-30˚ and seam dip-45˚. The differences of the highest stress increment between the seam dips of 30˚/45˚ and 45˚/60˚ are around 4.6MPa and 6.4MPa. The induced stress distribution along the pillar is presented in Figure 4.17 and data is obtained from three locations in the pillar. There is interesting circumstance occurred for the stress distribution that the stress magnitude near the tailgate of lower panel in case of seam dip-30˚ is 16.9 MPa. Although in cases of seam dip-45˚ and seam dip-60˚ are 9.5 MPa and 4.3 MPa. The reason of it is bottom end of chain pillar is connected to the roof of gateroad in steeper coal seams.

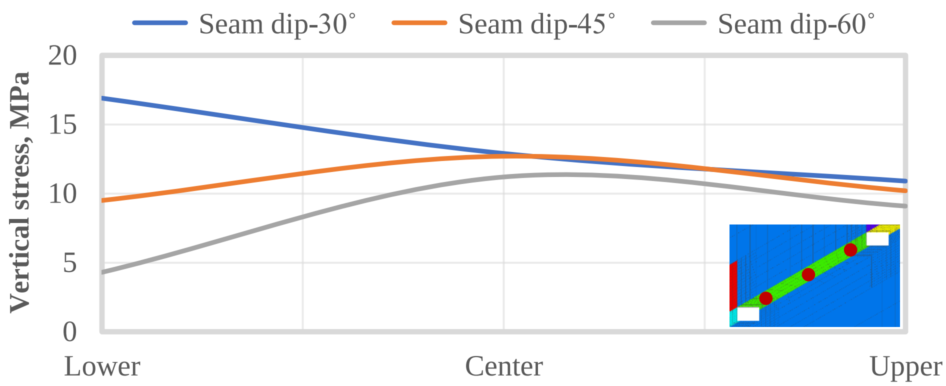


Figure 4.17. The induced stress distribution along the chain pillar

The higher abutment stress is usually occurred on the rib. General trend of induced stress is increased in the pillar center and then decreased on the two ends that are adjacent to the tailgate of lower panel and maingate of upper panel for the steep coal seam cases. In contrast, the induced stress for inclined coal seam case has gradually been decreased from lower end to the upper end of chain pillar. It can emphasize that the impact of panel excavation in an inclined coal seam on the abutment stress acted on the chain pillar is more significant than steep coal seam. Therefore, in the case of the seam dip-30˚, the wider chain pillar should be designed in order to maintain the proper stability of the maingate of the upper longwall panel.

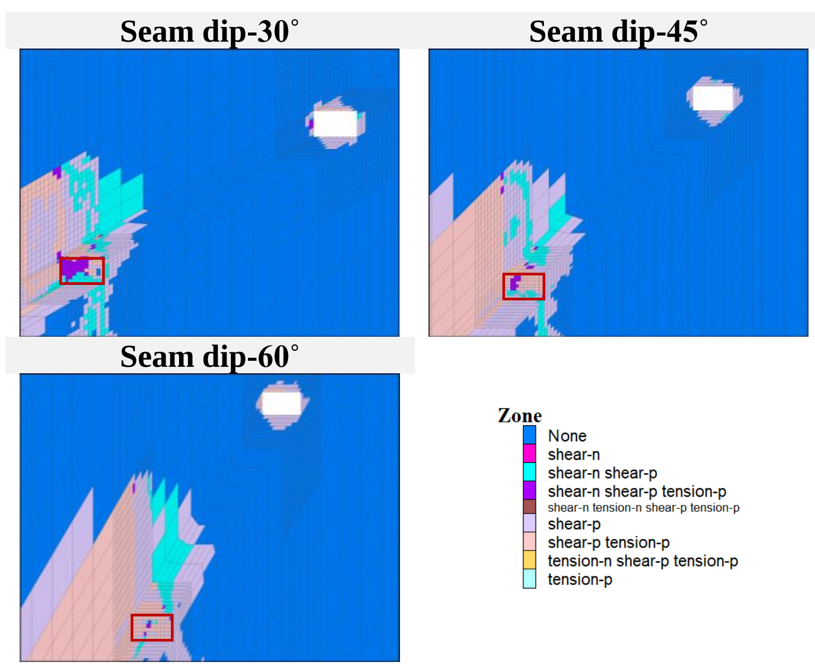


Figure 4.18. The fracture zone propagation through the chain pillar

However, another important factor that the way of fracture zone propagation through the pillar needs to be considered concurrently. Thus, Figure 4.18 presents the overall view of failure zone development and characteristics. According to analyzing results, the fracture zone's size is larger in the seam dip-60˚. It can be assumed that the caving mechanism in steep longwall panel excavation and the mining area's location below the upper gateroad could be reasons for the fracture zone propagation towards the chain pillar. Shear failure is dominant as similarly prescribed in other results.

4.4. The effect of mining depth

To investigate the effect of mining depths, which are 300m, 500m and 700m, on the longwall panel excavation and chain pillar conditions, a numerical model in which a seam dip angle of 45˚ and k ratio of 1 is set has been utilized. The NS open pit mine’s final pit limit, which is determined by the economical stripping ratio, is designed to be 300m. But, the company and investors have a very good expectations to continue the coal mining project further. As per their engineering group’s plan and geological exploration data, mine depth could be reached deeper than 500m. In order to clarify the uncertainties and doubts related to the potential mining depth and lifetime of the project, additional exploration work, especially drilling work into the deeper ground, should be done ahead of further economic and technical assessment for the NS coal mine. Regardless of their expectation and assessment, the three different range of mining depths are used to apply in a series of numerical analysis to define all types of impact associated with stability issues. Depending upon the mining depths, other factors that literally influence the mining design can be determined in each phase.

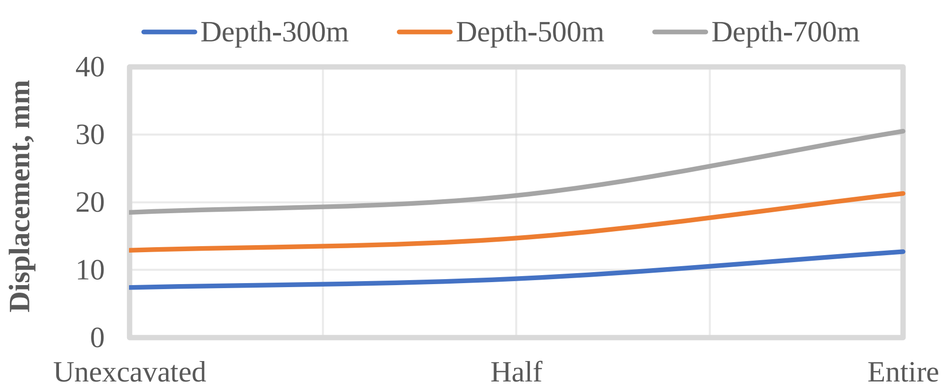


Figure 4.19. The relationship between the roof displacement and depth

When the mining depths are increased as 300m, 500m and 700m, the displacement values were increased correspondingly. Particularly, after the entire panel is excavated, initial values of displacement 4.9mm, 10.1mm and 16.4mm are ascended by 2.2 times, 2.5 times and 2.8 times. All those increases in percentage are quite systematic because mining depth increases with same quantities. This is principle related to mining conditions as similar as explained in the previous chapter. The highest displacement values are 11.3mm, 25.7mm and 45.7mm in cases of depths are 300m, 500m and 700m, respectively.

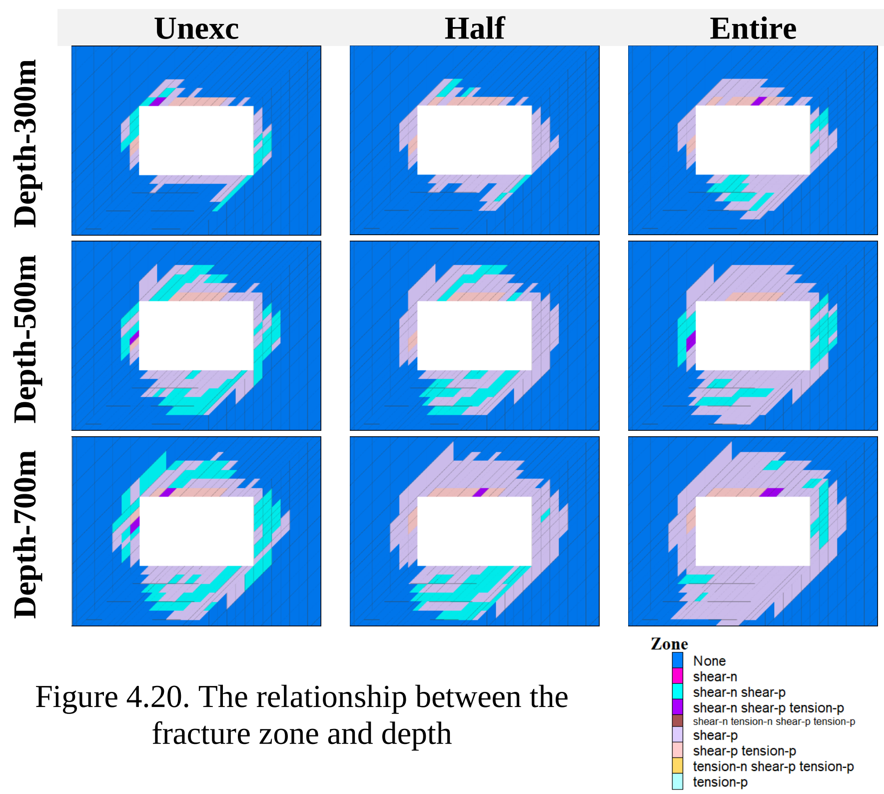


Figure 4.20. The relationship between the fracture zone and depth

The greater mining depth definitely results in larger and larger fracture zone development and propagation around the gateroad and through the chain pillar. As illustrated in Figure 4.20, once the mining depth is 300m and after the entire panel was excavated, roof and floor conditions, as well as the opposite rib of panel excavation, were apparently worsened. This phenomenon clearly shows the way how panel excavation elevates fracture zone propagation around the gateroad. Furthermore, when the mining depth is increased by 200m double, the fracture zones have wholly surrounded the gateroad, especially the roof and floor conditions became seriously debilitating. Shear failure type is very dominant in those analyzing results that can cause originating sudden sliding plane in the roof rock.

4.4.1. The induced stress characteristic

To understand induced stress redistribution around the gateroad and chain pillar and its adverse impacts on ground stability, same numbers of measurement points and locations explained in previous section that were obtained from numerical results were used. Vertical stresses of the upper gateroad ribs and three locations on the longwall face. Figure 4.21 can present the stress redistribution along the scanline set into the rib sides of upper gateroad with certain spacing of measurement points. Also, three phases that are ahead of panel excavation, half of panel excavation and entire panel excavation were considered in the analysis.

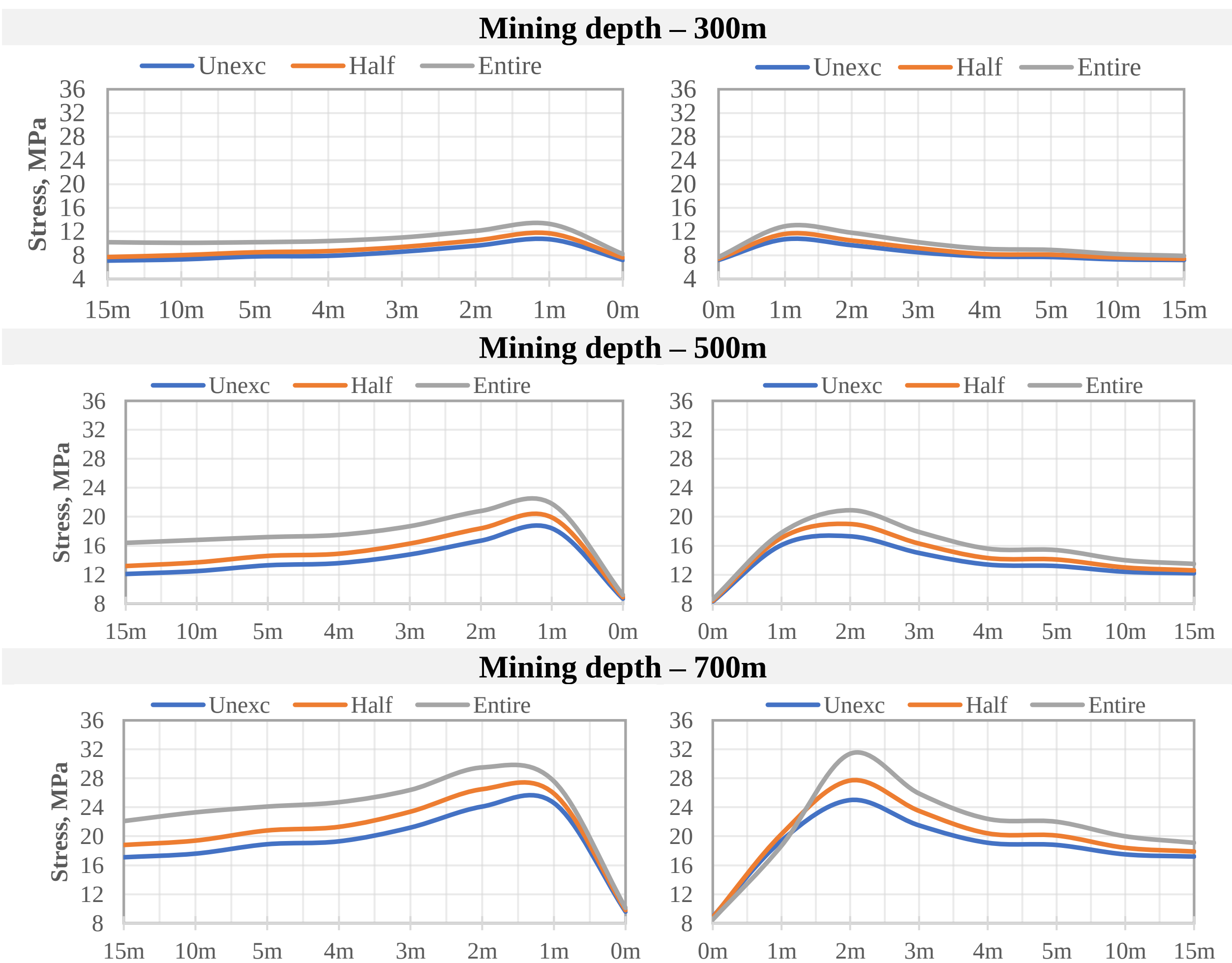


Figure 4.21. Stress distribution on the ribs of upper gateroad

When mining depths are 300m, 500m and 700m, the highest vertical stresses are 13.3MPa, 21.8MPa and 31.4MPa. Of course, these high stresses occurred in case of the entire panel is excavated. The stress released zone continues within 1m, and then high abutment stress occurs until around the next 5m into the rib on the panel side. This circumstance can be seen a bit differently on the other side as the stress released zone continues within 2m and then high abutment stress occurs until around the next 5m into the rib on the opposite side of the panel. A general trend can be seen that due to high abutment stress induced by panel excavation, the stress magnitude of the side adjacent to the panel is slightly higher than that of another side. Besides, there is progressive increase of vertical stress in each phase depending upon rate of panel advance. For example, in terms of greater depth of longwall mining before the panel retreats, the highest stress was around 25MPa and after the panel retreated entirely, the vertical increased up to 31.4MPa. This stress occurred on the rib side approximately 2m distant from gateroad surface. To realize whether it is possible to deal with this terrific high stress condition in deep underground coal mine, *K. Hang (2010)* and *C. Liu (2011)* introduced in their publication that in some underground coal mines where mining depth is over 1,200m in China and Inner Mongolia, accurately measured vertical stresses has reached around 32.8MPa. Those coal mines were still in run even though mining stress conditions are severe. According to their research, vertical stress in these numerical results can be reasonable and affordable to feasible for steep coal seam longwall mining at 700m depth.

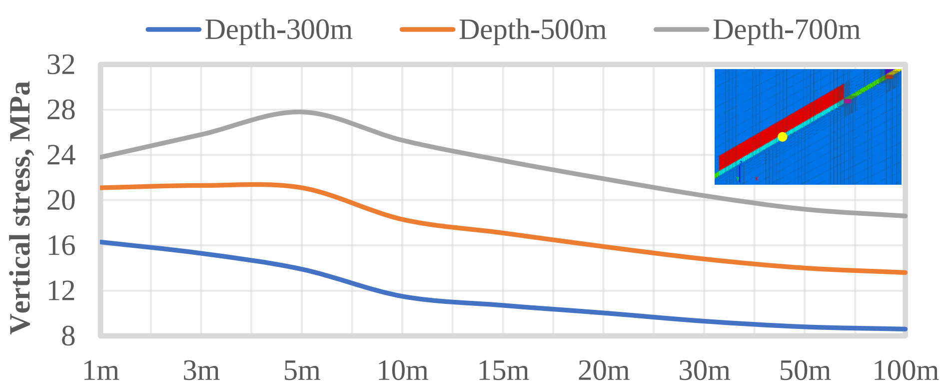


Figure 4.22. Stress distribution along the panel retreat direction

For the induced stress distribution along the panel retreat direction, Figure 4.22 shows that the highest stresses are 16.6 MPa, 24.1 MPa and 29.7 MPa for depths of 300m, 500m and 700m. These values were increased by 7.5 MPa and 5.6 MPa with mining depth increasing. The average stress increment is probably around 6.5 MPa per 200m increase in depth. Moreover, the stress along the panel becomes steady dramatically after 10m from the longwall face in case of a shallow mine. For the 500m and 700m mining depths, high stress near the longwall face dramatically decreases until 10m into the panel; since then, it has gradually decreased.

4.4.2. The stability of chain pillar

In Figure 4.23, it is seen that the massive enlargement of fracture zone propagation around the face has been depicted due to the changes in mining depth. The general trend of fracture propagation aims more at the vertical direction than the horizontal direction. Compared to the fracture zone development around the excavated panel in different seam dips, the shape of the fracture zone is relatively similar to the arc pattern, although the seam dip is 45˚ or quite steep. The shear type of failure characteristic is still dominant, and tension fractures only occur in the surrounding area of the longwall panel. These circumstances influence the abutment stress distributed through the chain pillar and upper gateroad in various ways, as shown in Figure 4.24. For the fracture zone conditions in 300m depth with various seam dips, the steeper coal seam mining had resulted in the serious potentiality of pillar failure, and nearly half of the chain pillar was in a dangerous situation that can be failed or collapse down towards the gob.

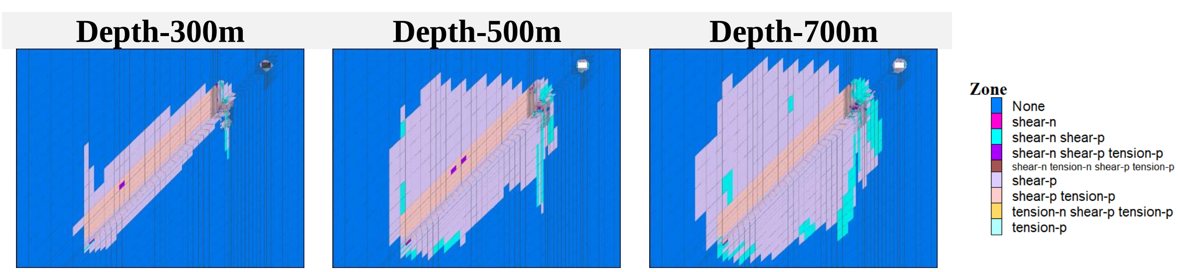


Figure 4.23. Fracture zone propagation around the panel excavation

In the greater deep mining case, stress on the lower part of the panel is higher than that of the upper part adjacent to the tailgate, and it has drastically become less than the other two cases. It can be explained as the greater mining depth results in high stress concentration on the lower part of longwall face excavation. Logically, this situation should be similar in several mining depth cases in terms of for the relationship between the depth and stress increment, but circumstances are different in other two cases.

Depth-300m	Depth-500m	Depth-700m

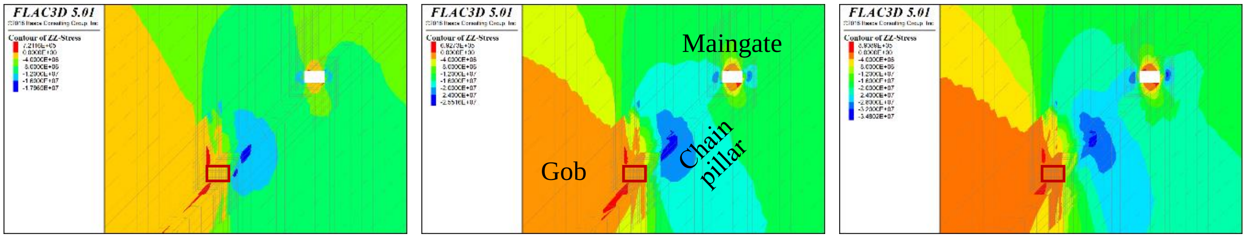


Figure 4.24. Induced stress distribution manner

Figure 4.25 shows the stress redistribution after the longwall panel is excavated. It is noted that when mining depth increases, high stress concentrated on the opposite rib side of the gateroad has connected to the stress on the rib of the upper gateroad. This situation presents that serious abutment stress has been acted on the chain pillar, leading to serious instability for gateroad serviceability and safety for miners.

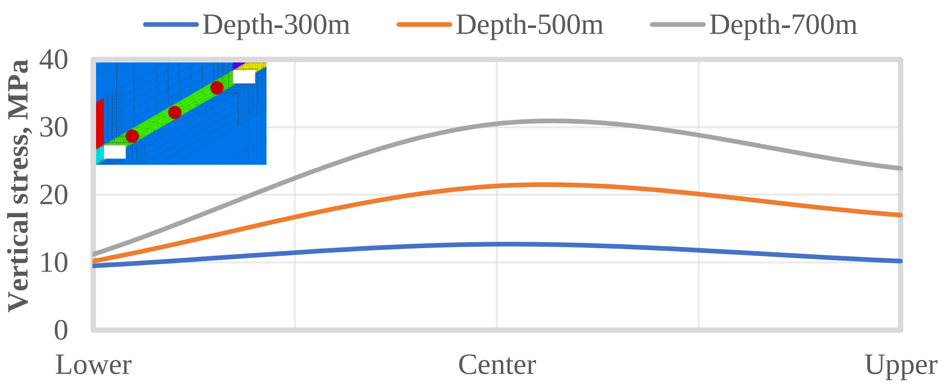


Figure 4.25. Induced stress distribution along the chain pillar

However, a 30m width of chain pillar can be proper for the 300m depth. In contrast, in order to locate away the gateroad out of the high abutment stress profile in greater depth mining, a much wider chain pillar needs to remain. Therefore, it also negatively influences the mining operation economically and technically, such as increasing the roadway drivage and mining cost and decreasing the coal recovery ratio. According to empirical formulas and calculation, Mark and Agioutantis (2018) recommend that for the tailgate loading condition, the stability factor (SF) should be about 1.4 in minimum where the roof is weak but could be as small as 0.8 where the roof is exceptionally strong. However, the pillar width of 30m is set constantly in the most of numerical simulations, the effect of 10m and 20m width of chain pillar have been investigated in case seam dip-45˚ and mining depth of 300m as similar as the NS coal mine.

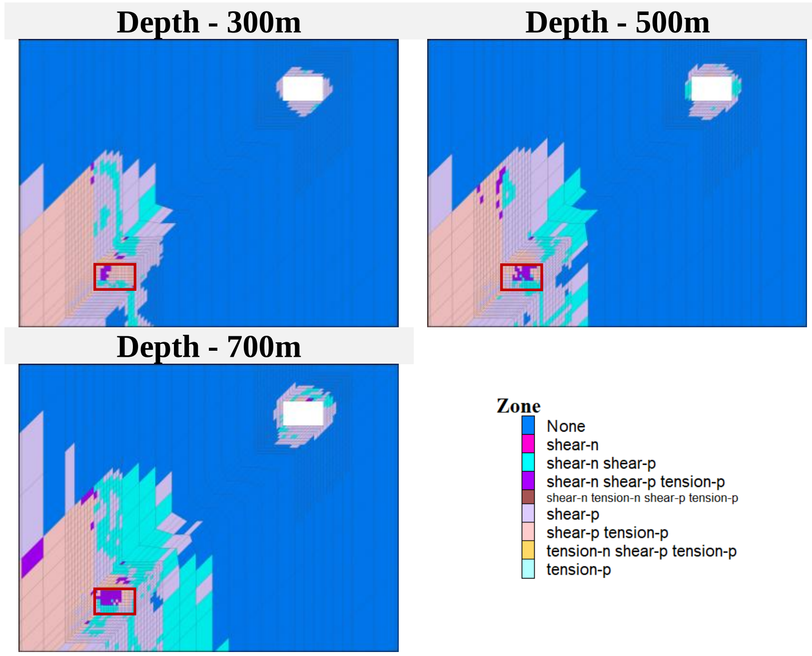
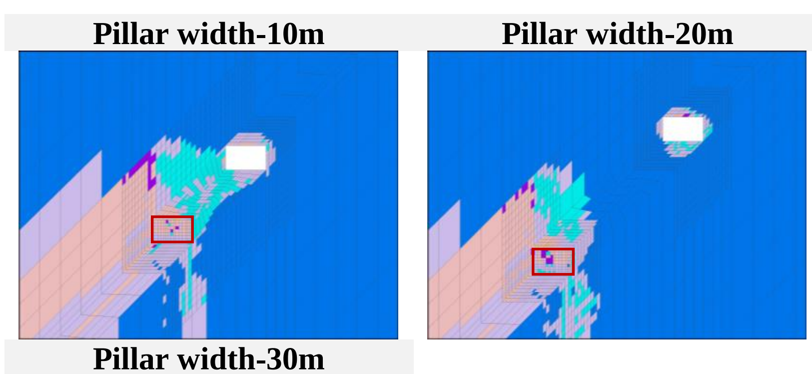


Figure 4.26. The fracture zone propagation through the chain pillar

The greater mining depth elevates fracturing increment around the tailgate of lower panel in both vertical and horizontal directions. Based on the analyzing results and discussions, when the mining depth increases up to 700m, approximately 1/3 of the chain pillar was failed with shear-type failure as the probability of pillar collapse risk has arisen in similar quantities simultaneously. However, it can be noted that although it is likely to be unsafe for the equipment and miners in the maingate of upper panel, the chain pillar width of 30m can still sustain the high loading and maintain roadway serviceability for the next panel excavation. The comparison of the different pillar width results is presented in Figure 4.27. It is seen that 10m pillar is completely failed after entire panel is excavated, in turn 20m pillar width is stable and be applicable for the NS coal mine case.



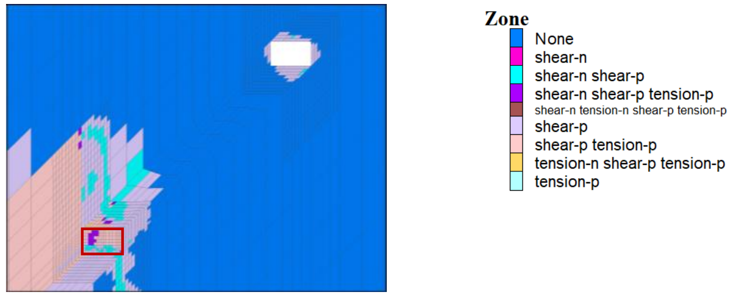


Figure 4.27. Fracture zone propagation with different pillar width

4.5. The effect of lateral stress ratio

To indicate the effects of the lateral stress ratios corresponding to the general trend of the relationship between the stress and depths, the k ratio will be ranging from 0.5 to 2.0. According to the general trend of in-situ stress regimes, when the mining depth is more than 400m, the lateral stress ratio becomes quite constant that fluctuates near 1. For the shallow mining depth, wide range of k ratio impacts have to be considered in detail. Especially, because of the adverse impact of higher horizontal stresses, approaches and solutions should be designated in advance for unexpected ground control situations and uncertainties.

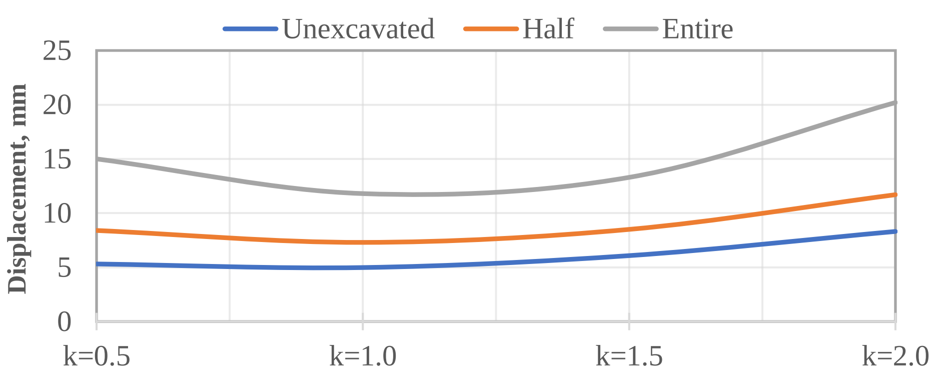
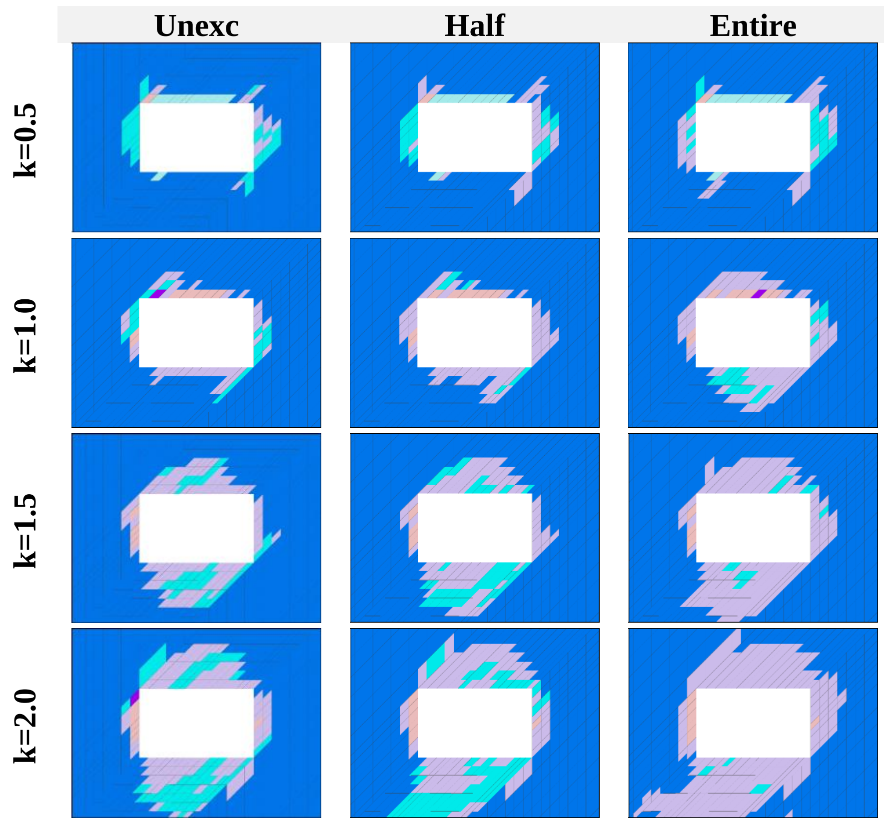


Figure 4.28. The relationship between the roof displacement and lateral stress ratio

As explained in the previous sections, the high k ratio dominantly results in higher displacement in the gateroad. The displacement value is a bit higher at the k ratio of 0.5 and then slightly decreases with the k ratio equal to 1, and it has been gradually increased until the k ratio raised to 2.0. The biggest increment of deformation was occurred in case of k ratio of 0.5. The difference in displacement value of that case, after and before the panel is excavated, was 2.8 times and from 5.3mm to 15.0mm in numbers.



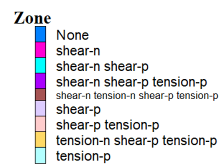


Figure 4.29. The relationship between the fracture zone and lateral stress ratio

The fracture zone has been propagated in a wide range when horizontal stress increases, especially in the vertical direction, towards to roof and floor more significantly. Regarding the fracture zone direction, fracturing tends to continue along the coal seam, particularly on the floor. It notes that the chain pillar of steep coal seam extraction has deteriorated and been highly yielded from the side of the gateroad to the panel. Also, it can be seen that there is a direct correlation between high horizontal stress and failure zone concentration that prevails on the roof and floor in substantial.

4.5.1. The induced stress characteristic

There was some scattered results of the effect of mining depths and seam dips on the stress distribution around the maingate of upper panel in the previous sections. But it is clear that mining depth apparently cause of high stress concentration and magnitude. Figure 4.30 shows that the way of how k different stress ratios affect the induced stress redistribution in several mining phases. These data were collected from 16 measurement points on the rib sides in certain spacing intervals.

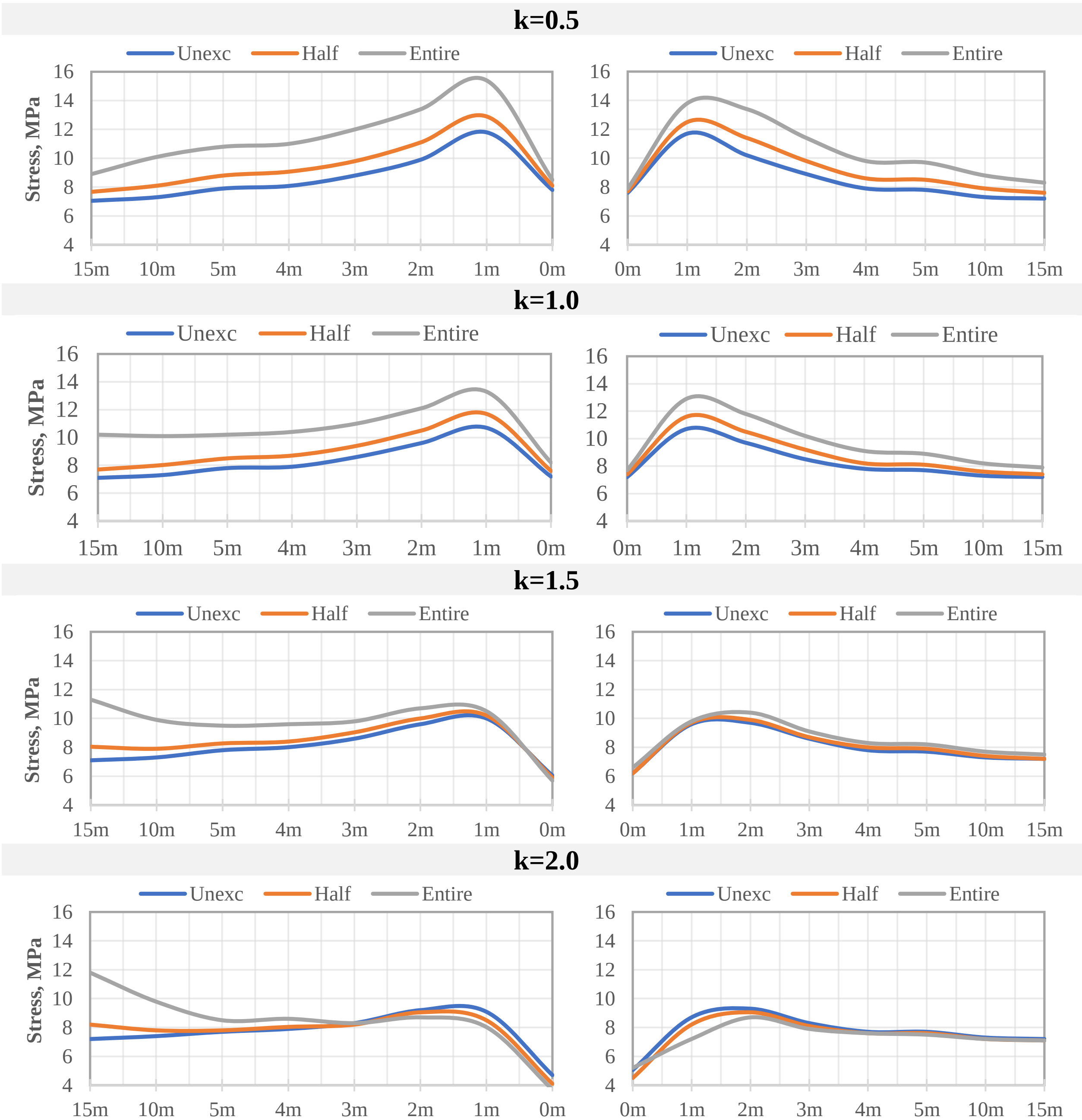


Figure 4.30. Stress distribution on the ribs of upper gateroad

According to the numerical results, there is a significant difference in vertical stresses for the small stress ratio after the entire panel is excavated in Figure 4.31. But this phenomenon will fade with the k ratio increasing to 2.0. In different mining phases related to panel retreat, a high k ratio can reduce the differentiation of vertical stress on the rib sides of the gateroad. However, while horizontal stress increases, the vertical stress on the rib side adjacent to the longwall face has drastically increased in the numerical results. It can be realized that this circumstance can express the adverse impact of high horizontal stress on the chain pillar stability. After 5m from the rib surface on the mined panel side, stress jumped from 8.5MPa to 9.79MPa and 11.8MPa at 10m and 15m, respectively. The highest vertical stress was 13.8MPa when the k ratio was 0.5 due to the high vertical stress concentration.

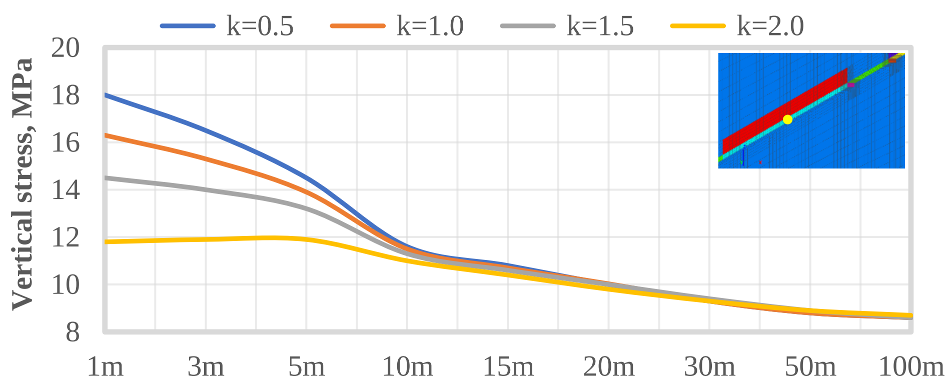
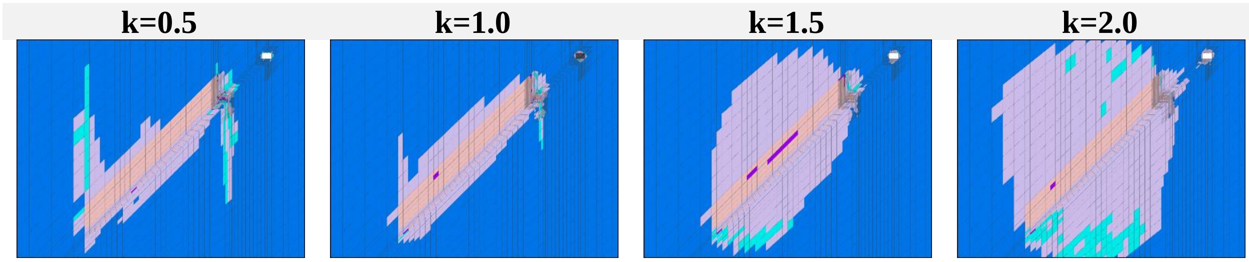


Figure 4.31. Stress distribution along the panel retreat direction

Similar circumstances can be found in the numerical results compared to the previous results. For example, the high-stress concentration only occurs within 10m from the longwall face. Beyond that range, the vertical stress condition goes into equilibrium and becomes significantly constant in magnitude. Definitely there have been direct relationship between the vertical stress and low lateral stress ratio near the longwall face excavation. The highest vertical stress values are 17.9MPa, 14.5MPa and 13.4MPa for k ratios of 0.5, 1.5 and 2.0, respectively. Nevertheless, the locations of the highest vertical stress through the longwall face are relatively different depending upon the k ratio. For instance, in the case of k ratio 0.5, 1.5 and 2.0, the highest values were on the lower, middle and upper part of the longwall face. It can be emphasized that locations of high-stress concentration transfer from bottom to top with the k ratio increase.

4.5.2. The stability of chain pillar

As shown in Figure 4.32, it is seen that fracture zone propagation in case of less lateral stress ratios is relatively smaller. Even so, when the k ratio is 0.5, the fracture zone has spread out substantially in the vertical direction at two ends of the excavated panel. For the higher stress ratios, the fracture zone has propagated uniformly in a round-shaped around the panel. Shear failure mainly prevailed.



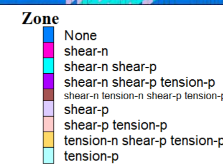


Figure 4.32. Fracture zones around the panel excavation

Figures 4.33 presents relatively strange and interesting phenomenon of stress redistribution through the chain pillar after the panel is entirely excavated. Even though the total vertical stress magnitude has substantially increased at a k ratio of 2.0, abutment stress on the center of the chain pillar has decreased. The higher stress is transferred from the upper corner of the rib toward roof area, as illustrated in Figure 4.32. Also, it can be explained that higher horizontal stress results in the chain pillar failure on the lower section, and then it cannot sustain the large loading. This phenomenon can be explained logically by comparing the results of the fracture zone in Figure 4.33 and the stress manner.

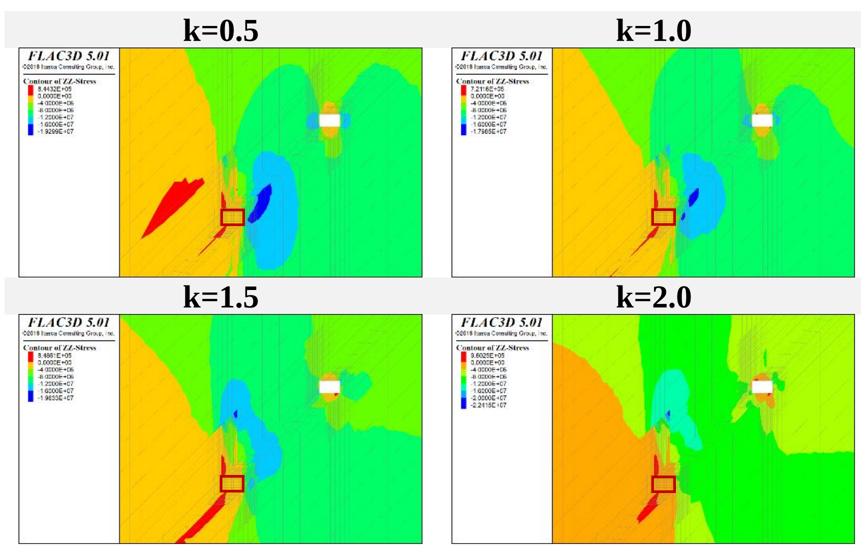


Figure 4.33. Induced stress distribution manner

The largest difference of stress reduction was occurred when k ratio is 2.0 The values has decreased from 13.3MPa to 10.2MPa on the center of the pillar. In terms of steep coal seam longwall mining, the appropriate locations of measurement are complicated rather than horizontal coal seam mining to illustrate the better description of whole manner. Moreover, induced stress redistribution is chaotic and irregular.

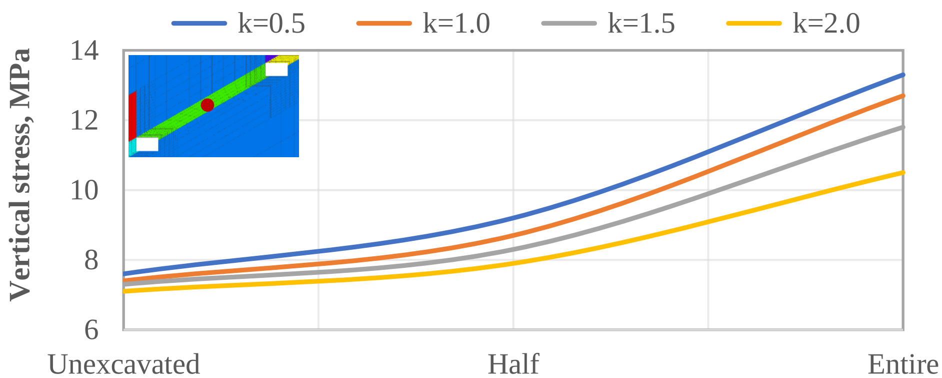
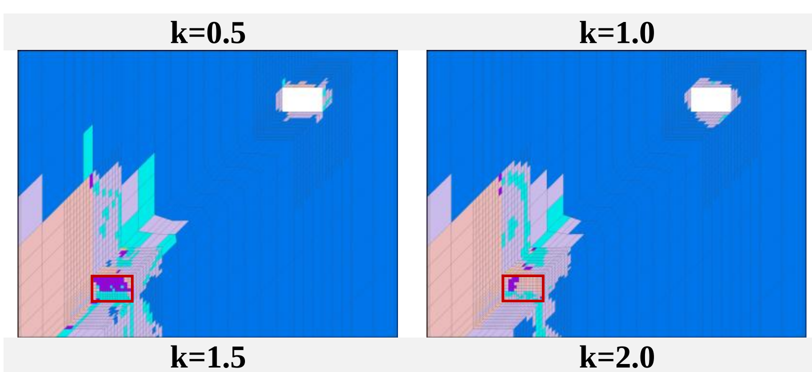


Figure 4.34. Stress on the center of pillar in the mining phases

Due to horizontal stress increment, the effect of vertical stress has faded, and the chain pillar is in an unstable condition with high-stress ratios. In the cases of k ratio 0.5, 1.0 and 1.5, a 30m wide chain pillar can still sustain the high abutment stress and maintain the safe working condition in the upper gateroad based on the analyzing results of the fracture zone. But, when the k ratio reached 2.0, most part of the chain pillar substantially yielded, and it elevated the potentiality of dangerous failure and massive rock falls. In this section, the highest and lowest values of stress magnitude were 13.3MPa and 7.1MPa, respectively. Applying the high-efficiency support in tailgate of lower panel such as cans or pumpable cribs which are most commonly used can be employed to reduce high abutment pressure acting on chain pillar and mitigate severe fracture propagation. Or the backfilling technology is one of the possible options but it influences adversely mining cost.



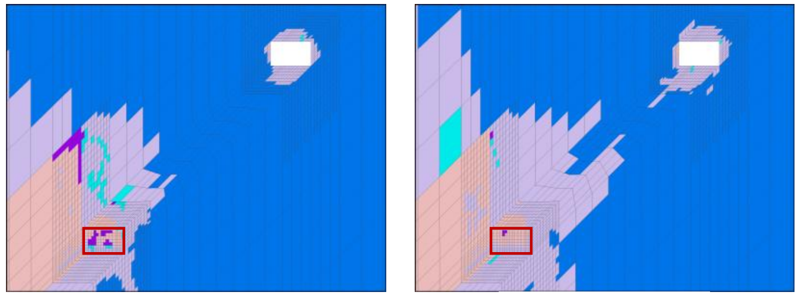


Figure 4.35. Fractures zones through the chain pillar

Based on the numerical analyzing results, particularly for the fracture zone propagation, 30m chain pillar is enough to maintain the maingate of upper panel in cases k ratio ranges from 0.5 to 1.5. Besides that, in order to enhance the coal extraction ratio, the narrow pillar whose width is less than 30m can be considered when low lateral stress ratios are applied.

4.6. The effect of rock mass deterioration

The effect of the deteriorated rock mass on the gateroad stability was significant. The displacement and fracture zone had drastically been worsened and presented clearly in the numerical results. The analyzing results that deteriorated rock mass properties were applied are more likely to represent the deformation characteristic and actual mining conditions. The substantial increases in displacement were 4.8 and 8.7 times, with initial rock mass deteriorating by 25% and 50%. In the case of the entire panel excavation and undeteriorated rock mass, displacement is only 11.3mm. It increases up to 54.8mm and 99.1mm after the Geological stress index is decreased.

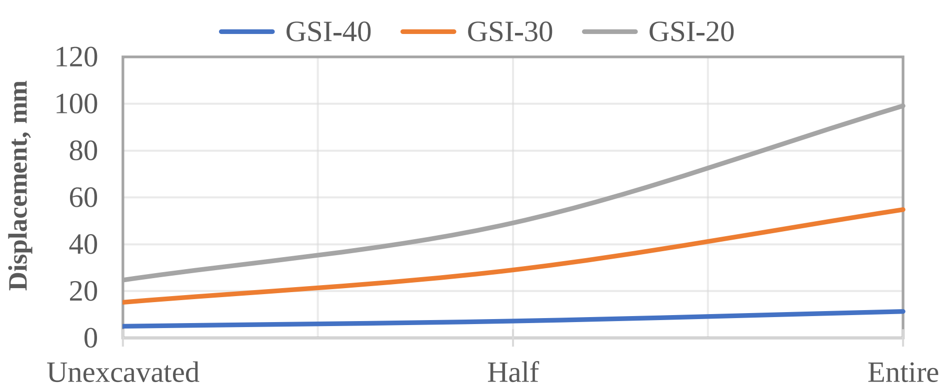
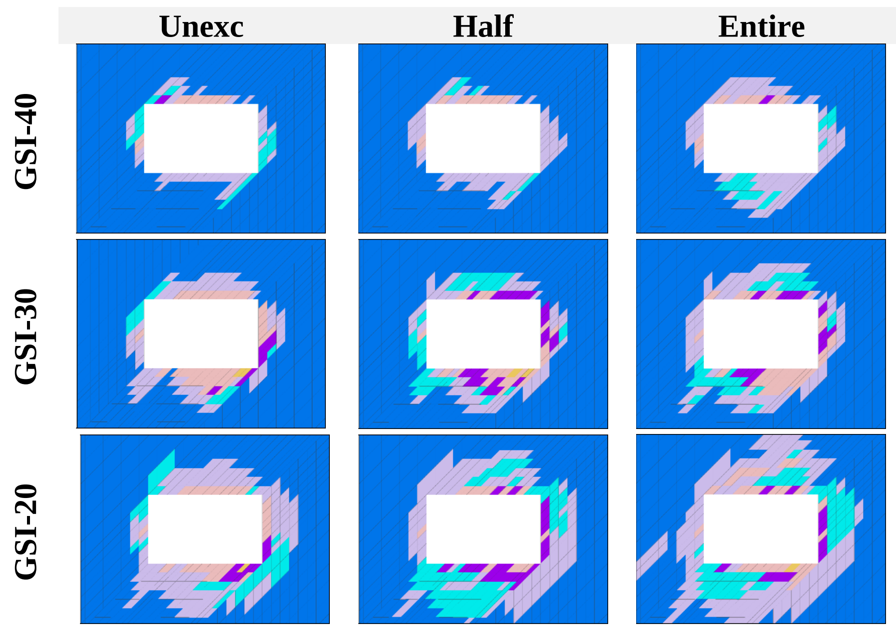


Figure 4.36. The relationship between the roof displacement and deteriorated rocks

This circumstance can probably be found in the actual mining situation. It indicates that before rock mass deterioration approach is applied in the fair and strong rock mass, the ground condition is very stable even in three different operational phases. The average value is around 7.83mm. To understand the effects of various rock mass properties, the fracture zone propagation of surrounding rock in the gateroad is important. So, the results of numerical analysis with several rock properties and mining phases are presented in Figure 4.37.



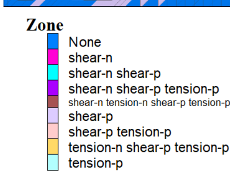


Figure 4.37. The relationship between the fracture zone and deteriorated rocks

As explained before, when the GSI is 40 and the rock is strong, although the entire panel is excavated, fracture zone propagation is not significant, and it aims to enlarge into the roof and floor slightly more than the rib sides. However, after GSI is reduced by 25% and 50%, the fracture zone has been propagated in all directions, and the rib condition worsened. Moreover, the fracture zone further overspread to the chain pillar, and shear failure types still prevail.

GSI-40 **GSI-30**

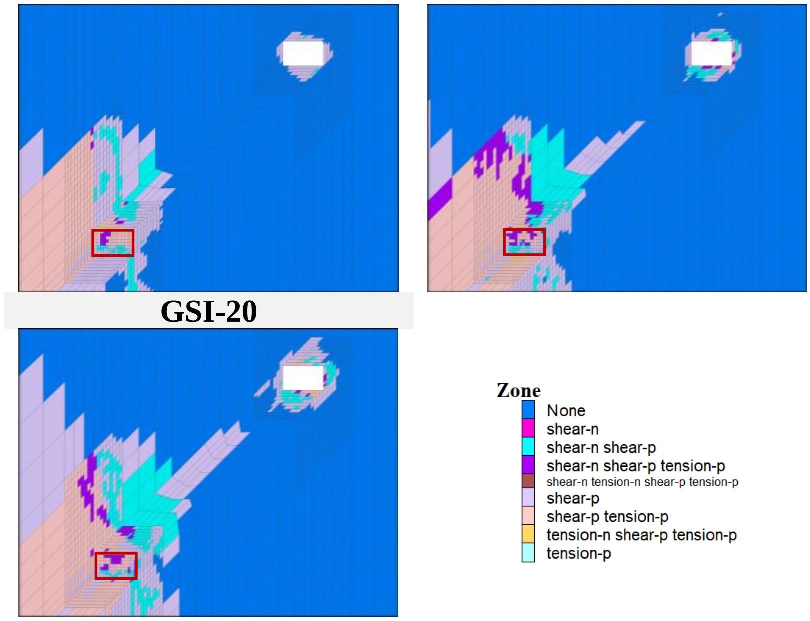


Figure 4.38. Fracture zones through the chain pillar

The analyzing results present that if the rock has deteriorated, and the chain pillar becomes more sensitive and reacts in a dangerous way. Rock mass deterioration firmly elevates the serious fracture zone propagation through the chain pillar, particularly the direction from gob area to the upper gateroad. This fracture is a shear failure type and is almost interconnected to the maingate of the upper gateroad when rock mass is deteriorated by 50% as GSI becomes 20. It is seen that 30m chain pillar is adequate to maintain the stability of upper gateroad. A wider chain pillar needs to be remained and should attempt to locate maingate of upper panel away from the high abutment stress profile. Or three- and four-entry systems can be proposed to provide safe mining operations regardless of some disadvantages such as low coal extraction rate, slow gateroad development rate and to increase the length of drivage.

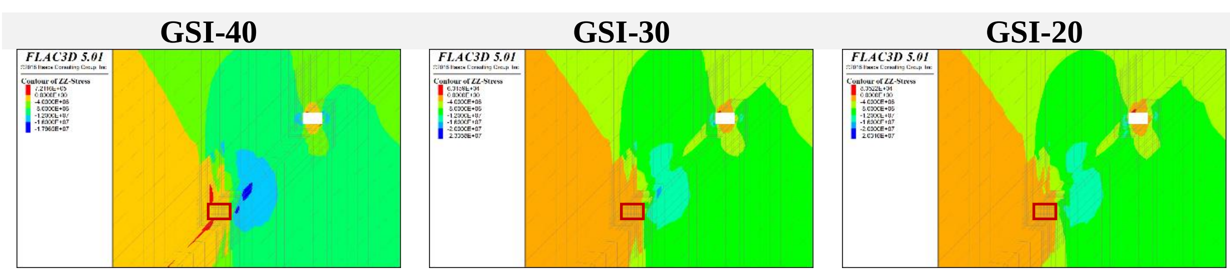


Figure 4.39. Induced stress distribution manner

Figure 4.39 presents that the upper gateroad could be located within a high abutment stress profile. Depending on the rock mass deterioration, the highest stress values are varied differently in each case. But it usually ranges in interval between 17.9MPa and 23.3MPa. Major stress concentration occurs in the upper corner of the lower gateroad, and magnitude is relatively consistent with different rock mass properties applied in the analysis. The contour of stress distribution is similar, but the magnitude is apparently varied on the panel excavation side.

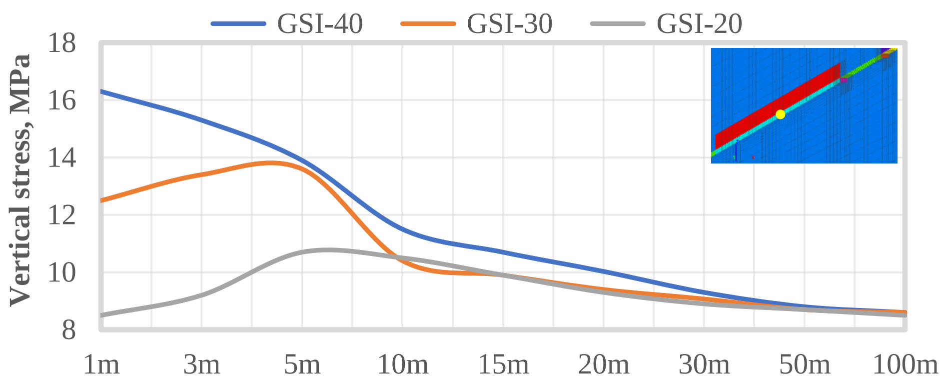


Figure 4.40. Stress distribution along the panel retreat direction

For the stress redistribution induced by the panel excavation, the strong rock mass results in higher stress concentration near the longwall face being excavated. The highest stress magnitudes are 16.5 MPa at 1m from the face with a GSI of 40, 13.6MPa and 10.7MPa at 5m from the face with a GSI of 30 and 20, respectively. It indicates that the higher stress magnitude distance away from the face is due to weak rock mass. Also, it can be realized that coal from the face in 5m has yielded already and not be able to sustain large loading acted on it. So, the face stability condition is dangerously unstable. Furthermore, this situation also seriously affects the chain pillar condition. As presented in Figure 4.41, stress data was collected from center of the chain pillar in different mining phases. After entire panel is entirely excavated, the stress of GSI 30 and 20 is similar and there is no substantial difference. However, the stress of GSI 40 is higher than that of two cases of deteriorated rock mass by certain percentages.

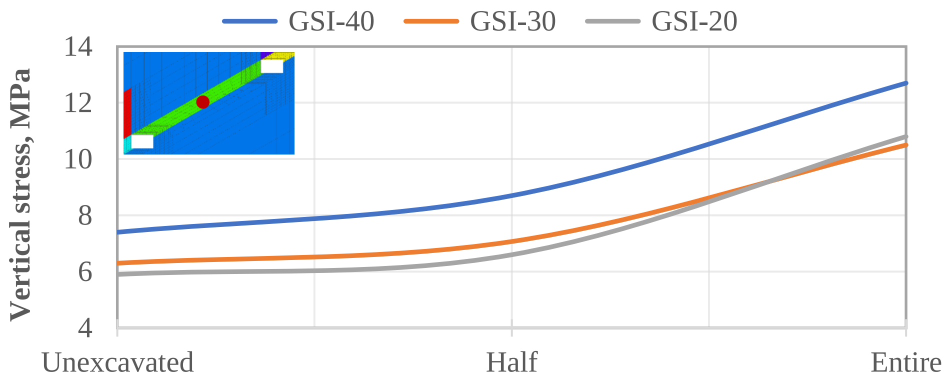


Figure 4.41. Stress on the center of pillar in the mining phases

4.7. The longwall face support characteristics in steep coal seam

The selection of face support is the key element for successful and safe longwall mining. There are generally three different faces supports for their mechanization. Hydraulic prop support is a common example of the manual support type. And rigid type standing supports can tolerate a small amount of displacement before being completely deformed, hydraulic and yielding supports should be considered in situations of intensive convergence. However, face support has rapidly been sophisticated from manual to fully mechanized and can be remotely controlled with computer-based automation. Since the mid-1990, many attempts have been made in order to apply fully-mechanized chock and shield support for steep coal seam longwall mining. For example, there is a number of steep coal seam longwall coal mines at a wide range of depths from hundreds to over thousand meters that fully mechanized shield support has been successfully and effectively utilized for a long time. In those cases, the average coal seam dip angle ranges from 53˚ to 60˚, and annual production reached 0.6Mt to 1.2Mt. Accident rates also have greatly reduced and safe working condition was realized, referred to in a research article by Y. Wu et al. Due to several technical and operational difficulties of steep coal seam longwall mining, heavyweight fully-mechanized support types are not appropriate. Hence, lightweight and easy-to-transfer hydraulic prop support is proposed in this study. Moreover, the high probability of support gliding, tilting and serious instability conditions in steep coal seam mining mainly occur due to the weight of the support itself and the weight of the collapsed roof rock on the support.

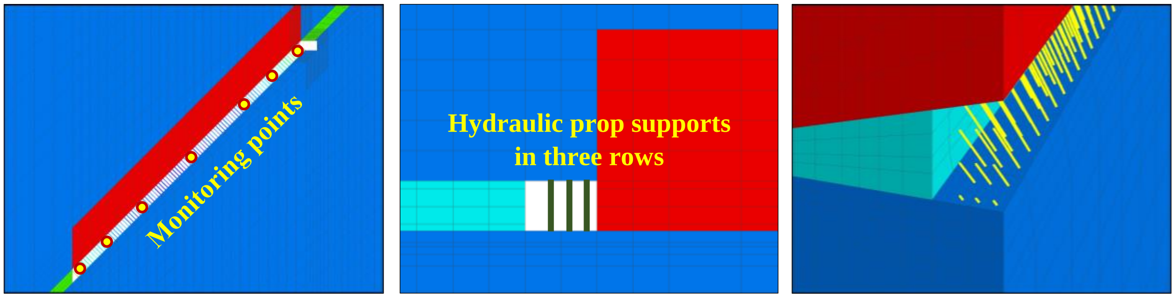


Figure 4.42. View of prop support in longwall face

The mechanical properties of steel arch support that is used as prop support in longwall face excavation are presented in Table 4.2. According to the yield strength of prop support, which is around 540.0MPa, if the beam axial stress exceeds this limit, then it can be assumed that prop support is no longer to sustain the load from collapsed roof rock and completely failed. The overall view of the prop support pattern in the longwall face is illustrated in Figure 4.42. The spacing of the prop support is 1m, and the spacing of the row is taken as 1m and 1.5m to investigate the stress reduction by sharing the loading on the prop support itself. The beam axial stress displacement parameters are selected as main characteristics, and various geotechnical and geological conditions have been applied in the analysis.

Table 4.2. The mechanical properties of prop support

№	Parameters	Unit	Value
1	Density	kg/m^3	7,800
2	Young's modulus	GPa	200
3	Poison's ratio	-	0.3
4	Cross-sectional area	cm^2	36.5
5	Yield strength	MPa	540
6	I_y	$\times 10^{-8}$m^4	732
7	I_z	$\times 10^{-8}$m^4	154
8	J	$\times 10^{-8}$m^4	22

The section does not aim to investigate the stability of the surrounding rock mass of longwall face excavation. The analyzing results will present how steep coal seam extraction affects the support system characteristics and distribution trend of axial stress along the seam dip in the face under various mining conditions. Seven measurement points are set with certain intervals along the seam dip shown in Figure 4.42. Seam dip angle in the numerical model was taken 45˚ as similar as the average seam dip angle in the NS coal mine. For the shallow mining depth, the axial stress curve shows fluently and symmetrical. Then, mining depth increases to 500m and 700m, the higher axial stresses occur near the upper and lower end of the face and it becomes not fluent. The highest axial stresses are 155.5MPa, 193.3MPa and 212.2MPa for the mining depths of 300m, 500m and 700m, respectively. It notes that stress magnitude increases by approximately 40MPa and 20MPa by depth increasing with 200m. The large gap between the high axial stresses was occurred when mining depth converts from shallow to moderately deep. Moreover, the beam compressive z-displacement are 5.45mm, 8.09mm and 10.9mm at each depth while tension z-displacement are 11.2mm, 17.6mm and 24.06mm. The z-displacement on the floor is around 2.2 times higher than that of the roof. Floor condition has been worsened substantially.

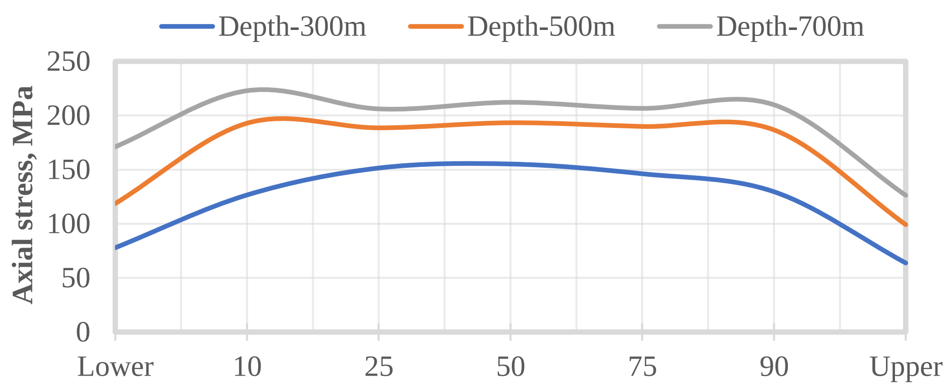


Figure 4.43. The relationship between the beam axial stress and depth

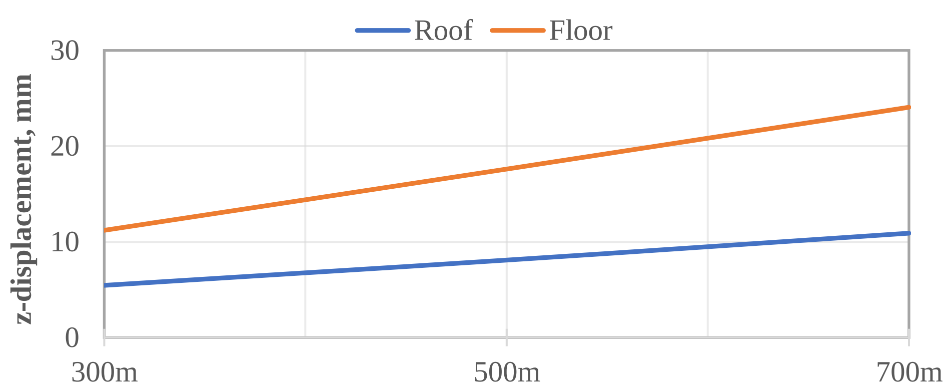


Figure 4.44. The relationship between the z-displacement and depth

Circumstances can be seen as similar to the author's expectation associated with the high lateral stress conditions. The load is apparently increased with the k ratio increasing to 2.0. By increasing the lateral stress ratio from 0.5 to 2.0, the location of the high axial stress moves gradually from the center of the face toward two ends. In each case, the high axial stresses have been obtained as 122.6MPa, 155.2MPa, 171.3MPa and 174.3MPa. There is no significant difference between the k ratio of 1.5 and 2.0 as well as this type of prop support can still sustain load from the roof rock. Besides that, the beam compressive z-displacement are 8.5mm, 5.45mm, 4.91mm and 3.27mm while tension z-displacement are 13.9mm, 11.2mm, 8.7mm and 7.01mm with stress ratio gradually decreases. However, the floor condition is still worse than the roof. The proper countermeasures and additional solutions should be considered to maintain sudden floor heave that affects face support stability seriously.

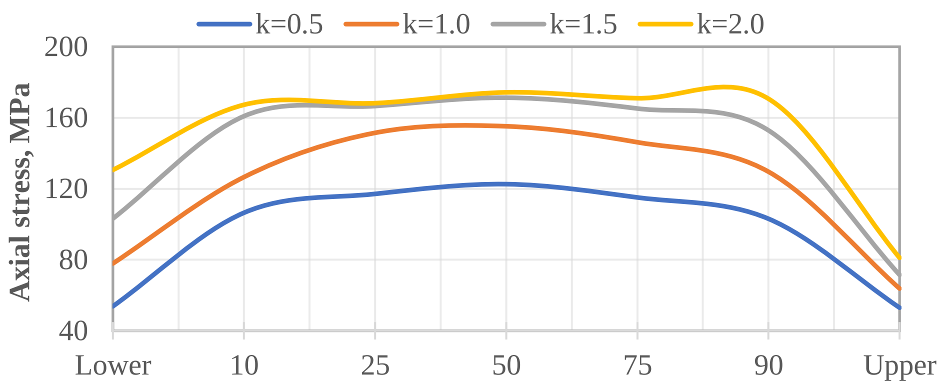


Figure 4.45. The relationship between the beam axial stress and lateral stress ratio

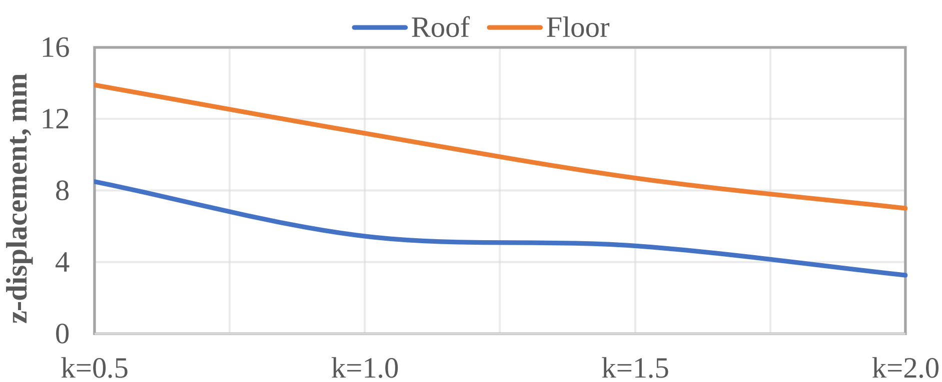


Figure 4.46. The relationship between the z-displacement and lateral stress ratio

Regardless of the mining depth and lateral stress ratio influences, the effect of rock mass deterioration on the beam axial stress and z-displacement distribution was investigated. The circumstance that is relatively similar to previous results was the floor condition became drastically worsened when the rock mass deteriorated by 25% and 50%. The highest beam axial stresses are 155.2MPa, 214.2MPa and 271.7MPa in the cases. Therefore, adequate and effective countermeasures that can improve floor conditions must be considered further. For example, a cable bolt is a widely used countermeasure against large floor heave and buckling in underground coal mining practices. However, installing the cable bolts or applying any countermeasures somehow affects the face advance rate, so considering strong floor conditions, the appropriate measures against floor heave that can be low-cost and have a small influence on the mining rate should be considered. Besides, bearing capacity of roof and floor rock has to be calculated. In order to carry out bearing capacity assessment for the roof and floor rock, a field investigation must be conducted and selection of the base plate shape and dimension plays key role as principal factors. Unfortunately, it was almost impossible to do field measurement and investigation in the NS coal mine due to numerous unexpected issues. The phenomenon of beam axial stress distribution with deteriorated rock is specific. A large amount of axial stress was observed near the lower end of the face. Value was 3.4 times more than that of the upper end, and the massive increase begins at a 10m distance from the lower end in the case GSI of 20. A similar phenomenon also occured in the case GSI of 30. Besides that, the beam compressive z-displacement are 5.45mm, 19.5mm and 26.5mm while tension z-displacement are 11.2mm, 61.3mm and 81.5mm.

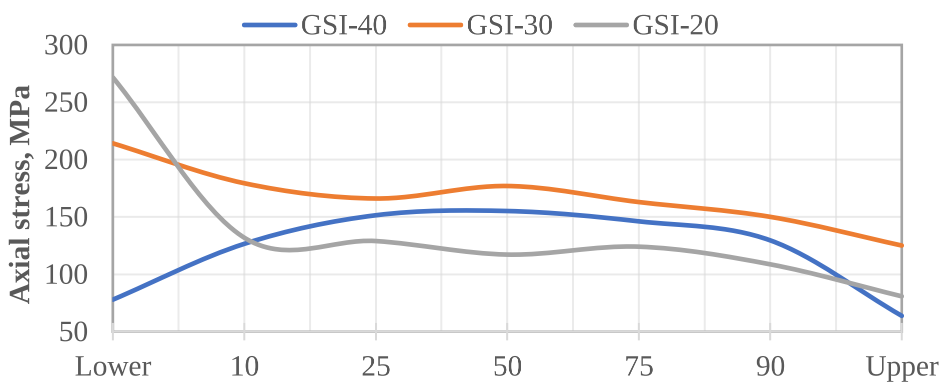


Figure 4.47. The relationship between the beam axial stress and deteriorated rocks

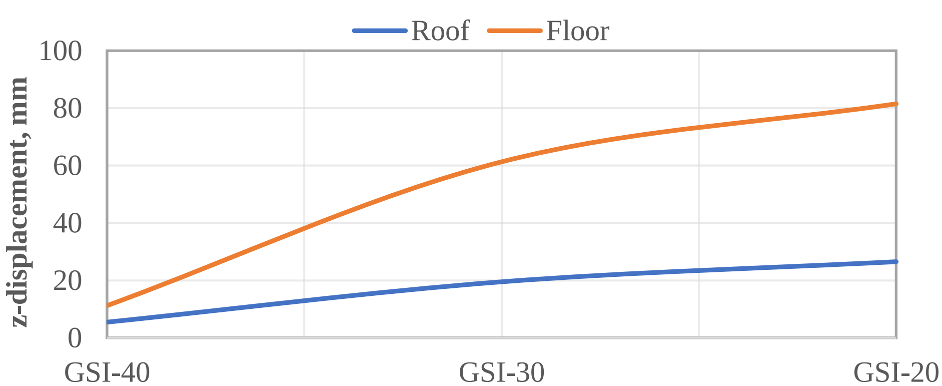


Figure 4.48. The relationship between the z-displacement and deteriorated rocks

The effects of several mining conditions with prop support spacing of 1m and row spacing of 1m have been studied, and all results were discussed. This section briefly emphasizes how the wider row spacing of prop support influences the beam axial stress and z-displacement in the face. It is obvious that wider row spacing of prop support can cause of dangerous roof falls, large deformation and loading. In order to investigate those circumstances, row spacing of prop support distanced from 1m to 1.5m in this case. The highest beam axial stress is increased from 155.2MPa to 191.2MPa, by around 23%, with row spacing of 1.5m. But, compared to the yielding strength of prop support, it still can sustain the large stress and loading, especially in the center of the face exaction. Besides that, the compressive z-displacement also increases from 5.45mm to 7.53mm although difference is not so large it has potential adverse impact on the support efficiency. The tension z-displacements are 11.2mm and 12.9mm correspondingly.

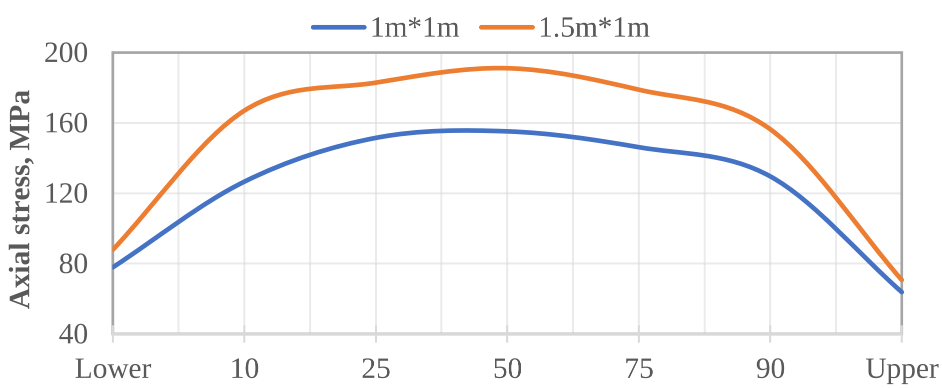


Figure 4.49. Axial stress distribution with different row spacing

Considering the technical issues related to the face support stability, including gliding, tilting and toppling, as well as specific features of steep coal seam extraction with applying longwall coal mining without backfilling technology, hydraulic prop support is one of the suitable options operationally and economically. Nevertheless, disadvantages and operational ineffectiveness can cause the panel slow excavation rate, small output, lack of automatization and safety problems. It also still has negative impacts on the decision-making process in terms of the appropriate mining design.

4.8. Summary

This chapter discussed the effects of various geotechnical and geological conditions and face support characteristics on steep coal seam longwall mining while the panel is extracted partially. Some essential mining conditions, coal seam dips, mining depths, lateral stress ratios and rock mass deterioration by reducing the GSI value have been applied to investigate the ground respond associated with rock deformation, fracture zone and vertical stress distribution and magnitude. Despite the specific caving mechanisms, panel excavation and gob modeling steps were assumed to be the same as conventional horizontal longwall mining. Firstly, seam dips do not influence the stability of the gateroad, while the effects of various mining depths and lateral stress ratios are significant. Moreover, due to the mechanical properties of intact rock mass, which are obtained from laboratory experiments, rock behavior should be made similar to actual conditions. Thus, rock mass deteriorated by certain percentages, and it resulted in larger deformation and fracture zone propagation around the underground excavation areas. it can be realized that in the case of a rock mass deterioration rate of 50%, the wider chain pillar needs to remain in order to maintain gateroad stability. In addition, although the lateral stress ratio effect usually occurs in shallow mines, its influence on ground behavior is considerable. In terms of the face support system, manual prop support has been proposed. The typical phenomenon of beam axial stress and z-displacement has occurred. The higher axial stress is usually concentrated around the middle of the longwall face when rock mass is competent and immediate roof rock is fair to strong. However, after the rock mass deteriorated, a substantial increase occurred on the bottom of the face. The beam axial stress and z-displacement were drastically increased, and it can cause of big failure and serious instability of the longwall face excavation. Even though longwall mining can be one of the applicable mining methods for steep coal seam, it has its own technical limitations related to the mining conditions, such as seam dip angle. It can be recommended that based on a series of numerical analysis results, a coal seam dip angle of 45˚ is feasible for applying longwall mining. Otherwise, considering the facts of several successful fully-mechanized longwall mining practices for steep coal seams that have employed shield supports in face, a seam dip of 60˚ is feasible from a geotechnical point of view.

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CHAPTER FIVE. **THE EFFECTS OF ORIENTATION OF MAJOR AND INTERMEDIATE HORIZONTAL STRESS ON THE MINING DESIGN**

5.1. Background

Chapter 3 considered the various geological and geotechnical conditions and their effects on the stability problems, especially how those situations can significantly increase displacement and fracture zone propagation around the gateroad. Those studies have been carried out using numerical simulations as parametric and qualitative assessments from a geotechnical point of view. In this chapter, the factor that can tackle serious safety problems and reduce mining operational costs related to support systems and roadway maintenance will be discussed and studied through a series of numerical analyses. Numerous historical studies and researchers have already recommended that the stability of roadways can be modified better by adjusting the orientation of the gateroad and high horizontal stress. For example, *McCulloch et al. (1975)* highlighted how roof fall issues were diminished substantially by adjusting the orientation between face cleats and roadways to 45˚. On the other hand, *Singh (1990)* has made a similar recommendation experienced in Indian coal mines and while *Rozier (1986)* suggested the development of longwall faces and gateroads at 60˚ and 30˚, respectively, to the face cleat in western Canada. *Galvin (2016)* preferred a layout of -30/30/60 for the gateroad, cut-throughs and the installation roadway/face so that horizontal stress impacts on the roadways are all reduced but not eliminated. Any layout of an appropriate angle cannot eliminate all negative aspects.

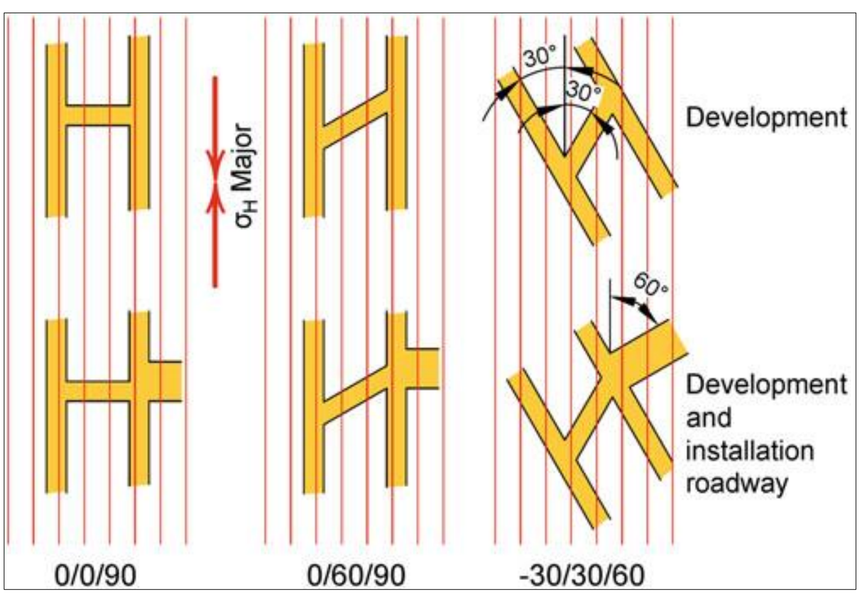


Figure 5.1. Proposed layout of gateroad by *Galvin (2016)*

5.2. Major horizontal stress directions

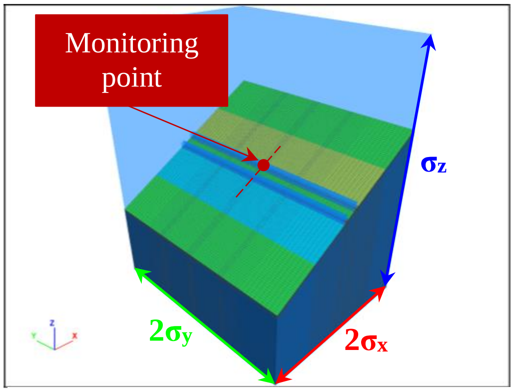


Figure 5.2. Initial condition of high horizontal stresses

The same model, variable and non-variable parameters used in Chapter 3 were utilized in this chapter. In order to investigate the correlation between the orientation of the gateroad and major horizontal stress, the horizontal stresses in the x and y axes will be applied in turn. Once the vertical stress and horizontal stress along the x-axis are equal, the horizontal stress along the y-axis will be doubled ($2\sigma_y>\sigma_x=\sigma_z$). Vice versa, the vertical stress and horizontal stress along the y-axis are equal, and the horizontal stress along the x-axis will be doubled ($2\sigma_x>\sigma_y=\sigma$). In these cases, the effect of the major horizontal stress direction will be investigated by measuring the deformation values and assessing fracture zone propagations around the gateroad. The in-situ vertical stress is calculated by the following equation:

$$\sigma_z = g\rho z \qquad (4.1)$$

where; σ_z- is vertical stress (MPa), g – a constant value of gravitational acceleration, ρ – rock mass density (kg/m^3), and z – mining depth (m). As recommended in some mining practices, if the orientation of the gateroad and high horizontal stress is set perpendicular or crossed with large angles, a high risk of the fracturing process occurs apparently. It notes that it maximizes the adverse impacts of high horizontal stress on the gateroad stability.in contrast, it minimizes the risk of larger aerial fracturing around the gateroad, especially on the roof and floor when the gateroad is orientated along the major horizontal stress. Besides these considerations, many factors must be considered in designing gateroad direction and layout, for example, coal quality consistency, coal thickness consistency, dip and dip direction, joint and cleat direction, and horizontal stress magnitude. In terms of longwall panel design, along the strike direction was proposed due to certain advantages compared to up-dip and down-dip directions. *Galvin (2016)* suggested the layout of the gateroad is 0/60/90 layout in which the gateroad continues to be orientated in the optimum direction and the installation roadway/face in the most adverse direction. Nevertheless, cut-throughs are orientated at 60˚ to the horizontal stress field to mitigate severe impacts on them. On the other hand, there is a higher likelihood that one mining direction will not be optimum for controlling the impact of cleating and jointing, thereby elevating the risk of injury and causing an increase in mining spans due to rib spall. Consequently, the rib spall expansion can cause substantial rock fall and roof rock to be loosened. The displacement and fracture zone around the gateroad were used to explain these phenomena combined with mining conditions.

5.3. The effect of coal seam dip

The correlations between the orientation of the gateroad and high horizontal stress are presented based on the analyzing results by showing the roof displacement and fracture zone, as illustrated in Figures 5.3 and 5.4. For the fracture illustration, the mining depth and seam dip angle are 300m and 45˚, respectively.

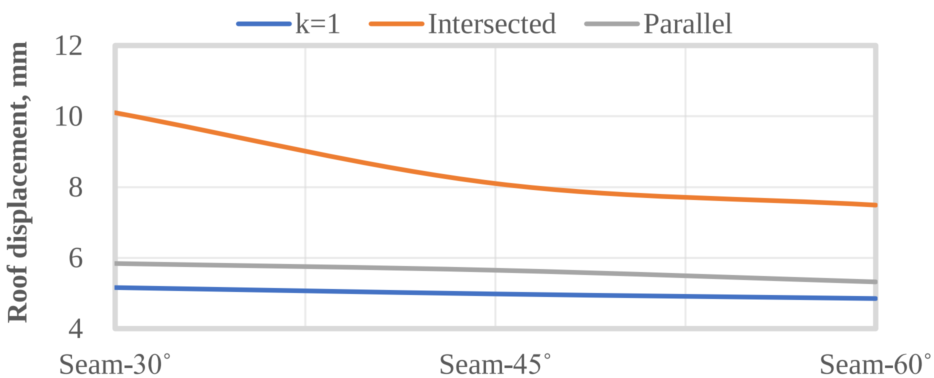


Figure 5.3. The relationship between the displacement and seam dip

The change of roof displacement in cases of stress ratio 1 and parallel to σ_H is relatively similar. Although various geotechnical and geological conditions were applied in the simulation, there is no substantial increase or decrease in deformation values, and it is steady. On the other hand, when the orientation of the gateroad and high horizontal stress is crossed by 90˚, and the coal seam becomes steeper as the dip increases up to 60˚, the roof displacement decreases significantly. In terms of displacement in the gateroad, it can be noted that this circumstance can be occurred because of the large thickness of coal that has remained on the roof due to the seam dip becoming steeper. Moreover, high horizontal stress parallel to the gateroad results in a larger displacement on the roof.

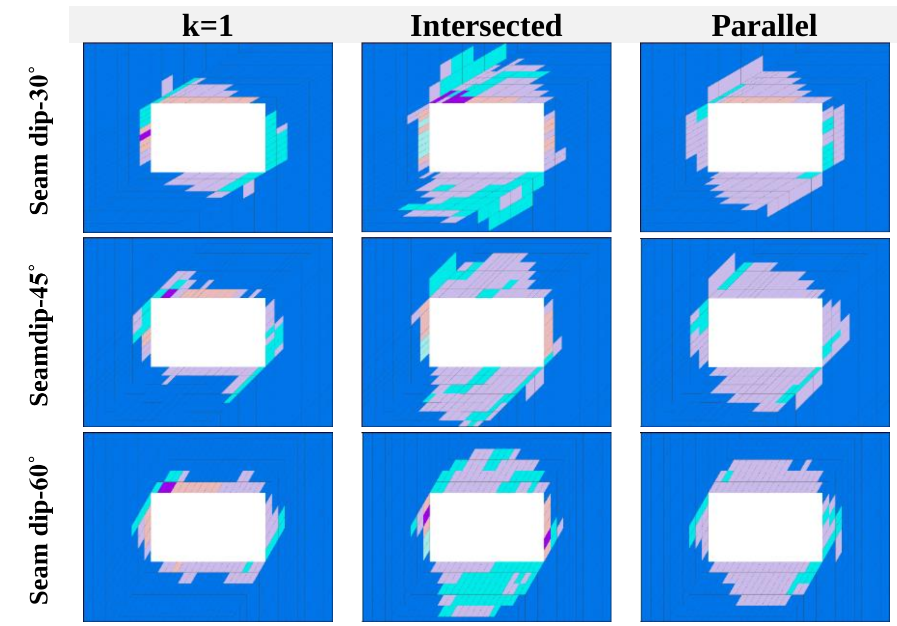


Figure 5.4 The relationship between the fracture zone and seam dip

The analyzing results of the fracture zone show that the gateroad is stable when the stress ratio is 1. After the horizontal stress parallel to the gateroad is doubled, the fractures are propagated in all directions, especially rather vertically than horizontally. Fractures on the rib sides are slightly increased compared to the case of stress ratio 1. When horizontal stress intersected to gateroad is increased, drastic changes of fracture propagation, particularly on the roof and floor, occurred, as shown in Figure 5.4. The roof and floor conditions are considerably worsened, which may lead to serious instability for the gateroad. Again, the adverse impacts of high horizontal stress are significant. As shown in Figure 5.5, the higher displacements occurred in the case of seam dip 30˚ with a stress ratio of 1 and major horizontal stress intersected with the gateroad. The highest displacement value is 11.2mm. For the other two cases of seam dip 45˚ and 60˚, the higher horizontal stress influences similarly, and the highest displacements are 8.71mm and 7.92mm, respectively. Effective practices in order to maintain the roof stability of the roadway, leaving a certain thickness of coal layer on the roof, are very common in underground coal mines. The difference between the highest displacement values is around 42%. The thickness of coal remained on the roof depends upon the coal seam dip angle that when seam dip become steeper, remaining coal thickness on the roof increases.

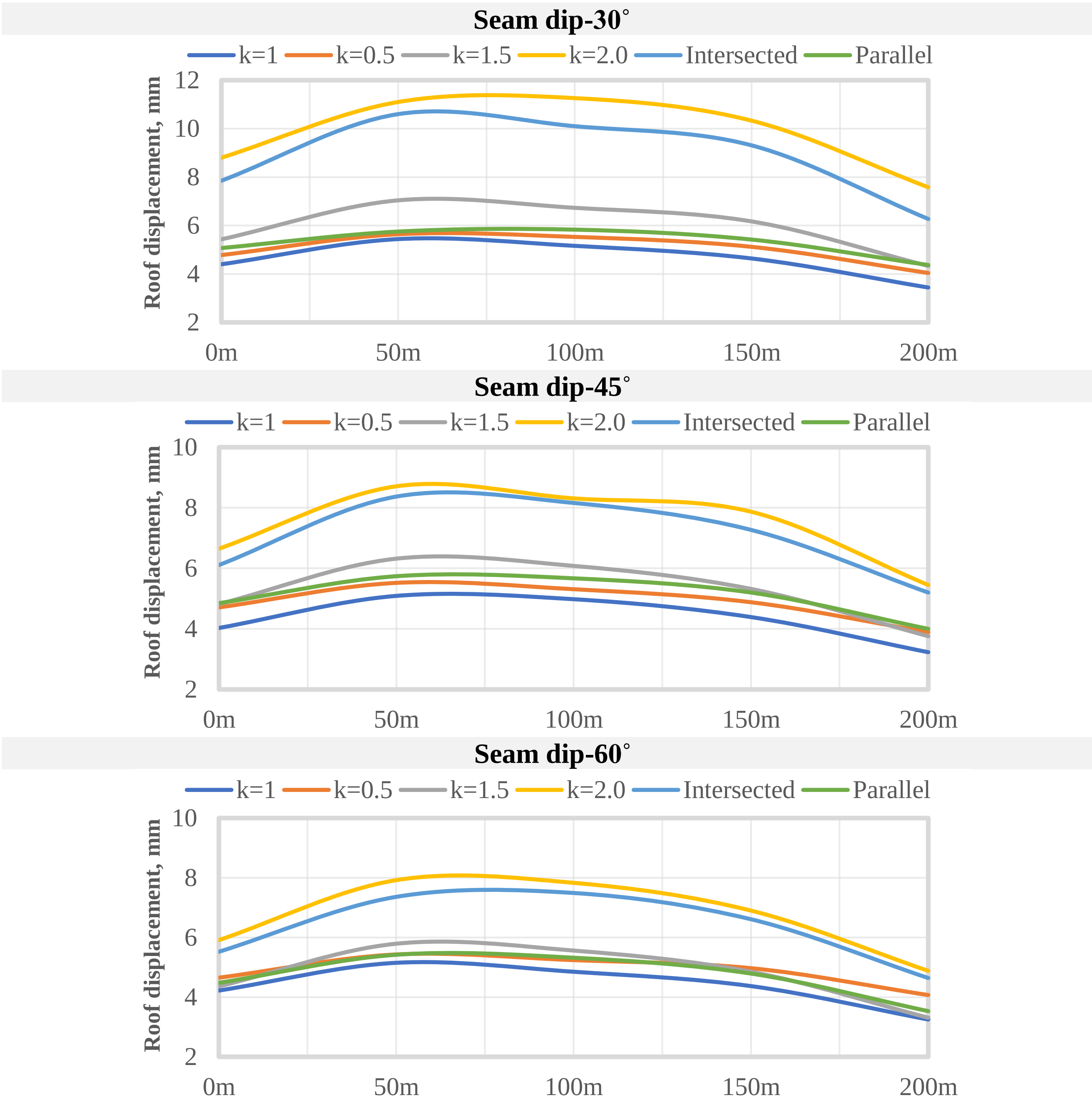


Figure 5.5. Displacement along the gateroad with various stress conditions

5.3.1. The induced stress characteristic

By applying three different schemes of in-situ stress conditions in the numerical model, the induced stress redistribution along the longwall panel and chain pillar has been investigated, as illustrated in Figure 5.6. The stress distribution manner, in which the k ratio of 1 and major horizontal stress is parallel to the gateroad and the panel is entirely excavated, is relatively similar and high stress usually concentrates on the rib side of the lower gateroad. When the major horizontal stress is intersected with the gateroad, the higher stress concentration area has moved up toward the roof.

The detailed induced stress distribution around the upper gateroad will be illustrated in Figure 5.8. The average vertical stress magnitude in the cases of k ratio of 1 and parallel major horizontal stress was substantially increased by approximately 4.0MPa which was from 17.6MPa to 21.8MPa.

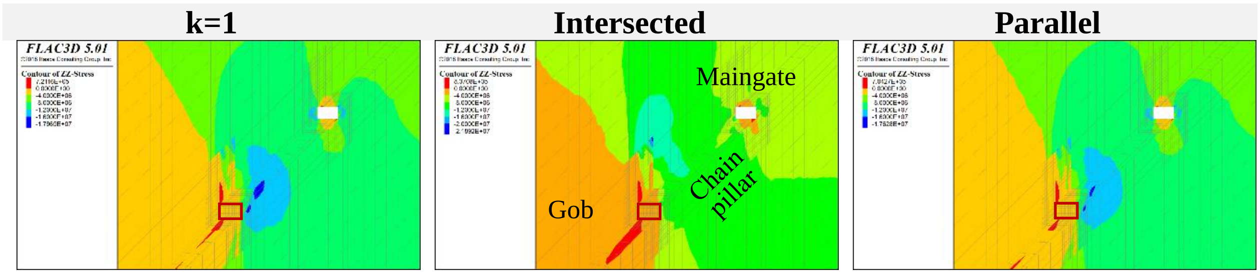


Figure 5.6. Induced stress distribution manner

In terms of the fracture zone propagation through the chain pillar after panel excavation, the analyzing results of the k ratio of 1 and parallel major horizontal stress cases are even similar, and the pillar is significantly stable and safe. The major horizontal stress is intersected with the gateroad and panel direction, the shear fracture has been propagated from the lower end of the pillar to the upper end in the large area and vice versa in direction. In addition, shear fracture mainly propagates in the rock mass rather than coal seam along the chain pillar.

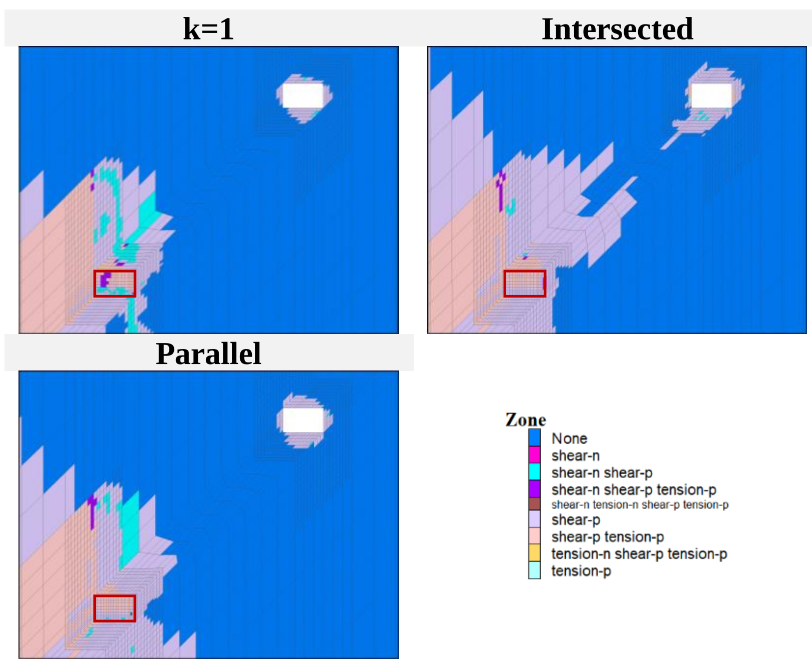


Figure 5.7. Fracture zones through the chain pillar

Based on this circumstance, there could be two suitable recommendations for maintaining the gateroad stability. First option is that larger chain pillar needs to be remained to locate away the gateroad from high abutment stress profile and prevent shear fracture interconnecting through the pillar. By doing this, coal recovery will be reduced significantly and it influences the mining profits and economic factors. Second option is that to reduce the angle between major horizontal stress and underground openings including gateroad, face/installation roadway and cut-throughs. For the steep coal seam longwall coal mining, intersecting angle between the orientation of gateroad and high horizontal stress can be reduced by changing the panel retreat direction. In this study, in order to provide adequate stability for the gateroad, "along the strike" panel retreat direction was proposed. However, it could be in-between direction that is more similar with combination of "along the strike" and "down the seam dip" or "up the seam dip" directions to substantially mitigate the technical difficulties from geotechnical point of view. As well as this approach has high potentiality to applied in the NS coal mine because as presented in Chapter 2, there is a considerable probability that the major horizontal stress intersects the coal seam strike by nearly 90˚.

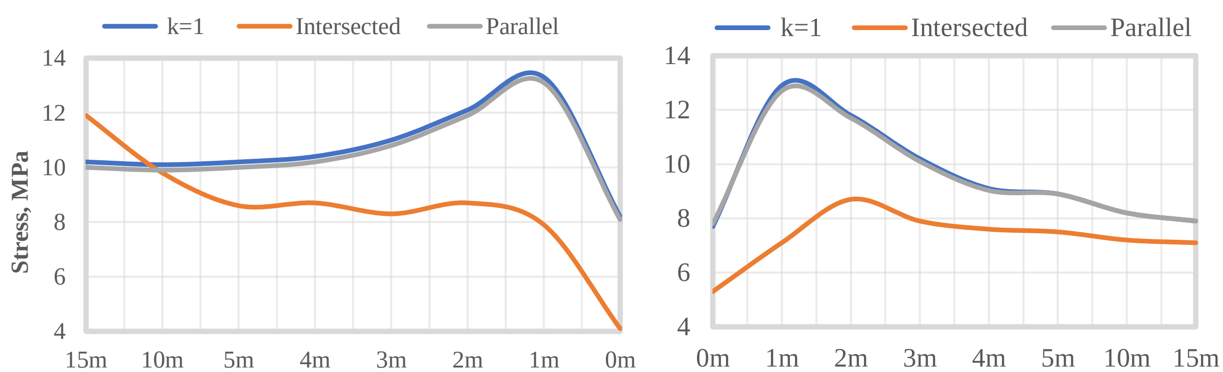


Figure 5.8. Stress distribution on the ribs of upper gateroad

For the induced stress distribution, three different stress magnitude has been investigated as shown in Figures 5.8, 5.9 and 5.10. The circumstances of the k ratio of 1 and parallel major horizontal stress cases are strictly same on the gateroad rib sides. Therefore, it can be seen that those two similar stress conditions can be ignored in further studies. But, when the high horizontal stress intersects with gateroad, stress around the gateroad ribs is relatively small and it increases dramatically toward gob area. The situation is related to that location of high stress concentration is changed near the lower end of chain pillar.

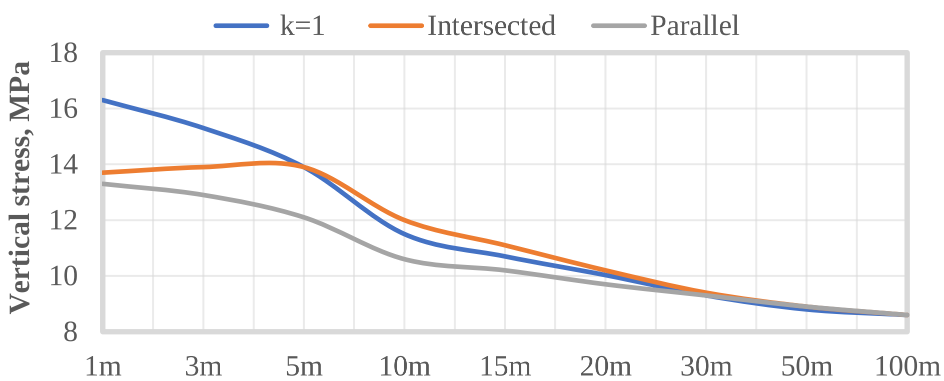


Figure 5.9. Stress distribution along the panel retreat direction

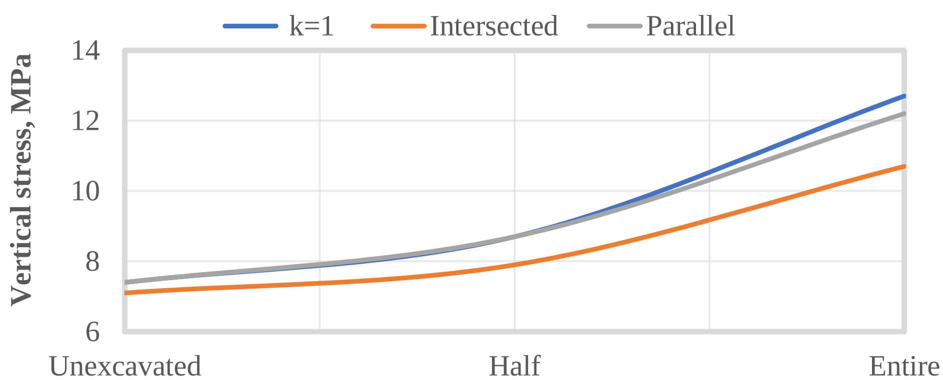


Figure 5.10. Stress on the center of pillar in the mining phases

During the face excavation, the vertical stress mainly concentrates 5m into the panel and gradually decreases. The highest stress value was 16.3MPa when the k ratio equals 1. According to induced stress magnitude along the face retreat direction, it can be noted that despite high horizontal directions intersecting and parallel to panel retreat direction, stress magnitude plays a more important role than orientation. It is answered by seeing the applied condition of k ratio of 1 in which horizontal stresses in two axes are doubled more than vertical stress. The relationship between the vertical stress magnitude and each mining phase in Figure 5.10 is quite obvious. The largest variance of stress magnitude between the mining phases and different stress conditions are 5.3MPa and 2MPa, respectively.

5.4. The effect of mining depth

There are dramatic changes in the increase of roof displacement corresponding to mining depth increases up to 700m. This phenomenon shows the correlation that the probability of high displacement occurs when the major horizontal stress intersects with the gateroad and mining depths increase. It can be expressed in numbers as the displacement value in case the major horizontal stress intersecting to the gateroad is raised by around four times, increasing from 8.1mm to 35.1mm. Also, a small variance between the other two stress condition cases is observed and shown in Figure 5.11.

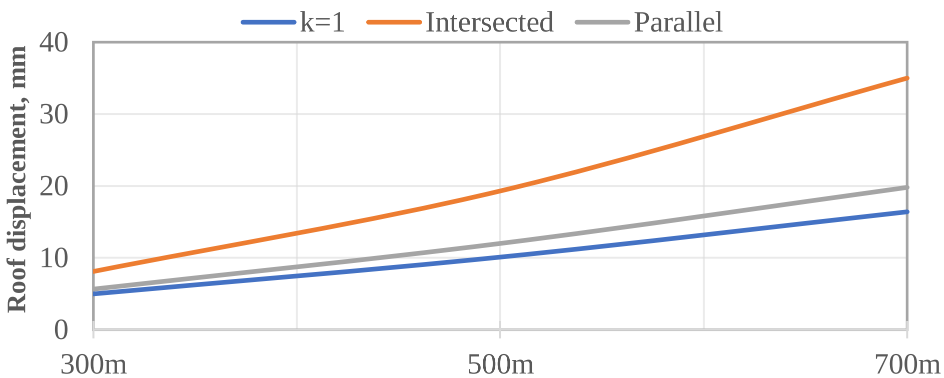


Figure 5.11. The relationship between the displacement and depth

For the fracture zone development, greater mining depths elevate the rock loosening, raising the risk of rock falls and massive floor heave. Changes in the fracture zone are shown clearly in Figure 5.12. Furthermore, the major horizontal stress intersecting with the gateroad mainly elevates the fracture intensity towards the roof and floor. Major horizontal stress parallel to the gateroad results in more rib side mitigation than other cases. The substantial coal spalls and bumps on the rib sides can further cause the roof rock instability due to serious horizontal stress.

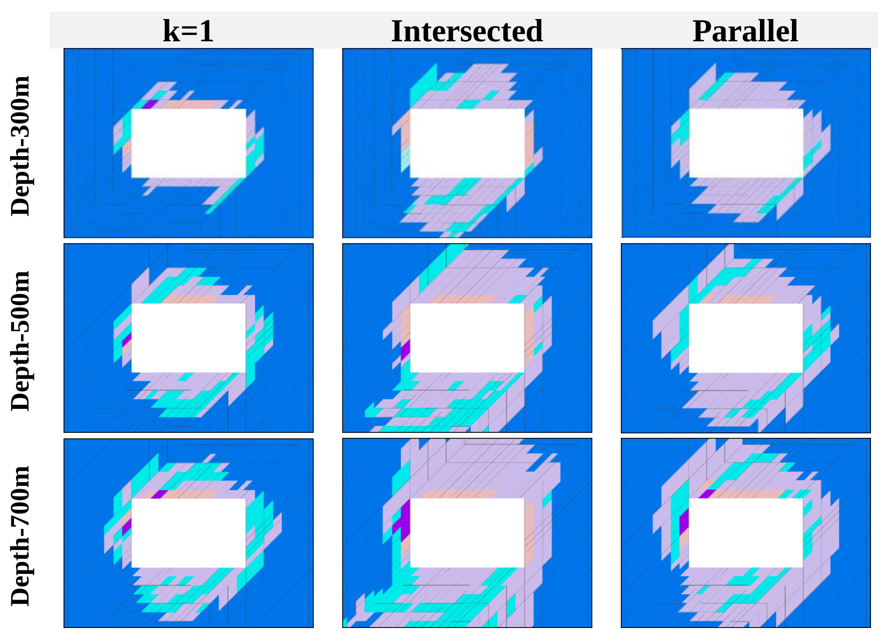


Figure 5.12. The relationship between the fracture zone and depth

Regarding the chain pillar conditions, mining depths have significant adverse and direct impacts on underground mining stability. As shown in Figure 5.13, the size of fracture zone propagation in the case of high horizontal stress intersecting with gateroad through the chain pillar is much larger than that of the other two cases. It can be summarized that the chain pillar of 30m width intersects with high horizontal stress at greater depths is inadequate, and a larger dimension should be considered. Besides that, it is seen that the pillar entirely failed in the case of 700m mining depth.

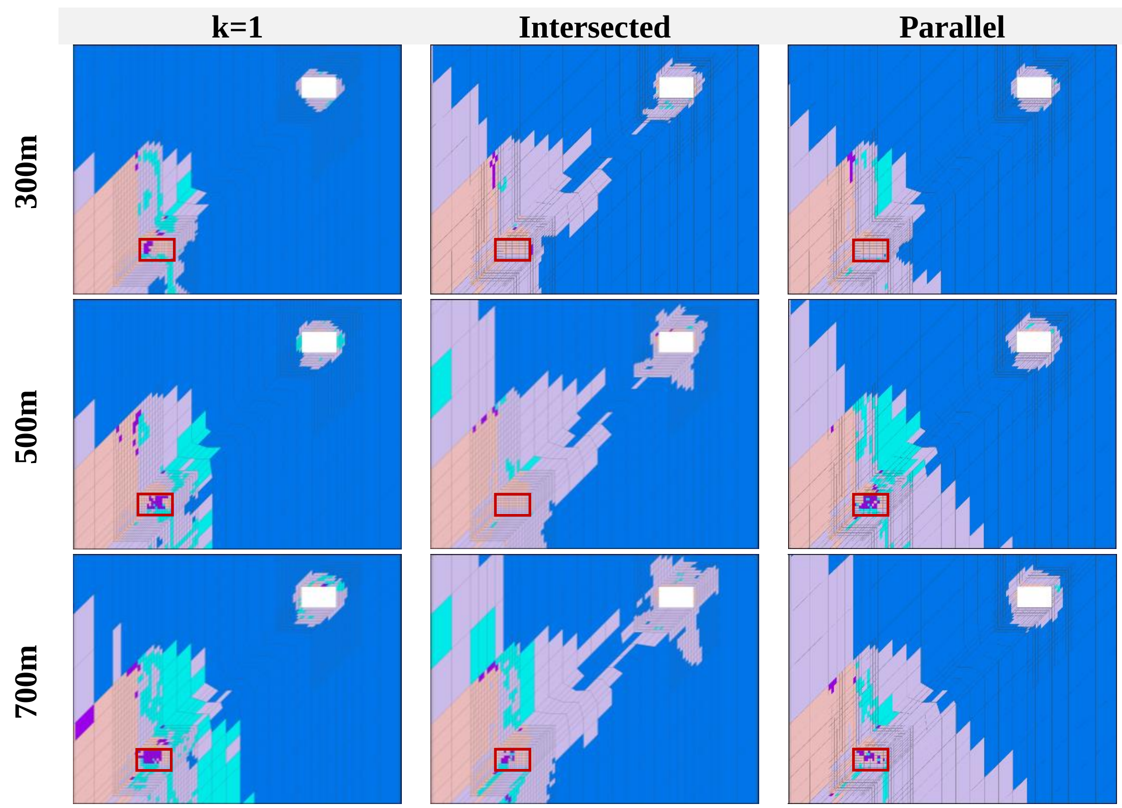


Figure 5.13. Fracture zones through the chain pillar

5.5. The effect of intermediate horizontal stress

Two cases of intermediate horizontal stress magnitude and direction are assumed to investigate the influences on the gateroad stability and ground behavior response. Comparing 5 scenarios for the applied stress conditions in the numerical analysis, the displacement and fracture zone development are used to explain. Figure 5.14 shows that the major horizontal stress still plays the main role to affect the deformation of surrounding rock in the gateroad as well as in three different mining phases. The displacement values of the cases that k ratio of 1 and major horizontal stress parallel to the gateroad were relatively similar.

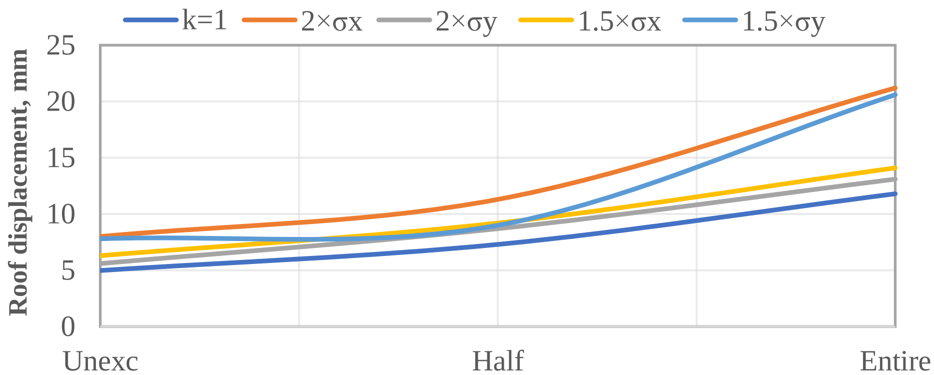


Figure 5.14. The relationship between the displacement and various stress conditions

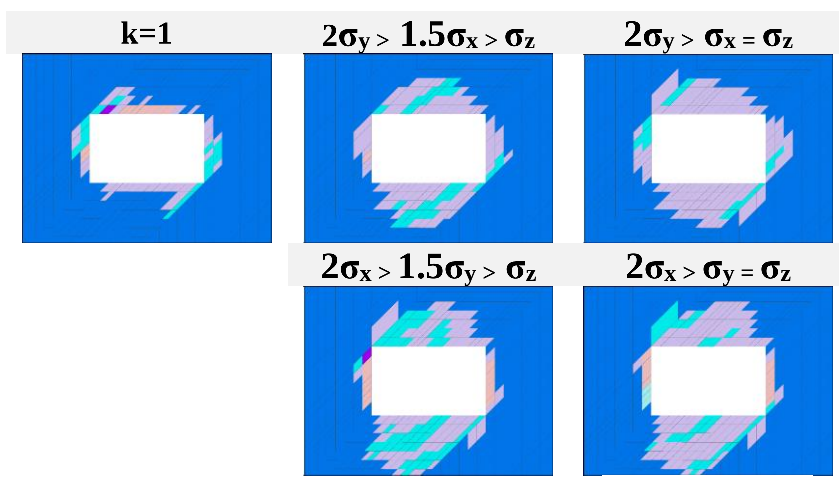
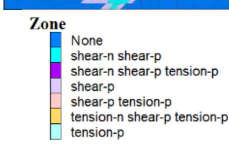


Figure 5.15. The relationship between the fracture zone and intermediate stress



However, in cases of the major horizontal stress intersecting with the gateroad the displacement values have been dramatically increased with entire panel excavation. For the fracture zone propagation around the gateroad, there is no large difference even though intermediate stress is applied in two cases. Based on these analyzing results, intermediate stress cannot affect significantly the fracture development around the underground openings as shown in Figure 5.15. Therefore, the effect of the panel excavation is assumed to clarify the circumstances. After entire panel is extracted, fracture zone propagation through the chain pillar was not varied and still remain in same manner except only few types of shear fracture were changed.

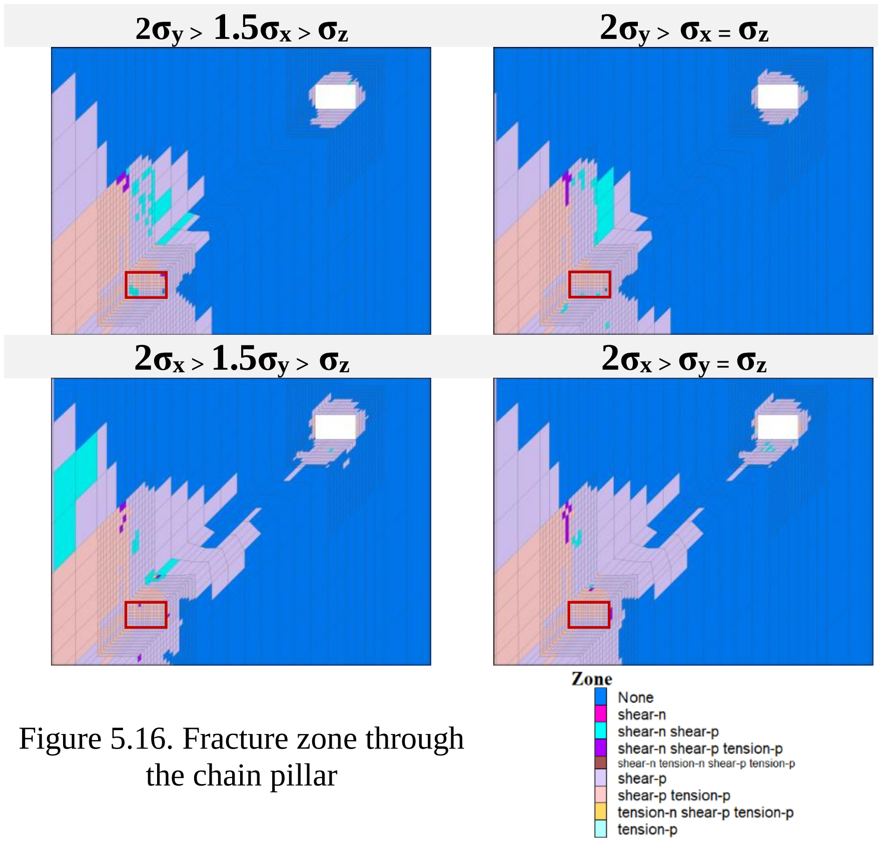


Figure 5.16. Fracture zone through the chain pillar

According to the numerical results, it can be summarized that the intermediate horizontal stress does not affect the deformation and fracture zone development substantially. The high horizontal stress still plays the main role to worsen the ground condition and the stability of openings.

5.6. Summary

Designing an adequate gateroad layout that involves the orientations and high horizontal stress direction can be a key factor in improving safety issues associated with ground control and stability, as well as enhancing the economic diversity of underground coal mining projects, especially in terms of longwall mining. Therefore, the interaction between the orientation of the gateroad and major/intermediate horizontal stresses has been investigated to understand the negative and positive impacts on stability. It is realized that the high horizontal stress intersecting with the gateroad direction has significantly large adverse impacts on the surrounding rock compared to other cases, which are k ratio of 1 and parallel horizontal stress direction. It is critical to find the optimal angles that can less influence deformation and fracture zone propagation in the gateroad. In the case of that, high horizontal stress intersecting with gateroad dominantly results in serious deformation and large fracture zone propagation in the surrounding rock mass, particularly on the roof and floor. Besides, parallel high horizontal stress has slightly more affected the larger fracture on the rib sides. The effects of mining depth are relatively obvious, and the proper dimension of chain pillar in greater deep mining should be considered when gateroad is designed to intersect high horizontal stress direction at nearly 90˚. Also, the high horizontal stress direction and magnitude still play the main role in the ground stability even though various intermediate stress conditions were applied in the numerical simulation. It notes that regional and local stress measurement and investigations need to be conducted at each mining case. Nature is diverse, and geology is complex. In addition, for the NS coal mine, it can be appropriate that the longwall face direction can be in-between "along the strike" and "down the seam dip" or "up the seam dip". By applying this design of face direction, numerous advantages will be followed up with. Although the gateroad becomes inclined, the inclination of the longwall face can be reduced and provide an opportunity to mitigate the adverse impacts in terms of the orientation of the gateroad and high horizontal stress.

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CHAPTER SIX. **CONCLUSIONS**

The development of steeply dipping underground coal mining is a necessary task for the coal mining sector in Mongolia nowadays. Coal export has been an essential source of international trade for the last couple of decades. Some ongoing coal mining projects have been run in southern Mongolia near the border between Mongolia and China. Due to the poor infrastructure development in that region, particularly high transportation costs from the mine to the border, a number of open-pit mines could be transferred to underground mining despite specific unfavorable geological and geotechnical conditions. Considering those reasons mentioned and not, the Narynsukhait coal mine and its specific geological features are very interesting and challenging for the mining engineers. This mine has several coal seams that seam dip and thickness range from 8˚-80˚, the average is around 43˚, and 1.7m-60.2m, the average is around 30.7m, respectively and the second biggest open-pit coal mine that annual coal production is over 12Mt. Therefore, by duscussing the research results, the potentiality of steeply dipping longwall coal mining was studied based on the a series of numerical simulation and field data. The each chapter of thesis has been concluded as follows:

Chapter 2. Although any research on in-situ stress investigation and measurement has not been conducted yet in Mongolia, the essential trend of the primitive stress regimes can be determined by comparing several engineering studies that have been carried out to investigate in-situ field stress regimes in underground coal mines at a wide range of depths ranging from hundreds to thousand meters in several provinces in China and Inner Mongolia. Thus, different in-situ conditions and lateral stress ratios corresponding with several mining depths had been considered and employed in the numerical analyses. Since rock masses have inherent characteristics of the different field conditions, rock masses in the NS coal mine can be classified accurately with qualitative and quantitative assessment schemes, as RMR and GSI values were calculated at 45 and 40, respectively.

Chapter 3. Ahead of starting the longwall panel excavation for steeply dipping coal seam, the specific features for the longwall mining design and layout of the roadway, gateroad, longwall face/panel and pillars have to be designated completely. In this chapter, three common practices of longwall face directions are introduced with respect to the ground characteristics and technological capabilities. However, considering the noticeable advantages and fewer technical difficulties, the longwall face direction of "along the strike" can be preferred compared to the other two options. But, in this case, the potential difficulties associated with face excavation and its support stability can be raised. The main characteristics of this issue will be discussed in the next chapter. The effect of seam dip angle on the stability of gateroad especially changes in displacement and fracture zone propagation, is negligible, while the relationship between the mining depth and ground behavior of surrounding rock mass in gateroad is strong. When the mining depth increases, the drastic increase and enlargement of displacement parameters and fracture zone have occurred apparently. Regarding the general trend of correlation between in-situ stress conditions and mining depths, lateral stress ratios of 0.5, 1.0, 1.5 and 2.0 were applied to understand the adverse impacts in the mine. Based on analyzing results, it can conclude that the higher stress ratio results in larger deformation and fracture zone development significantly. On the one hand, the adverse impacts of a higher lateral stress ratio, for example, a stress ratio of 2.0, is much greater than the effects of seam dips and mining depths from shallow to moderate deep mining. Moreover, in order to reduce the error difference between empiric/numerical study and actual mining situations, the rock mass deterioration approach by reducing the Geological strength index value was utilized. Because initial data of rock properties obtained from laboratory experiments made on an intact rock sample is doubtful to represent the reality of rock mass in the actual mine site. The stability of the gateroad in the analyzing results was worsened substantially. In the cases of that rock mass is strong and very competent as well as not deteriorated, the effectiveness of different supports was not clear, and results were in very stable condition. Using the rock mass deterioration approach in analysis, it can be realized that the passive type support, steel arch, is more effective in improving gateroad stability than active type support, rock bolt. There was another mentionable circumstance. By remaining thicker coal on the roof due to the coal seam dip becoming steeper, the roof condition had improved effectively. This experience has been widely applied in many underground coal mining practices. But, it also has a negative influences that elevate the risk of methane and spontaneous coal combustion during mining operations. So, the optimization of the ventilation system needs to be considered the main role to improve safety issues. Also, standing supports such as steel arches resist the airflow through the gateroad and cause of small air quantity. Those factors and considerations have to be studied in further research.

Chapter 4. This chapter discussed the effects of various geotechnical and geological conditions and face support characteristics on steeply dipping longwall coal mining while the panel is extracted partially. Some essential mining conditions such as coal seam dips, mining depths, lateral stress ratios and rock mass deterioration by reducing the GSI value have been applied to investigate the ground respond associated with rock deformation, fracture zone and vertical stress distribution and magnitude. Despite the specific caving mechanisms, panel excavation and gob modeling steps were assumed to be the same as typical horizontal longwall mining. Three distinct mining phases that are before panel excavation, half of panel excavation and entire panel excavation were involved as well. Firstly, seam dips do not influence the stability of the gateroad, while the effects of various mining depths and lateral stress ratios are significant. Moreover, due to the mechanical properties of intact rock mass, which are obtained from laboratory experiments, rock behavior should be similar to actual conditions. Thus, rock mass deteriorated by certain percentages, and it resulted in larger deformation and fracture zone propagation in the surrounding rock mass of gateroad, chain pillar and longwall face. Based on the those results, it can be realized that in the case of a rock mass deterioration rate of 50%, the wider chain pillar needs to remain in order to maintain gateroad stability. In addition, although the lateral stress ratio effect usually occurs in shallow mines, its influence on ground behavior is considerable. In terms of the face support system, manual prop support was employed and typical phenomenon of beam axial stress and z-displacement has occurred. The higher axial stress is usually concentrated around the middle of the longwall face when rock mass is competent and immediate roof rock is fair to strong. However, after the rock mass deterioration, a substantial increase in both parameters has occurred near the lower part of the face. The beam axial stress and z-displacement were drastically increased, and it can cause of big failure and serious instability of the longwall face excavation. Even though longwall mining can be one of the applicable mining methods for steeply dipping coal, it has its own technical limitations related to the mining conditions, such as seam dip angle. Despite the facts of successful mining practices for steeply dipping longwall coal mining, it can be recommended that the possible seam dip angle that the longwall mining method can apply is around 60˚ or less from geotechnical and technical point of view.

Chapter 5. Designing an adequate gateroad layout that involves the orientations and high horizontal stress direction can be a key factor in improving safety issues associated with ground control and stability, as well as enhancing the economic diversity of underground coal mining projects, especially in terms of longwall mining. Therefore, the interaction between the orientation of the gateroad and major/intermediate horizontal stresses has been investigated to understand the negative and positive impacts on stability. It is realized that the high horizontal stress intersecting with the gateroad direction has significantly large adverse impacts on the surrounding rock compared to other cases, which are k ratio of 1 and parallel horizontal stress direction. It is critical to find the optimal angles that can less influence deformation and fracture zone propagation in the gateroad. In the case of that, high horizontal stress intersecting with gateroad dominantly results in serious deformation and large fracture zone propagation in the surrounding rock mass, particularly on the roof and floor. Besides, parallel high horizontal stress has slightly more affected the larger fracture on the rib sides. The effects of mining depth are relatively obvious, and the proper dimension of chain pillar in greater deep mining should be considered when gateroad is designed to intersect high horizontal stress direction at nearly 90˚. Also, the high horizontal stress direction and magnitude still play the main role in the ground stability even though various intermediate stress conditions were applied in the numerical simulation. It notes that regional and local stress measurement and investigations need to be conducted at each mining case. Nature is diverse, and geology is complex. In addition, for the Narynsukhait coal mine, it can be more appropriate that the longwall face direction can be in-between “along the strike” and “down the seam dip”. By applying this design of face direction, numerous advantages will be followed up with. Although the gateroad becomes inclined, the inclination of the longwall can be reduced and provide an opportunity to mitigate the adverse impacts of the orientation of the gateroad and high horizontal stress. Moreover, by optimizing the direction of longwall face, the direction of gateroads, faces, cut-throughs and installation roadways can be flexible to sustain the severe ground impacts. The simplified illustration of in-between longwall face direction is shown in Figure 6.1.

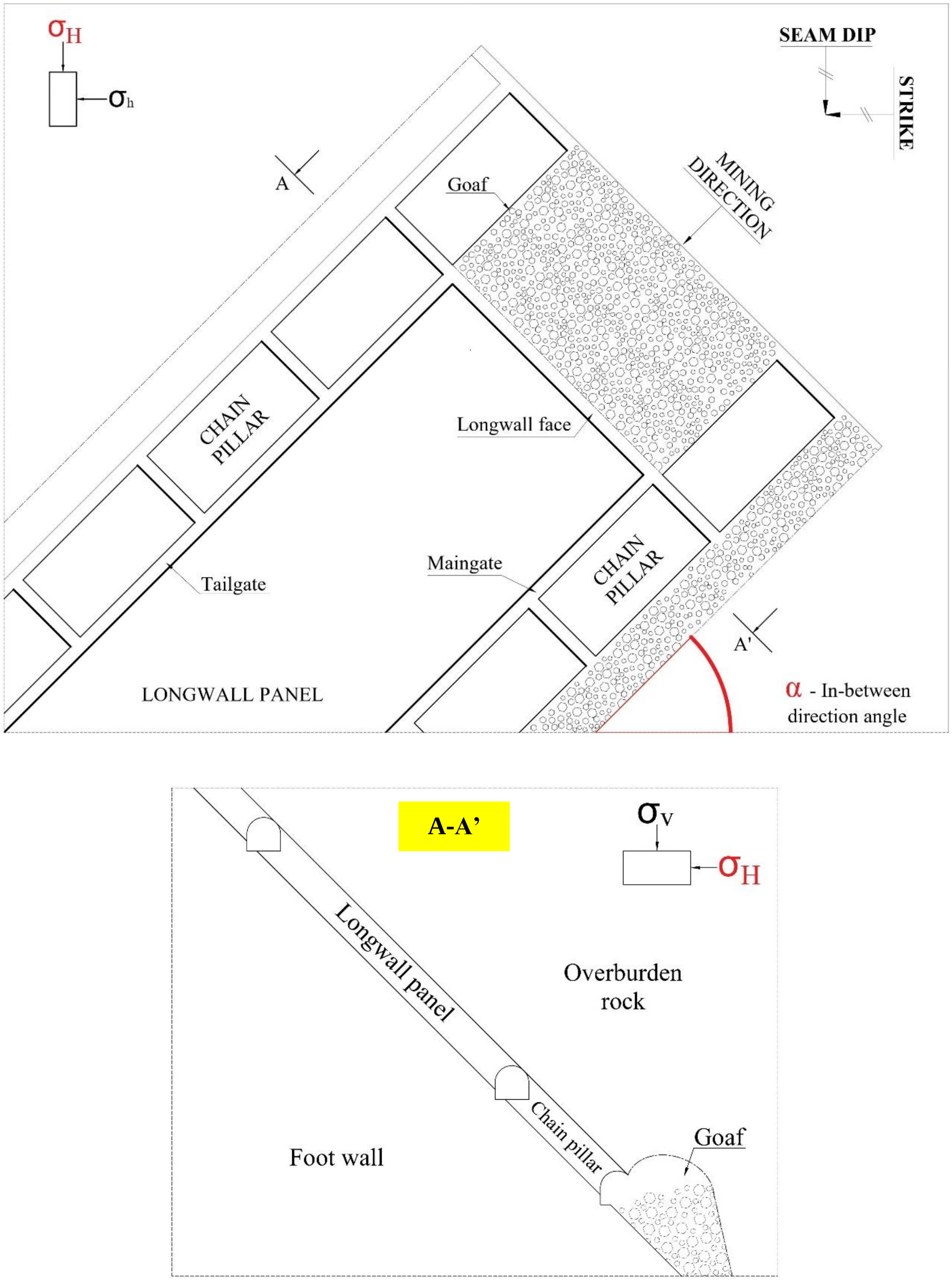

Figure 6.1. Schematic view of in-between face direction in steeply dipping longwall coal mining