APPROPRIATE DESIGN OF UNDERGROUND MINE FOR VEIN TYPE TIN-TUNGSTEN DEPOSITS IN MYANMAR

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MARCH, 2022

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AS A PARTIAL FULFILLMENT OF THE REQUIREMENTS FOR THE DEGREE
OF
DOCTOR OF PHILOSOPHY OF ENGINEERING

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ABSTRACT

Myanmar is rich in mineral resources such as base metals and rare earth elements and is one of the major tin and tungsten producing countries in the world. Although large-scale exploitation of Myanmar's mineral deposits began in the mid-1970s, many deposits of these mineral resources found in Myanmar so far have still not yet been developed. Many of the major mines currently in operation were developed in the early 20th century, others are still unplanned. Most of the operating mines are small-scale and manual extraction is conducted. In order to increase the production and maintain stable supplies of mineral resources, it is indispensable to expand the mining scale and improve the mining efficiency by mechanized operations in the future. The stability of underground excavations has become an important issue in the underground mining operation due to mine enlargement and extraction of deeper mineral resources. Open stope mining method and cut and fill mining method are commonly used as underground mining methods for vein type deposits. However, the application of open stope and cut and fill methods for underground mining generally encounter failures due to the geological structure and high regional stress condition. Although the stoping method is influenced by environmental effects related to the induced stress of natural conditions, the structure and geometry of the quartz-vein type, which is a typical (Sn-W) deposit in this study area, could also have influence. Controlling the stability of stope is crucial, not only for the safety of the mineworkers and equipment but also for preventing the environmental damage due to the mining activities. For this reason, it is an urgent task to study and introduce an environmentally friendly mining method that can minimize the environmental impact due to the expansion of the mining scale. The Hermyingyi Sn-W mine which is targeted in this research is one of the major Sn-W mines in Myanmar located in the Daewi region. In this mine, greisen and mineralized quartz veins are extracted by the overhand open stope mining method. However, the mining method is still small scale by manual operations. Due to the high grade of Sn-

W deposits in this area, it is necessary to expand the cross section of the mining face for introduction of equipment in order to increase the production and improve the safety. From these backgrounds, the purpose of this research is to understand the geotechnical features of vein type Sn-W deposits and to develop an appropriate design of underground mine by means of field investigations, laboratory tests and numerical simulations with $FLAC^{3D}$ver. 5.0 and $Phase^2$ver. 7.0 software. This dissertation consists of six chapters and the main contents in each chapter are listed as follows:

Chapter 1: This chapter describes the mining industry and the general information of Sn-W deposits in Myanmar, the background of this research, the types of mining methods for vein type deposit and their features, the factors influencing the stability of stope and subsidence due to the underground mining activity. The outline of the dissertation is also described in this chapter.

Chapter 2: This chapter describes the mining conditions of Hermyingyi Sn-W underground mine. The exploration works around this mine area have been conducted in the 1900s. The mining area is located in the southern part of Myanmar which is characterized by complex tectonic structure exacerbated by the Sagaing fault which crosses the mine site. According to the geological setting, the host rocks are Mergui (meta-sedimentary) group, alluvial, and Irrawaddy formation. The major granite and associated Sn-W Deposits is in the Southeast Asia Tin Belt. The Sn-W ores occur in granite, aplite, pegmatites, greisen, and quartz veins. The overhand open stope mining method is applied in this mine with small scale and manual excavation. From the results of field investigations and laboratory tests, it was found that the rock mass condition in this mine site is very poor due to the fracture system especially at shallow depth. The problem statement of the research area, which is associated with the current mining method being used, and the objectives of the research are also presented in this chapter.

Chapter 3: This chapter discusses the stability of the stope and sill pillar for a single vein extraction in different geological and mining conditions. The sill pillar is the ore/rock mass left between the adjacent stopes in order to prevent the collapse of working stope. The open stope mining method and cut and fill mining method are considered to be applied. From the results of a series of numerical simulations, it was found that the stabilities of the stope and sill pillar are decreased significantly with decreasing Geological Strength Index (GSI) of rock mass, increasing the width of the stope and the stress ratio of the horizontal ground stress to the vertical one. In cases where the GSI of a vein and hanging wall and footwall rock masses are smaller than 38, the open stope mining method cannot be applied and the cut and fill mining method has to be applied in order to maintain the stabilities of the stope and sill pillar. Moreover, in the current mining condition that the GSI is around 30, the dip of the vein is 80 degree, the height of the mining section is 40 m, the stress ration is 1.0, the depth is 150 m, if the width of stope is changed from 2.0 m to 5.0 m, the cut and fill mining method with waste rock fill should be applied and the sill pillar should be left at least 10 m in thickness.

Chapter 4: This chapter discusses the stability of the stope and the crown pillar in different geological and mining conditions in order to extract shallow deposit. The crown pillar is the ore/rock mass remained between the uppermost stope and the surface in order to prevent surface subsidence due to the mining activities. From the simulation results, in the case that the rock mass condition is less than 55 in GSI, and the dip of the vein is gentler than 70 degree, the stabilities of the hanging wall of the stope and the ground surface are decreased significantly. Especially, in the case that the rock mass condition is around 30 in GSI and the overhand open stope mining method is applied, as the failure zone around the stope is expanded and reach to the surface, 30 m or more thickness of crown pillar has to be left in order to prevent the surface subsidence due to the mining activities. Based on these results, the applications of support systems such as rock bolt and shotcrete and backfilling system were discussed as countermeasures for stabilities of the stope and the crown pillar. As the stope is supported by 5 m length of cable bolt and shotcrete, the thickness of crown pillar can be decreased from 30 m to

25 m. Moreover, as the stope is backfilled with cement and waste rocks, the crown pillar can be decreased up to 20 m. It can be concluded that the optimum mining operation can be done according to the grade and/or price of ore by applying these countermeasures.

Chapter 5: This chapter discusses the appropriate mining method for extraction of multiple veins in different geological and mining conditions. In the case that the rock mass condition is around 30 in GSI, the depth is150 m, the dips of veins are 70 degree, the width of the stope is 5 m, the stabilities of the adjacent stopes is dramatically decreased when the distance between them is less than 15 m. Therefore, 5 m length of rock bolts have to be installed with 1 m spacing in the hanging wall and footwall of each stope. Moreover, when the shallow deposit is extracted, in situations where the rock mass condition is around 30 in GSI, the dips of the veins are gentler than 60 degree, the stabilities of the hanging wall of the uppermost stope and the ground surface are decreased obviously and the potential of surface subsidence becomes high. In this condition, the uppermost of the veins can be extracted with cable bolts and shotcrete applied. After an artificial strong layer is formed by backfilling the uppermost stope with cement paste, the lower veins can be extracted. By applying this countermeasure, the effect of mining activities of multiple veins extraction on the surface can be controlled and the stability of working stope can be improved.

Chapter 6: This chapter concludes the results of this dissertation.

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Fukuoka, Japan

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CHAPTER ONE

1 Introduction

The underground mining method is one of the best option to apply for optimizing the environmental impact at the surface area of mineral exploration for sustainable development of mineral exploration. Myanmar contains substantial undeveloped and undoubtedly undiscovered resources of a range of other commodities (Gardiner, Robb, & Searle, 2014). Recently, the application of shallow surface mine is better than underground mine due to a number of reasons ranging from economic to safety issues. Currently, the research area adopted the open stoping where the economic conflict often arises between the needs to promote mine safety and economic development through natural resources development and to prevent the environmental condition of mine area. Because small-scale mining and primitive smelting of tin by Burmese and Chinese long preceded the commercial exploitation of tin and tungsten in Myanmar.

The study area is located in the complex structure and around the coastal region. Therefore, the stress redistribution in surrounding rocks resulting from the mining, the low stiffness of the filling body due to the existence of the void space, and the influence of on-going mining activities are the main reason for the occurrence of the surface movement and ground fissures. Nevertheless, according to the situation, the open stope method is an adventure for the mine safety, worker, and transportation from the mined-out area. Therefore, the researcher highlights the significance of evaluating the subsidence of shallow underground mines and compared it with the geometry of the problem statement at the tin-tungsten deposit mine by using simulation.

The technique of filling into the stope is the preferable method for optimizing the instability of the underground mine. Even though the failure has still occurred surrounding mined-out area due to particular reasons such as inappropriate supporting method or severe geological condition that possibly disturb the surface of the mine.

Therefore, it is important to analyze the good countermeasure method to prevent that problem.

In order to explore the tin-tungsten metal mine development and control the instability of mine, the mining industry needs to fully understand the parameters of the problem statement and plan of mine design. Also, the result of this dissertation might be utilized for optimizing the instability of the underground mine. In this case, Myanmar can take advantage of its tin wealth by developing a large and sustainable tin industry and become a determinant of the future market balance of the tin industry.

1.1 Background

Myanmar is perhaps one of the largest producers of tin-tungsten in the world for over the next decade because the sudden rise of Myanmar tin has the potential to refocus global tin production (Kettle, Pearce, Lin, & Sykes, 2014c). The tin-tungsten deposits are located along the southern part of Myanmar as shown in Figure 1.1. The potential tin-tungsten deposits were widely discovered around the Dawei district in Myanmar as shown in Figure 1.2. The mining company was constructed for the production of tin-tungsten metal around the Dawei district. Since 1980, tin-tungsten minerals (Sn-W) have been produced in this area. Among them, the Hermyingyi Mine is one of the main production of (Sn-W) deposits mine in Myanmar since World War II.

Background of mineralogy in Myanmar is being endowed with gemstones and ore deposits such as tin, tungsten, copper, nickel, gold, zinc, lead, and nickel. Myanmar can be divided into three principal metallotects: Wuntho-Popa Arc, comprising subduction-related granites with associated porphyry-type copper-gold and epithermal gold mineralization; the Mogok-Mandalay-Mergui Belt hosting both significant tin–tungsten mineralization associated with crustal melt granites, and key orogenic gold resources; and the Shan Plateau with massive sulfide-type, lead-zinc deposits as shown in Figure 1.3. On the other hand, in general, the geology of Myanmar is shaped by the complex ongoing tectonism characterized by the mobile belts. Moreover, the minerals belts currently existent are 8; tin-tungsten belt, antimony belt, lead-zinc-silver belt, gold-iron-copper belt, nickel-chromite-copper-gold-platinum belt, iron-manganese belt, the precious belt, and iron-gas and coal belt as shown in Figure 1.1.

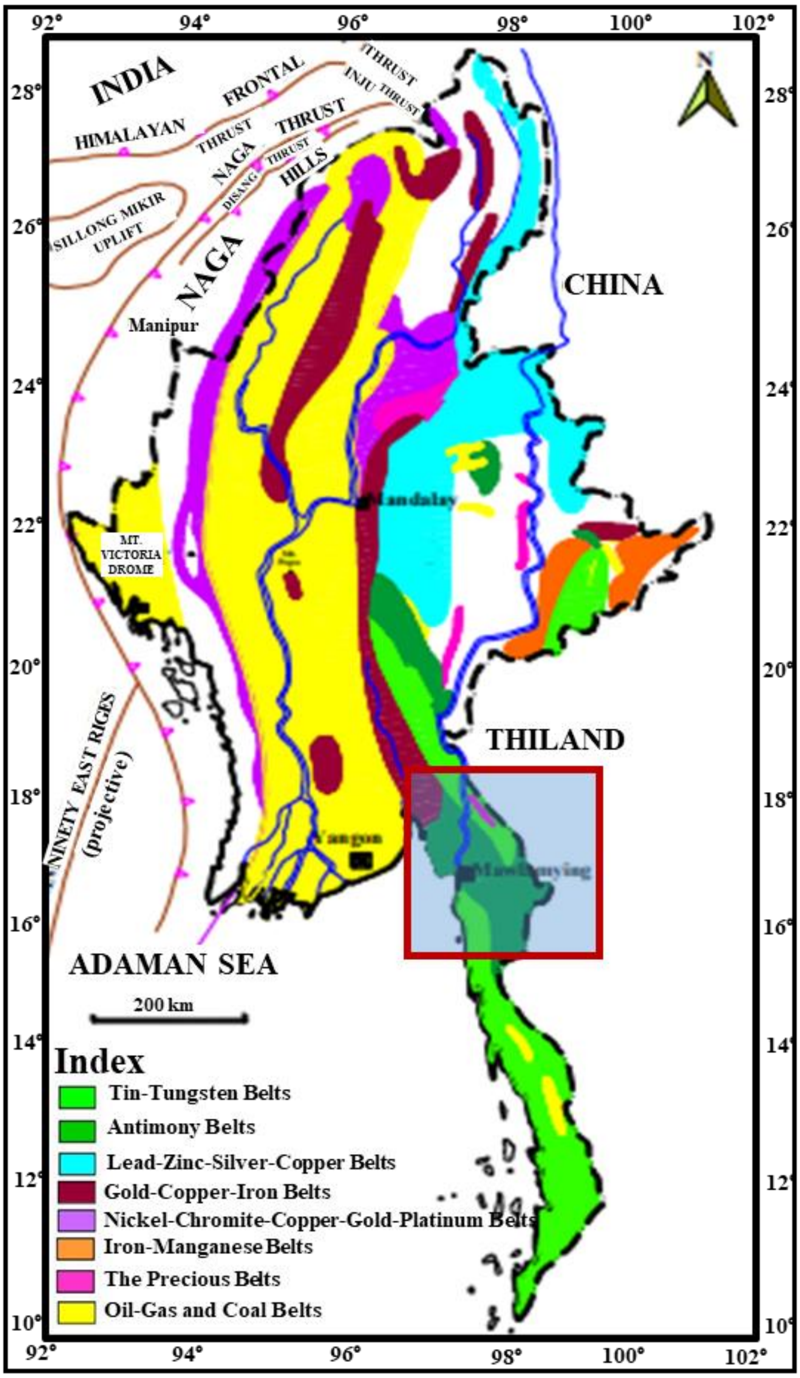


Figure 1.1. The mineral belts of Myanmar (after U Soe Thi Ha, 2006, unpublished).

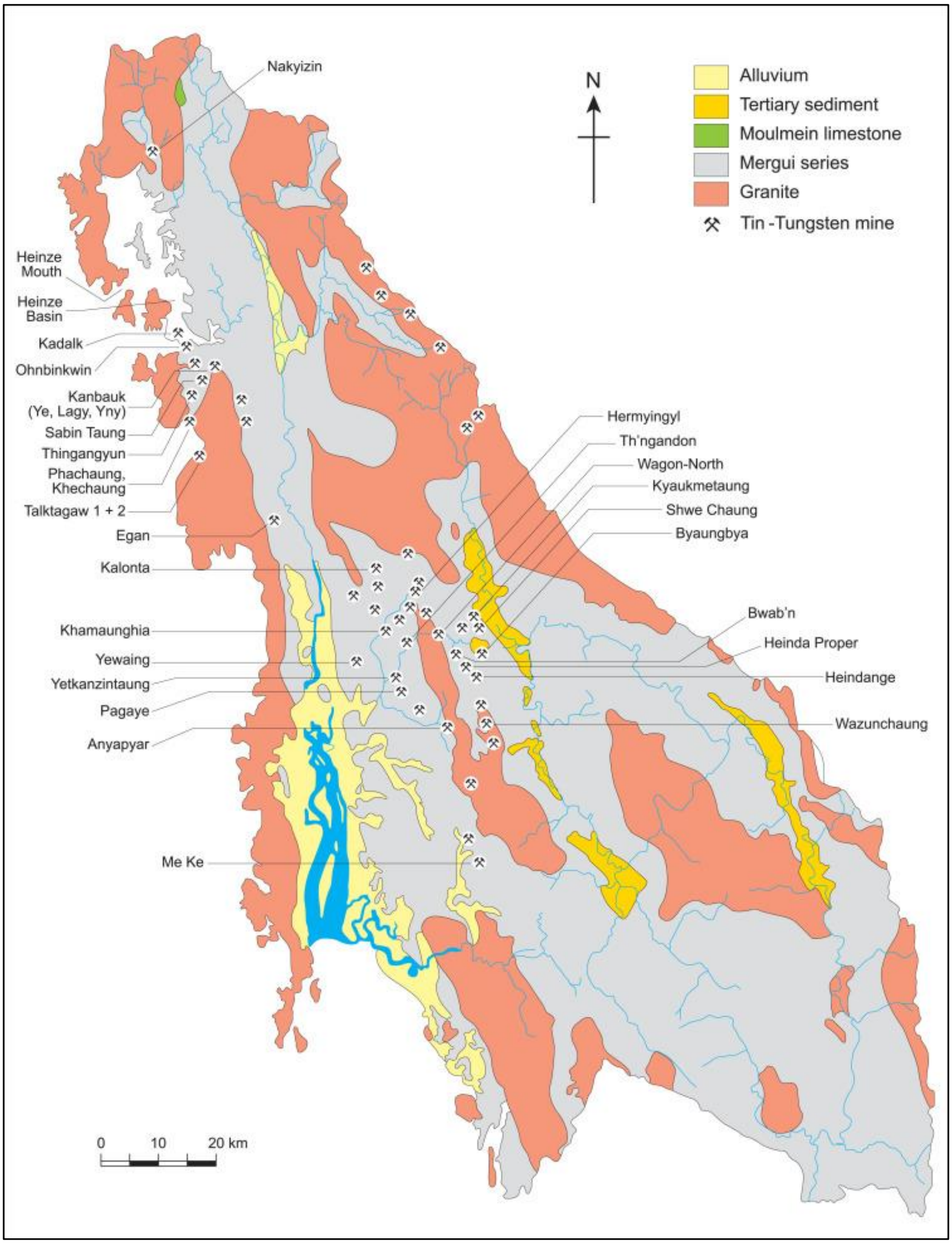


Figure 1.2. Described the influence of Tin-Tungsten mining at Dawei District, Myanmar.

The open stoping underground mining method was adopted and developed for tin-tungsten deposits mined in the research area in Myanmar. Because the selection of the mining method used depends on the characteristics of the ore body, particularly thickness and dip, and the competency of the surrounding rock. In Myanmar, unsupported methods of mining are used to extract mineral deposits that are generally roughly tabular (plus flat or steep dipping) and are generally associated with strong ore and surrounding rock. Therefore, these methods are termed unsupported because they do not apply any artificial pillars to assist in the support of the openings. Even though, a generous amount of roof bolting and localized support measure should be used often. However, the mining systems need to be greatly improved across the country not only for exploitation but also for the safety of mine workers and transportation of metal.

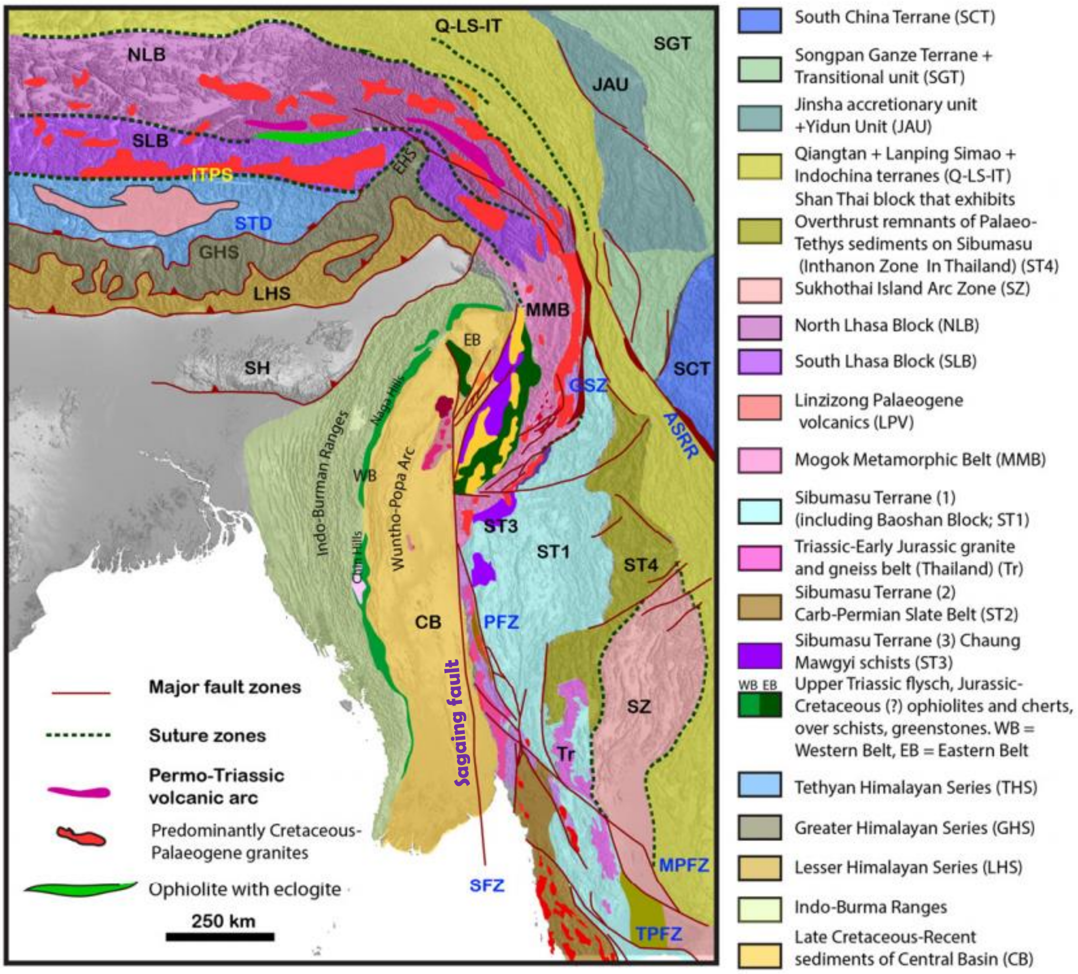


Figure 1.3. Structural and tectonic provinces of Myanmar (Mitchell, et al., 2007).

1.2 Tin-Tungsten Deposit in Myanmar

Iron, silver, and cassiterite deposits have been exploited in Myanmar since antiquity, basically in the Tannintharyi area. Cassiterite and wolframite are usually associated together with the ore deposits mixed concentration existing. Over 122 Sn and W occurrences are known in Myanmar. They are found in the Tanintharyi area and along the western part of Shan State See Figure 1.1., where wolfram seems compressively to replace the cassiterite towards the north (Coggin Brown & Heron, 1923). The Tanintharyi region of southern Myanmar and parts of eastern Myanmar belong to the South-East Asia tin-tungsten belt which runs from the tin islands of Banka and Billiton in the Indonesian Archipelago via Malaysia, Thailand, and Myanmar as far as China; it supplies 65 percent of the world's current production and a large percentage of the world's tungsten. Most of the tin ores are mined in placers with the tungsten extracted from veins. The tin-tungsten ores are bound to the pneumatolytic-hydrothermal dyke system of granites. These tin and tungsten bearing granites of Myanmar belong to the westernmost of the three granitoid belts of the South-East Asian peninsula. For the most part, they belong to the Tertiary intrusion generation of this region. The country-rock of these intrusive masses consists of the clastic of Mergui Series, Taungnyo Group, Mawchi Series, and Lebyin Group shown in Figure 1.4. The study area forms a part of the southern continuation of the Shan-Thaninthari Block.

The tin-tungsten mineralization is often discourteous and frequently occurs in parallel or intersection sets of veins, in lenses and stringers of rapidly varying thickness. The contact with country-rock either district, often characterized by a muscovite-biotite selvage or it is formed by a greisen zone. The veins frequently strike parallel to the granite bodies arranged in the general NNW-SSE strike of the mountain range.

Ore-bearing quartz veins occur in granites in the contact zone to their country rocks, in particular in roof regions of instructions and the surrounding country rock. According to Coggin Brown & Heron, 1923, high concentrations of cassiterite and wolframite are found in particular in several (cm) thick stringers and lenses in the contact zones. Apart from cassiterite, scheellite and wolframite, the paragenesis of the ore-bearing quartz veins consists of molybdenite, pyrite, chalcopyrite, covellite, galena, spalerite, stibnite, bismuthinite, pyrohotite, arsenopyrite, siderite, calcite, cerussite, fluorite, and beryllium (Coggin Brown & Heron, 1923).

Tin placers are found in eluvial, colluvial, fluviatile and lacustrine sediments. They are generally comparable to those of the Kaksa and Mintjan placer of Indonesia. Therefore, near-shore marine tin placers were found offshore of the coast of the Dawei and Mergui districts.

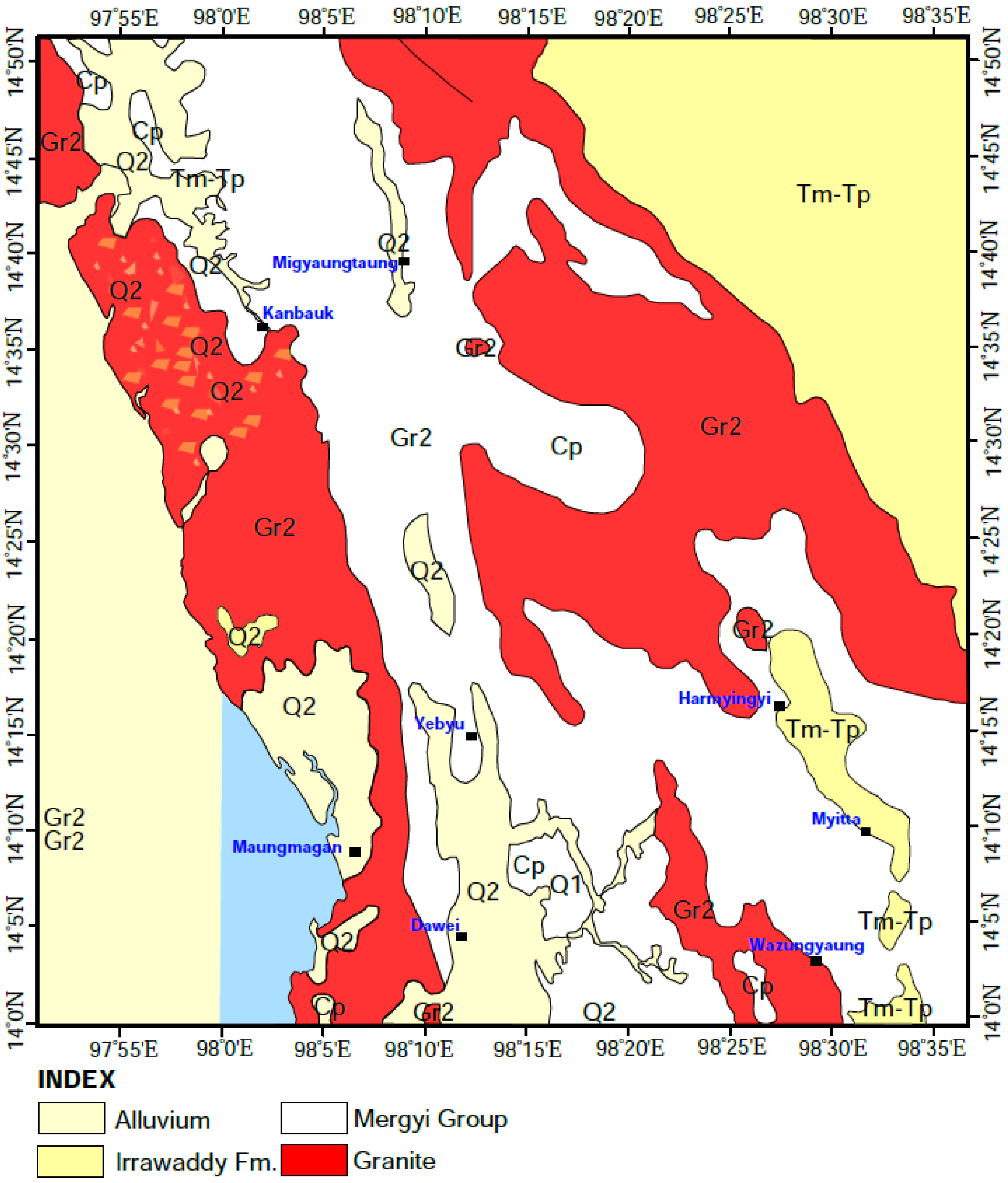


Figure 1.4. Regional geological map of the Dawei District, Tanintharyi Division after Bender, 1983.

Many of more than 120 tin-tungsten occurrences are located in the Dawei district, where, they are associated with three NNW-SSE oriented granite belts of the

Coastal Range, the Central Range, and the Frontier Range in Figure 1.3. Of the three important producing mines in this district, the primary ore deposit at Hermyingyi and the tin placer deposit at Heinda are situated in the Central Range, and the primary and the placer deposits at Kanbauk are located in the Coastal Range granite. Other actual productive mines are Yadanabon, Myanmatti, and Mawchi.

1.3 Tin and Tungsten Production; A Global Perspective

1.3.1 Tin

Tin (Sn) has been used and traded by man for more than 5,000 years, it has been found in the tombs of ancient Egyptians, and was exported to Europe from Cornwall, England, during the Roman period. The principal ore tin is cassiterite (SnO_2) but some tin is produced from sulfide minerals such as stannite (Cu_2FeSnS_4). Tin is a silver-white metal of low melting point, is highly ductile and malleable, resistant to corrosion and fatigue, can alloy with other metal, is non-toxic, and is easily recycled. At temperatures below 13°C, it can change to an amorphous grayish powder known as grey tin.

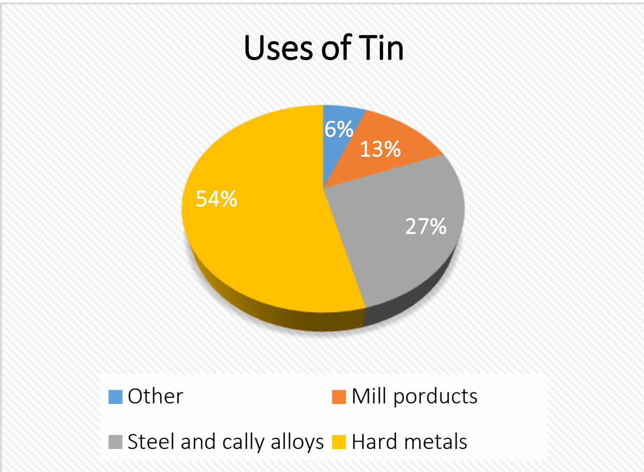


Figure 1.5. The main uses of Tin in the world.

Tin has many important uses throughout the world, particularly tinplate which is used as a prospective coating on steel for food packaging. The dominant driver of tin demand for much of the 20th century has been tinplate, which constituted some 40% of tin used in the 1970s (ITRI, 2012a). The tin is used in the production of the common tin and cally alloys as 27%, mill product as 13%, hard metal as 54%, and other 6%, receptively as shown in Figure 1.5. Moreover, it is also used with titanium in the aerospace industry.

Depending on the economy, the tin price crash of the mid-1980s, the result of the collapse of the International Tin Agreement (Anonymous, March, 1986), resulted in two decades of depressed global tin prices as shown in Figures 1.6(a) and (b).

The following economic and social impact on the traditional tin-producing regions of Southwest England, Australia, and Southeast Asia (primarily Malaysia and Thailand) were considerable, resulting in a legacy of mine decommissioning and the relocation of tin production to cheaper producers such as Brazil, China, and Indonesia (Kettle P. P., 2014b) (Thoburn, 1994).

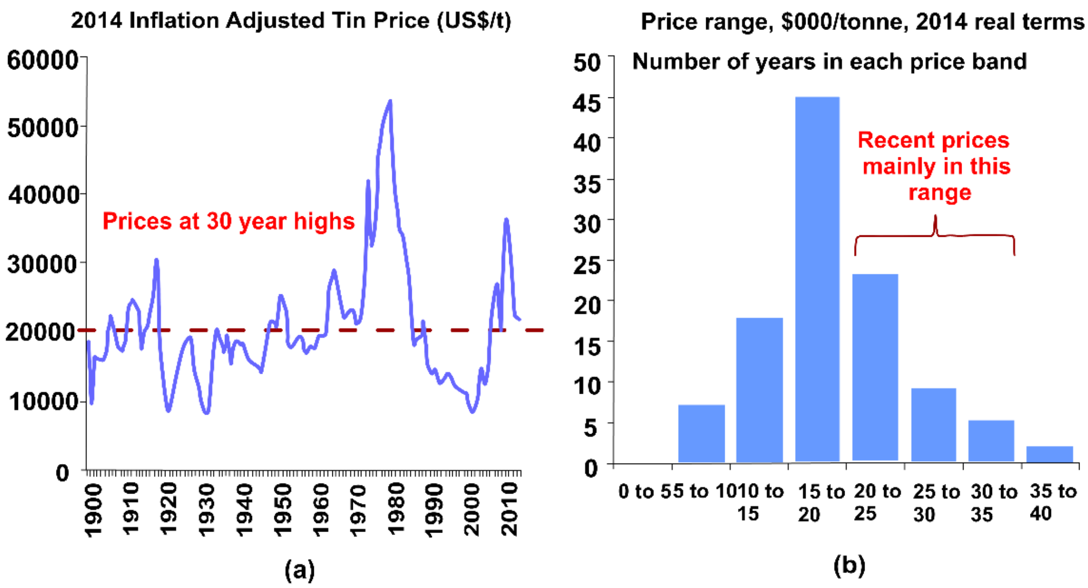


Figure 1.6. The graph shows resulted in two decades of depressed global tin price: (a) Long-term tin price history and (b) Long term tin price histogram.

However, since 2008, global tin prices have recovered to reach a consistent high of ca. US$20,000/t (per metric tonne) in real terms, a level not seen for over30 years. This has largely been driven by a combination of new demand usage, especially for solder in the electronics sector; the economic growth of China; and by a restricted supply pipeline, the result of a lack of significant new exploration projects (Kettle, Pearce, Lin , & Sykes, 2014b) and (Kettle, Pearce, Lin, & Sykes, 2014c)

1.3.2 Tungsten

In nature of tungsten occurs as wolframite, $(Fe, Mg)WO_4$, and scheelite, $CaWO_4$. Tungsten is usually supplied as a mineral concentrate. In 2013, the tungsten metal was 71,000 tonnes in world production and was dominated by China. The major use of tungsten is with cemented carbides. Tungsten carbide is used for cutting in wear-resistant material. And tungsten and its alloys are amongst the hardest of all metals. Tungsten alloys are used in electrodes, filaments, wires and components for electrical heating, lighting and welding applications.

1.4 The Current State of Tin Mine Supply

The world's single biggest tin mine producer was the giant San Rafael Mine in Peru, up to the time of the apparent recent rise of the Man Maw Mining Complex in Myanmar. However, San Rafael is an age in rapidly, with average worked ore grades of 2.5 wt % tin (wt% Sn) (Minsur, 2014). From a high of over 30,000 t/d in the early 2000s, annual production has been in decline since 2006 and is now close to 25,000 t/a. Although a tailings operation is planned, it is not enough to replace the scale of the main underground mine (Kettle, Pearce, Lin , & Sykes, 2014b; Kettle, Lin, Tianhua, Mulqueen, & Davidson, 2015b). Central Africa is another tin jurisdiction that has seen a recent surge and subsequent decline in production. At its peak between 2007 and 2010, it was responsible for about ca. 5% of global tin production (some 15,000 t/a).

The majority of resources are located in the North and South Kivu regions of the Democratic Republic of Congo (DRC) (Melcher & et.al, 2105), an area ravaged by civil war between 1996 and 2003. Many of these mines were controlled by militias, who used the revenues to fund their conflicts. This so-called "Conflict Minerals" trade led to international efforts for supply chain monitoring to stop the supply of Conflict Minerals into the global market (Kettle, Lin, Tianhua, Mulqueen, & Davidson, 2015b).

Table 1.1. Estimate global tin production 2009-2014 (Mt) by (ITRI, 2015).

Country	2009	2010	2011	2012	2013	2014	%Change
China	86,700	95,600	102,000	89,800	96,600	103,400	19.26
Indonesia	102,900	99,700	104,800	92,200	94,200	86,300	-16.13
Myanmar	600	1,400	4,700	4,800	17,000	30,000	4900.00
Peru	37,500	33,800	29,000	26,100	23,700	23,100	-38.40
Bolivia	19,600	20,200	20,400	19,700	19,300	19,800	1.02
Brazil	10,400	7,400	800	10,600	11,600	12,100	16.35
Austria	5,900	6,400	5,100	5,900	6,200	6,900	16.95
DR Congo	13,100	10,600	6,000	5,100	5,100	6,700	-61.83
Other Africa	3,300	3,200	5,100	5,700	6,200	5,000	103.03
Malaysia	2,400	2,700	3,300	3,700	3,700	3,600	50.00
Other Asia	3,400	4,200	4,000	5,800	5,900	6,400	88.24
Russia	300	500	300	400	600	400	33.33
World total	286,100	285,700	293,500	269,800	290,100	303,700	6.16%

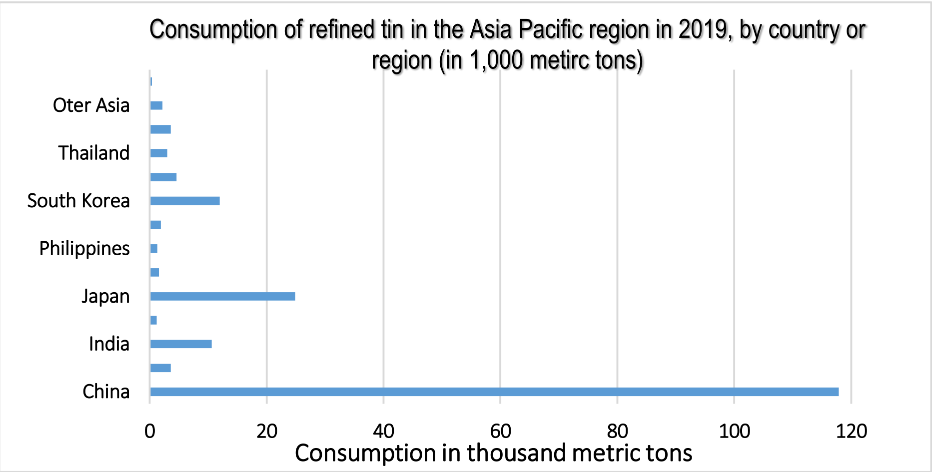


Figure 1.7. Consumption of refined tin in the Asia Pacific region in 2019, by country or region (in 1,000 metric tons)

At present, Indonesia and China are each responsible for about one-third of the global tin mine supply. Table 1.1, the surprise rise of Myanmar tin production in 2014 means it ranked third globally for tin mine supply, being responsible for about 10% of world production. However, at present, the tin mining industry faces a number of challenges in global tin supply (Sykes, Gardiner, Trench, Robb, & Davis, 2015).

Indonesia has suffered from artisanal mining rushes since the 1990s. A characteristic of these rushes is their short-term boom-and-bust nature. Production rises rapidly before quickly collapsing, due to both the low cost and exploitable nature of the deposits, and to the difficulty in predicting grade and tonnage. Indonesian production appears to have peaked at about 100,000 t/a in 2006 and has since gone into decline, with levels now struggling to top85,000 t/a (Kettle P. , 2011; Kettle, Pearce, Lin , & Sykes, 2014b). It is likely that Indonesian tin mine production will struggle to hold at current levels and may decline (Sykes, Gardiner, Trench, Robb, & Davis, 2015). The Chinese industry is rapidly maturing, with grades falling at existing operations and few green fields projects. China is now typically a net importer of tin (Kettle, Pearce, Lin , & Sykes, 2014b; Sykes, Gardiner, Trench, Robb, & Davis, 2015).

In summary, several of the major tin mining countries are now facing problems in maintaining previous supply levels. Figure 1.7, Major tin producers; tin mine production in 2014; the line indicates percentage change 2009–2014 by Kettle et al. (2014b; 2015b).

1.5 Literature Review

1.5.1 Deposit geology

This deposit consisting tin, tungsten, wolframite, calcite, quartz, veins, stockworks, stratabound replacement deposits, and vein type of granite rock in meta-sedimentary rocks at or near granite contacts. According to Simon, 1993, a vein types deposit is composed of a deposit of ore filling a fissure, or fissures, in rock (Simon, 1993). Figure 1.8 shows the vein system of the Heymyingyi Sn-W Deposit Mine (Bender, 1983). This term is non-genetic and is non-restrictive with regard to orientation. The history of vein-type ores illustrates a marked change from their being the major source of most metals, up to the late 19th century, to their being the source of a more limited range of metals today, and in the foreseeable future. Although this volume falls short of a review of all commodities and deposit types, it is justified on the grounds that it contains many new descriptions and represents a time-capsule of the late 20th-century thinking on the geological setting, classification and genesis of vein-type ores. An ore deposit consists of one or more ore bodies.

Even though, the ore body types can be discovered near at the research area. An ore body is a mass of rock that contains ore and from which a commodity of value will be extracted. Not all ore within an ore body will be extracted. Ore bodies are divided into reserves and resources. Reserves are ore that is economically feasible to mine and for which there are no legal or engineering impediments to mining, while resources are ores that may potentially be extracted at some point in the future. Engineering constraints is one of the factors that will influence what ore is economic to mine shown in Figure 1.9. The mining of an ore body may be from a river stream, an open pit, an underground mine, or a combination of the two.

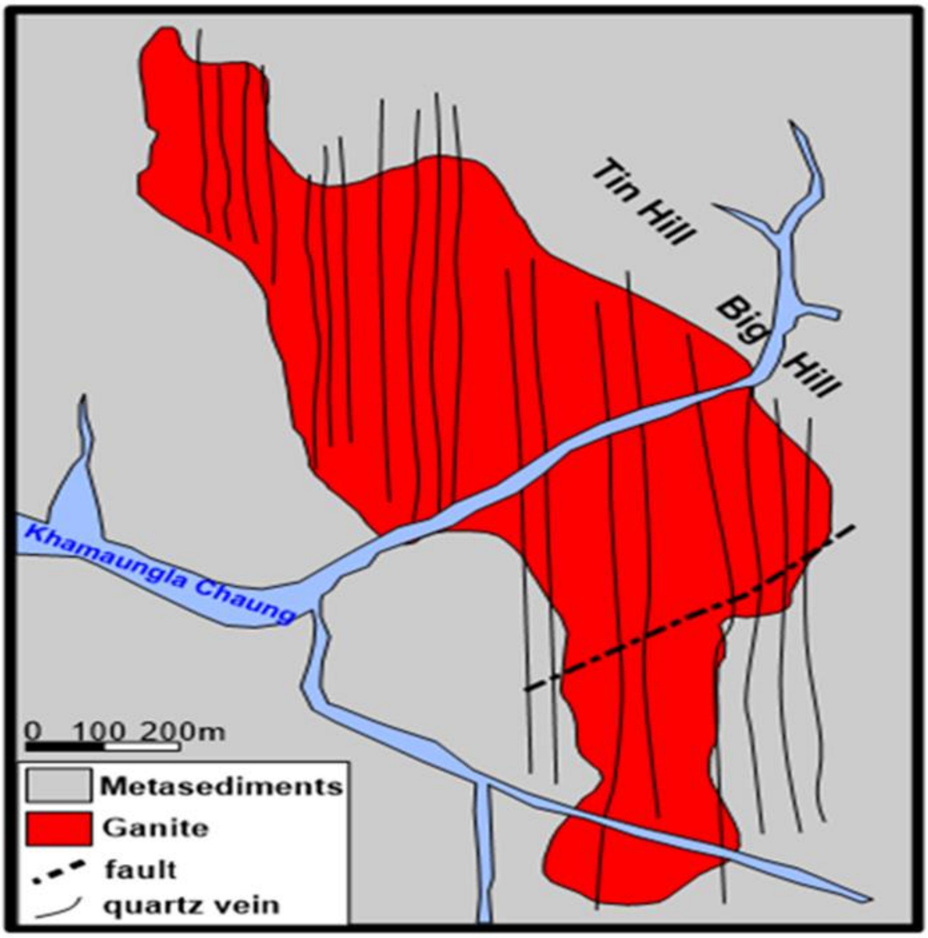


Figure 1.8. Deposit type in the research area after Bender, 1983.

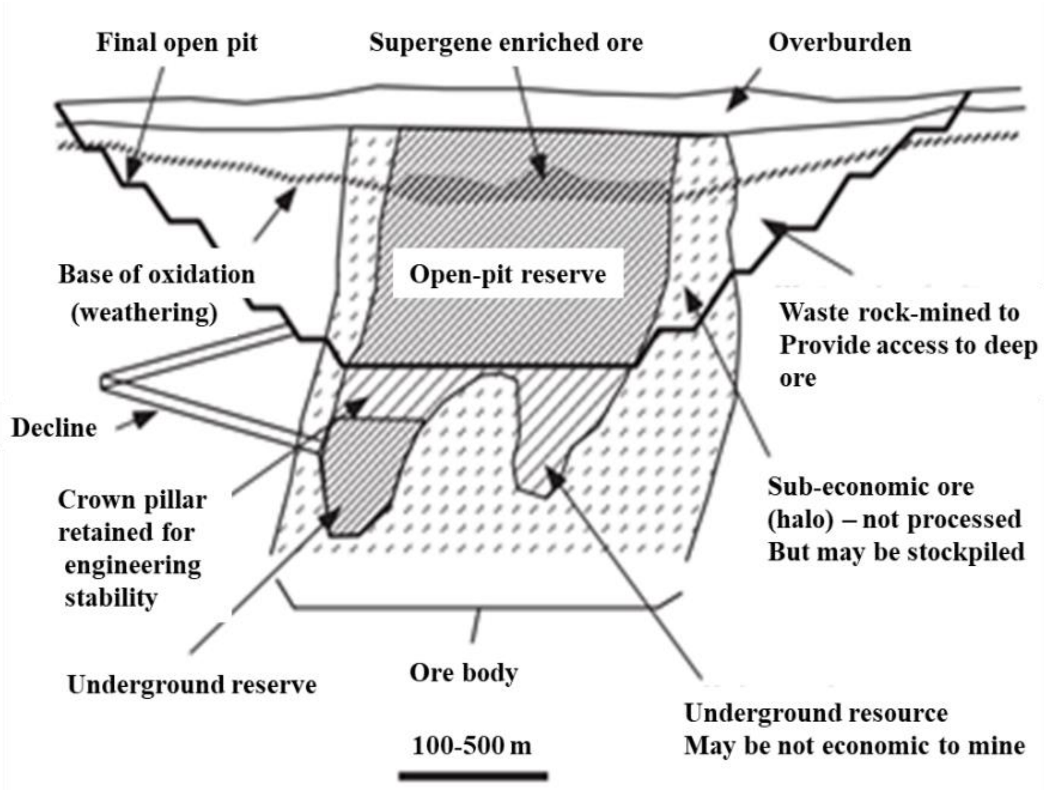


Figure 1.9. Diagram of the type of ore body deposit.

1.5.2 Classification of underground mining method

The usage of mining methods in the research area is consisting of the open stope method, the room and pillar method, the shrinkage stoping method, the cut and fill method, and the sub-level stoping method. Nevertheless, mineral production in which all extracting operations are conducted beneath the ground surface is termed underground mine. Underground mine methods are usually employed when the depth of the deposit and/or the waste to ore ratio (stripping ratio) are too great to commence a surface operation. Once the economic feasibility has been verified, the most appropriate mining methods must be selected according to natural/geological conditions and spatial/geometric characteristics of mineral deposits. Considering include:

Spatial/geometric characteristics of the deposit concerned: the shape, size, thickness, plunge, and depth.

- Strength of the hanging wall, footwall, and ore body.
- Economic value of the ore and grade distribution within the deposit.

The selection of underground mining methods is primarily based on the geological/spatial setting of the deposit. Candidate methods can therefore can be chosen and ranked based on estimated operational/capital costs, production rates, availability of labors and materials/equipment, and environmental considerations. The method offering the most reasonable and optimized combination of safety, economics, and mining recovery is then chosen.

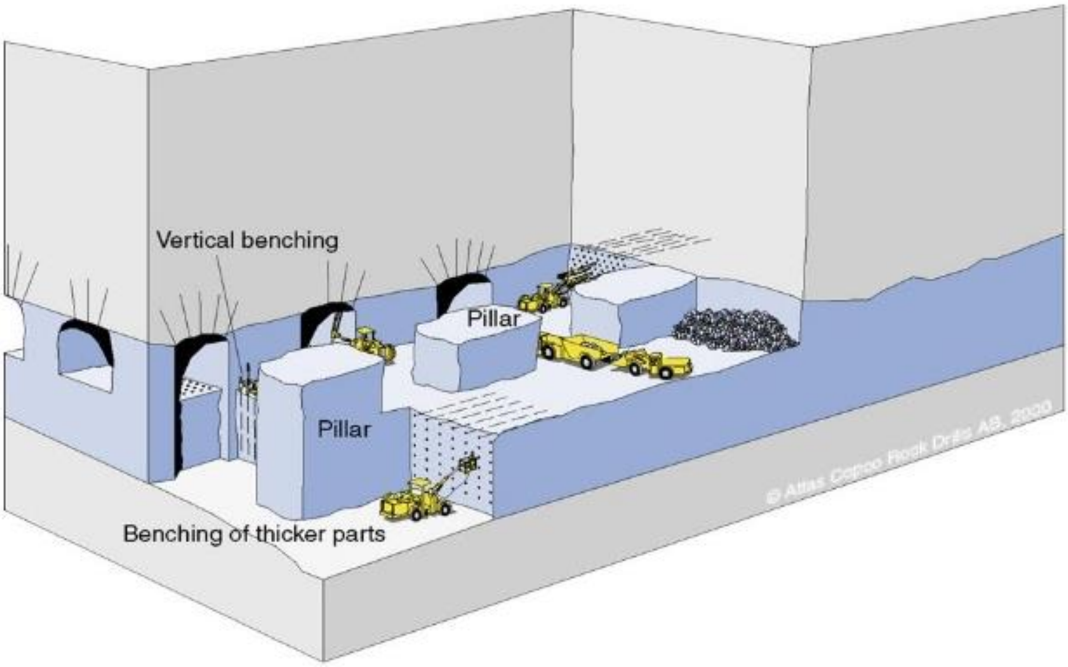
According to the current available underground mining methods, we should consider the following mining system for Sn-W Deposit Mine.

(1) The cut and fill method
(2) The room and pillar method
(3) The open stoping method
(4) The sub-level stoping method
(5) The shrinkage stoping method

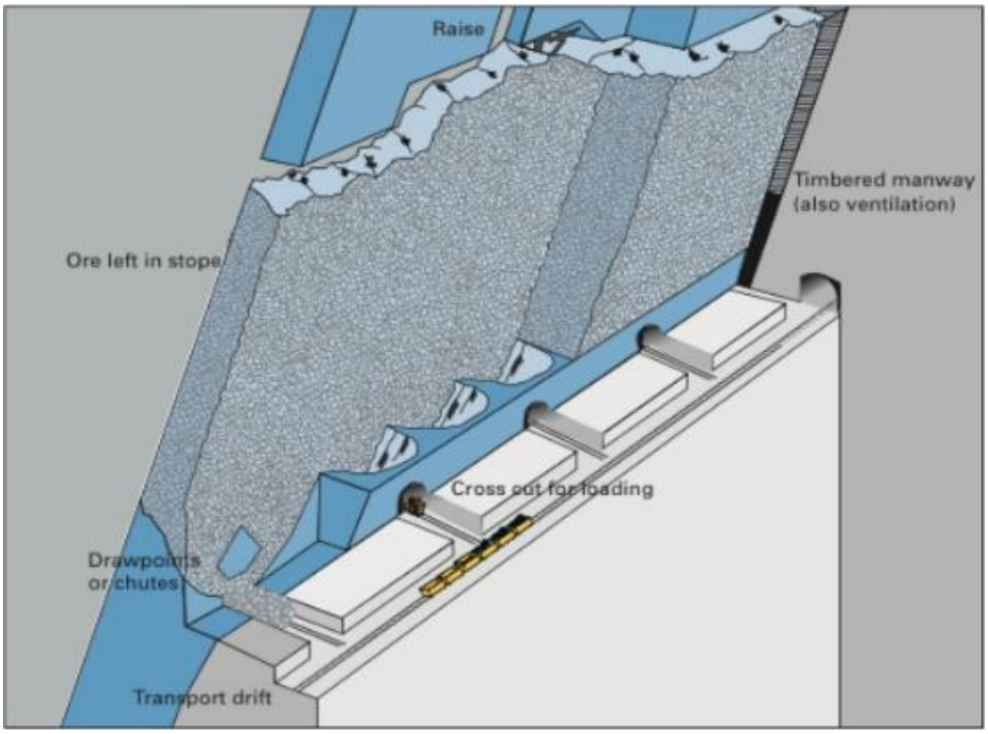
Unsupported methods are essentially naturally or self-supported and require no major artificial system of support to carry superincumbent loads, instead relying on the natural competence of the walls of the openings and pillars. This definition of unsupported methods does not exclude the use of rock/roof bolts or light structural sets of timber or steel, provided that such artificial supports do not significantly alter the load-carrying ability of the natural structure. The following are considered unsupported methods:

- The room and pillar mining method
- The shrinkage stoping method
- The sub-level stoping method

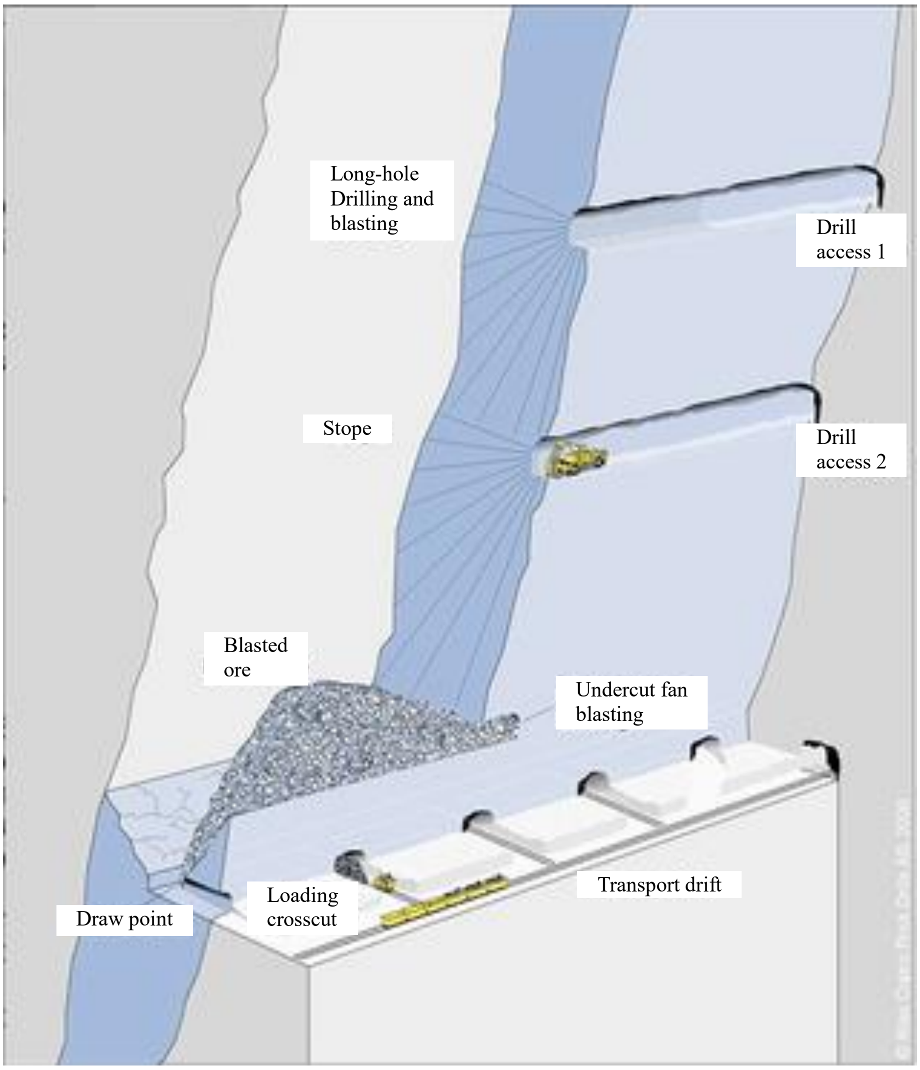
Figures 1.10 (a), (b), (c) show the unsupported methods, where, room and pillar mining is employed for the extraction of flat-dipping and tabular deposits, whereas the shrinkage and the sub-level stoping is applied to vertical or steeply inclined ore bodies.



(a)



(b)



(c)

Figure 1.10. Illustrate the configuration of the mining method in the research area: (a) The room and pillar method, (b) The shrinkage stoping, and (c) The sub-level stoping.

The shrinkage stoping is used in steeply dipping, relatively narrow ore bodies with regular boundaries. Ore and waste (both the hanging wall and the footwall) should be strong, and the ore should not be affected by storage in the stope. The shrinkage stoping has in the past been very popular, particularly in non-coal mining. Gravity can be utilized for ore transportation and broken ore stored within the stope may function as a working platform and temporary wall supports. It was quite attractive in the period before mechanization became widespread when small-scale operations on vein-type deposits prevailed. However, with rising costs, the scarcity of skilled labor, and the trend toward mechanization, the method has been largely displayed by the sublevel stoping and the cut and fill. The sub-level stoping will therefore be the only unsupported method examined in detail here.

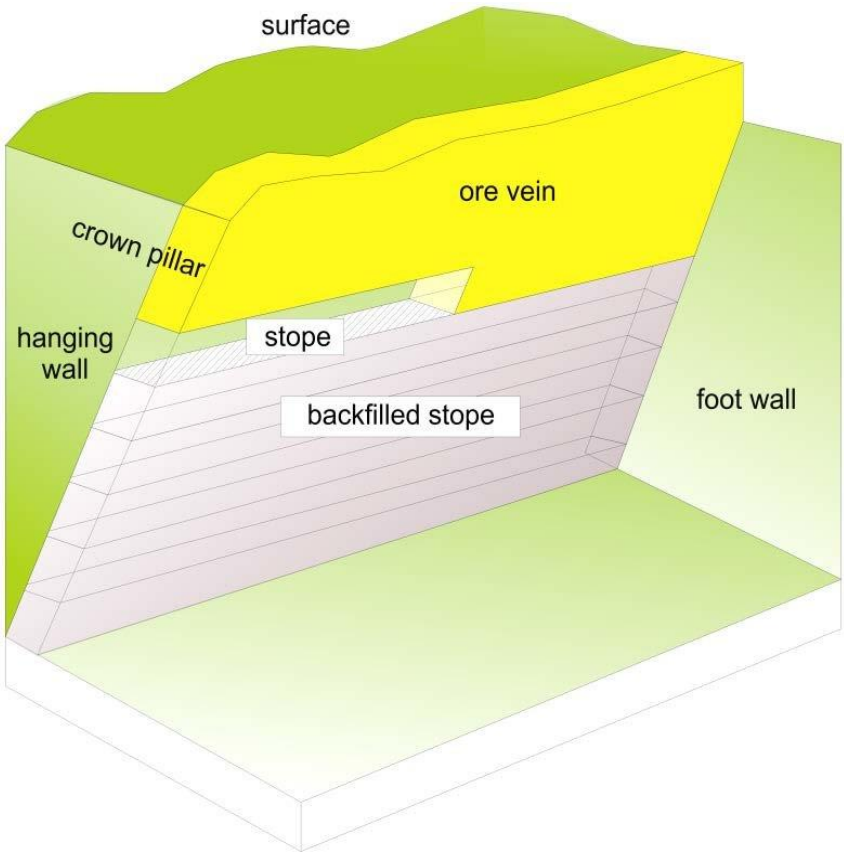


Figure 1.11. Illustrate the configuration of the cut and fill method in the research area.

Supported methods require substantial amounts of artificial support to maintain stability in exploitation openings, as well as systematic ground control throughout the mine. Supported methods are used when production openings (stopes) are not sufficiently stable to remain open during operation, but the opening is required to help open to prevent caving or surface subsidence. In other words, the supported class is employed when the other two categories of methods, unsupported and caving, are not applicable. The supported class of mining methods is intended for application underground conditions ranging incompetency from moderate to incomplete. The design of artificial support systems to provide varying degrees of the natural rock structure is a prerequisite. The most satisfactory forms of artificial support are backfilling/stowing, timbers/results, cribs/packs, and hydraulic/frictional props. There are three specific methods in the supported class such as cut and fill stoping, stull stoping, and square-set stoping. Although, the cut and fill methods is applied in the research area.

The supported methods in the cut and fill have declined in use since World War II, primarily because the cut and fill is the only method that itself to mechanization as shown in Figure 1.11. The stull stoping and square-set stoping are infrequently used and relatively unimportant today, because of excessive labor intensity and very low productivity, in addition to a scarcity of skilled work forces and available timber resources. Only the cut and fill stoping will be described later in detail.

1.5.2.1 Underground operation in general

Figure 1.12 shows a schematic mine layout for an underground mine as an explanation of common mining terms. An inclined shaft is excavated for transportation of personnel and supplies in addition to a vertical shaft for hoisting mind ore and a ventilation shaft for enhancing circulation of fresh air with the aid of a fan. Alternatively, an inclined shaft can be used for main ore transportation and ventilation, and the vertical shaft can act as the main thoroughfare for employees and materials. The functions of inclined and vertical shafts are usually versatile in mining operations.

Main levels are arranged for main haulage and connection between underground key facilities. A small number of sublevels may be situated between main levels within a stoping area for production. Raises and winzes are either vertical or sub-vertical openings driven between one level and another or the surface. Raises are driven upward, and winzes are driven downward. A ramp is an inclined opening connecting levels or stopes to enable the passage of vehicles and an ore-pass is a vertical or sub-vertical hole through which ore is transferred to lower levels.

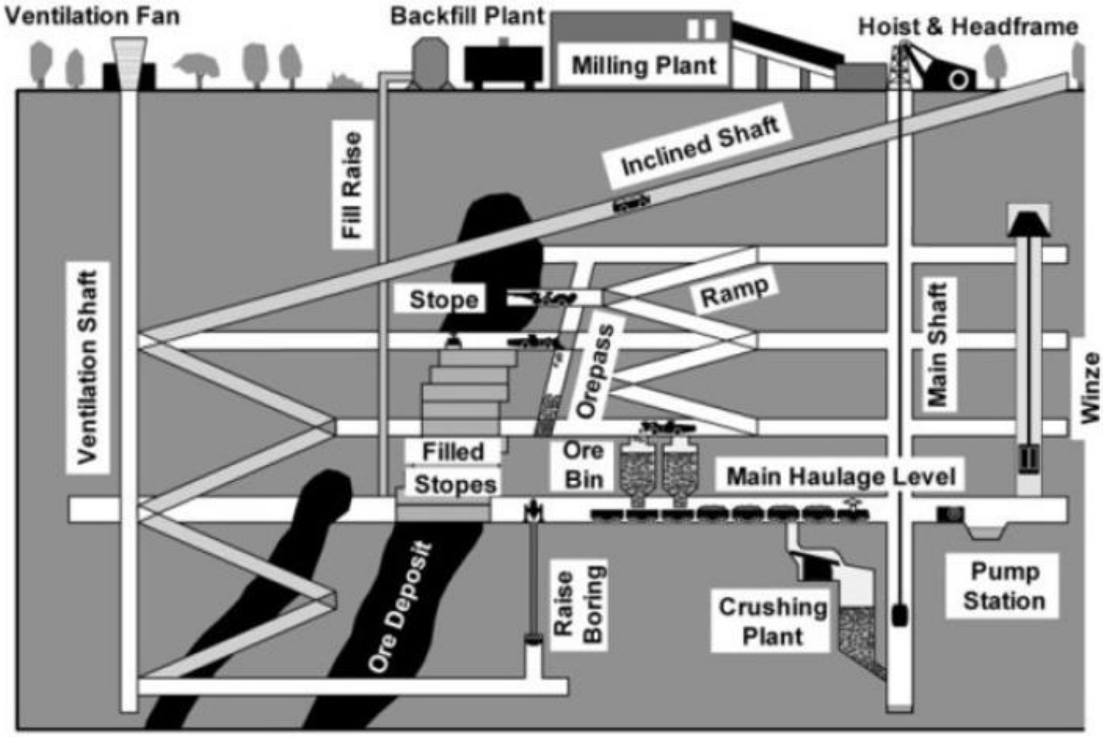


Figure 1.12. Example of layout of an underground mine.

If backfilling (to fill mined-out spaces with rock, soil, or tailings) is part of the underground operation for improving recovery, safety, and mitigation surface subsidence, a fill raise will be constructed for transporting backfill materials prepared at the backfill plant on the surface to underground stopes. If necessary, additional ventilation raises will be driven to improve the underground climate.

The primary requirements underground are rock breakage and material handling. If the target is sufficiently soft (e.g., coal and rock salt), mechanical excavation utilizing continuous miners, shearers, ploughs/plows, and so on, can be employed. In contrast, drilling and blasting are usually employed when the ore is too hard for cutting. A drifting jumbo equipped with ore or multiple rock drills is employed penetrating horizontal blast-holes and used for driven horizontal shaft and inclined ramps as well as production by crosscutting in the room and pillar mining and the cut and fill stoping. Fan/ring/parallel longhole drill rigs are commonly employed for large-scale production in the sublevel stoping and the sublevel caving. Ammonium nitrate and fuel oil (ANFO), slurries, and emulsions are widely used as explosives in mining operations. The explosive is usually charged by hand if available in cartridge form, or pumped into blast-holes for liquid or bulk explosives. The explosives are then fired by electrical or other means.

The most common equipment for material handling such as loading and hauling excavated ore are slashers, gathering-arm loaders, overhead loaders, load-haul-dump units (LHDs), and rubber-tired cars and trucks, as well as transportation by conveyor, rail, and gravity flow. Ore is loaded into underground equipment and transferred to orepass, where the ore is dumped. A draw point or chute is usually situated at the outlet of the pass on the lower level, where the ore is loaded and transported to the underground ore bins or directly to the main haulage level. Finally, ore is collected to facilitate transportation by conveyors or skips through shafts (see Underground Mining Transportation Systems).

The auxiliary/supporting operations required underground include:

1. Healthy and safety: ventilation, gas control (particularly in coal mines), dust suppression, noise reduction.
2. Ground control: supporting (rock-bolting, timbering, setting steel arches, etc.), scaling (removing rock fragments from working roofs).
3. Power supply and lighting.
4. Drainage and flood control: pump stations and sumps are usually constructed at the bottom level to collect and drain water from underground after removing suspended solids.
5. Maintenance and repair of equipment: underground workshops and warehouses.

In addition, surface facilities such as an administration office, milling plant, hoist & headframe, electric substation, emergency power generator, air compressor, and tailings pond are necessary.

1.6 Research Outlines

This dissertation includes six chapters; they are as follows;

Chapter 1: This chapter introduces the background of mineralization and exploration of (Sn-W) deposits in Myanmar and the global price, production, and demand of tin-tungsten. To meet the objective, the literature review is important for the dissertation also discussed in this chapter.

Chapter 2: In this chapter, the research explains the background and mining plan and it is the method. Depending on the initial background of mine production was analyzed the instability of the mined-out area. Therefore, the objective was included in this chapter.

Chapter 3: Stability of single vein's stoping area is the main evaluation of this chapter. The study has been completed to understand which is the important point such as various geological conditions and different stress ratios, after the excavation of stope under comparing with opening stope and filling stope, as a result, it is controlled with different types of filling material.

Chapter 4: This chapter discusses the stability of crown pillars in order to advise how to maintain the thickness and the ratio of span of crown pill from the based on the various inclined vein. From these results, the countermeasure is recommended for the control of the instability of the crown pillar.

Chapter 5: This chapter introduces the instability of stope at multiple veins excavation because in this study area is lied under a parallel vein system according to the regional cross-section map. Therefore, the chapter has analyzed the stability of stope under parallel excavated veins under a variety of geological conditions such as GSI value, stress ratio, the sequence of mine excavate, and so on. As a result, the unstable condition has been observed at the weak geological strength. On the other hand, the guidance of mine design is included in this chapter.

Chapter 6: This chapter has concluded the dissertation.

1.7 References

Anonymous. (March, 1986). *Resources Policy*, 2-3.

Bender, F. (1983). Geology of Burma; Gebruder Borntraeger. *Berlin/Stuttgart, Germany*, 293.

Coggin Brown, J., & Heron, A. (1923). The Geology and Ore Deposits of the Tavoy District. *Geol. Sur. India 1923, 44*, 157-354.

Gardiner, N. J., Robb, L. J., & Searle, M. P. (2014). The metallogenic provinces of Myanmar. *Appl. Earth Sci (Trans. inst. MIn. Metall. B) 123(1)*, 25-38.

ITRI. (2012a). *Tin for Tomorrow: Contributing to Global Sustainable Development .* St Albans, October 2012: ITRI Ltd.

ITRI. (2015, June 1). YTC Forecasts Decline in Myanmar Tin Production.

Kettle , P. (2011). St. Albans 16 April: ITRI Ltd.

Kettle, P. P. (2014b). *Tin Industry Rewiew: Managing the Next Tin Crisis.* St. Albans: ITRI Ltd.

Kettle, P., Lin, C., Tianhua, R., Mulqueen, T., & Davidson, V. (2015b). *CRUMonitor Tin: All Eyes on Supples,*. St, Albans 19 February: CRU Group Ltd in associated with ITRI Ltd.

Kettle, P., Pearce, J., Lin , C., & Sykes, J. (2014b). *Tin Industry Review: Managing the Next Tin Crisis.* St. Albans: ITRI Ltd.

Kettle, P., Pearce, J., Lin, C., & Sykes, J. (2014c). *Tin Industry Review: Managing the Next Tin Crisis.* St. Albans: ITRI Ltd.

Melcher, F., & et.al. (2105). Tantalum-(niobium-tin) mineralization in African pegmatites and rare metal granites: constraints from Ta-Nb oxide mineralogy, geochemisty and U-Pb geochronology. *Ore Geol.Rev. 64*, 667-718.

Mitchell, A., Htay, M. T., Htun, K. M., Win, M. N., Oo, T., & Hlaing, T. (2007). Rock relationships in the Mogok Metamorphic belt, Tatkon to Mandalay, central Myanmar. Journal of Asian Earth Sciences, 29, 891–910.

Simon, J. H. (1993). *Vein-type ore deposits: Introduction.* St. Catharines: Department of Geological Sciences, Brock University.

Sykes, J., Gardiner, N., Trench, A., Robb, L., & Davis, R. (2015). The economics of tin mining in the early 21st century. *Resource Policy*, In preparation.

Thoburn, J. (1994). The tin industry since the collapse of the International Tin Agreement, . *Resour Policy 20*, 125-133.

Zaw, K. (1990). Geological, Petrological and Geochemical Characteristics of Granitoid Rocks in Burma: With Special Refernce to the Associated W-Sn mineralization and Their Tectonic Setting. *Journal of Southeast Asia Earth Sciences*, *4(4)*, 293-335.

CHAPTER TWO

2 Overview of the Research Area

2.1 Introduction

According to chapter one, the tin-tungsten metals have been essential for daily usage in life. This chapter gives a brief overview of the history and background of the mines being researched. The Hermyingyi Sn-W deposit is a typical quartz vein type Sn-W deposit that contains about 622,000 t of ore (WO_3+Sn) with a combined grade of 0.35%. The mine was discovered in 1909 with systematic exploration commencing in 1911 and had been a major tin-tungsten producing mine in the world until World War II. The production at the mine spans for more than 100 years. The potential metal production is excellent in this mine with high-grade ore as the evidenced by previous research work. Depending on the mineralogy, different methods of mining have been used such as the cut and fill, the open stoping, the shrinkage, and the room and pillar method

2.2 Study Area and Mineralogy

2.2.1 Study area

The study area (Latitude 15°14´N, longitude 98°21´E) is located approximately 40 km northeast of Dawei Township, Thanintharyi Division, Southern Part of Myanmar as shown in Figure 2.1. Dawei Township about 622.6 km far from Yangon, Myanmar. According to the transportation, it takes 12 hours by car or about 2 hours from Yangon by plane to Dawei, and from Dawei Town it takes about 5 hours by car to the research mine site.

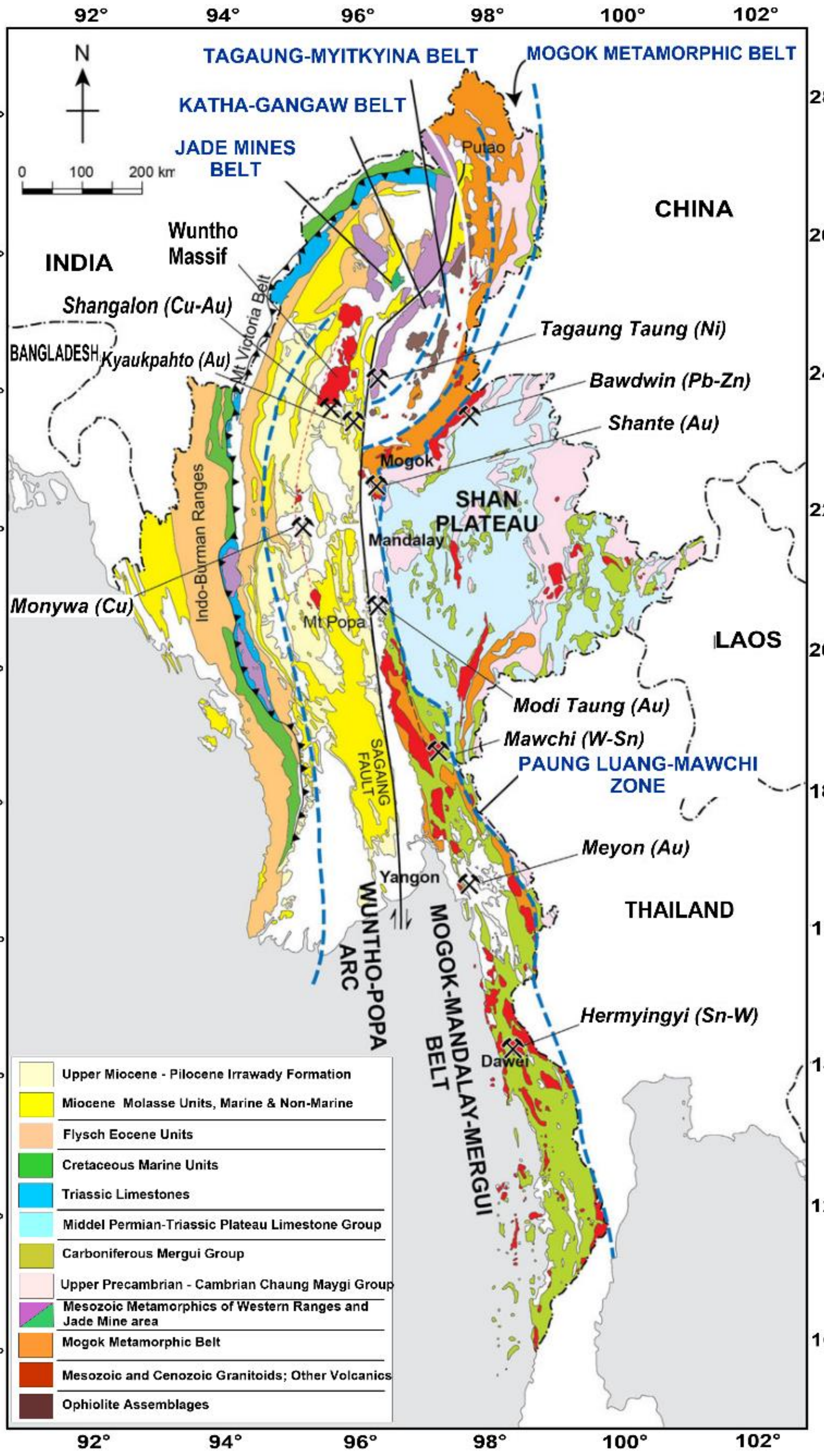


Figure 2.1. Describe the location of the study area.

2.2.2 Mineralogy of research area

Figure 2.2 (modified after (Bender, 1983)) show the Hermyingyi Deposit hosted in a quartz-cassiterite-wolframite vein system which cuts the Hermyingyi granite stock 1400 m in length and 550 m width located just north of the Central Range Granite. The ore-bearing quartz veins are mainly hosted by the Hermyingyi monzogranite which intruded into the Carboniferous metasedimentary rocks of Mergui Series (after (Cobbing, Mallick, Pitfield, & et al, 1986)).

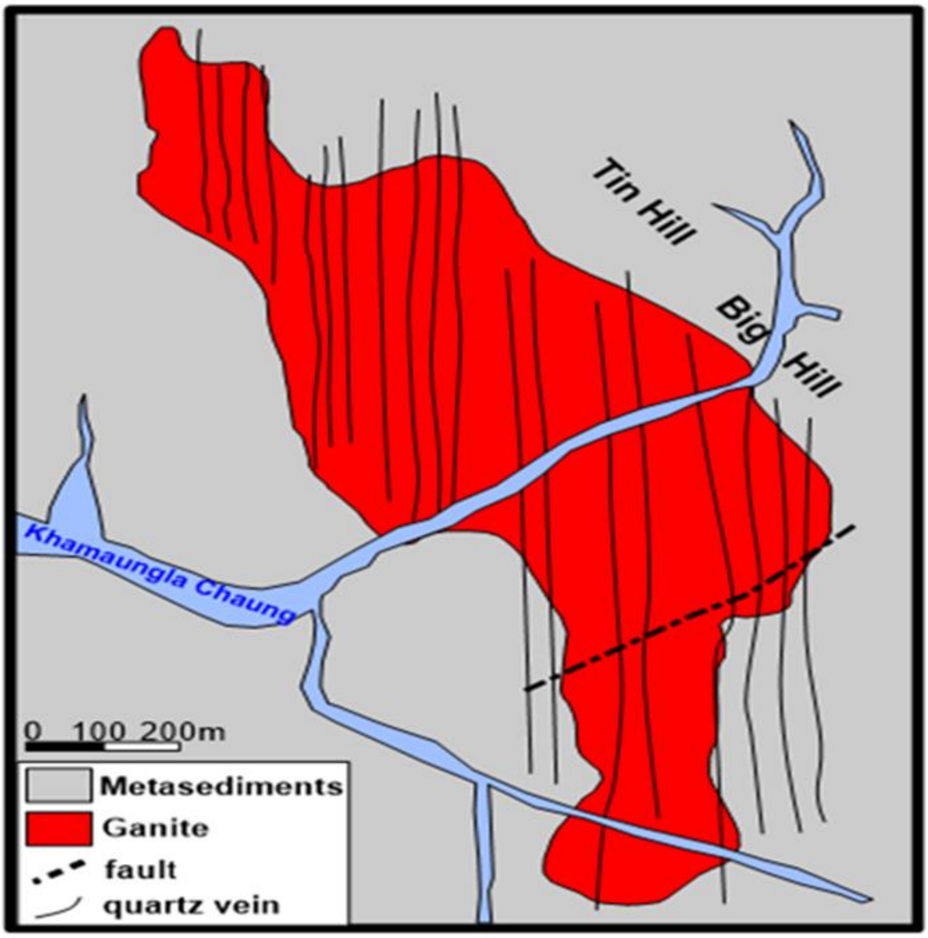


Figure 2.2. Detailed geological map of the Hermyingyi Mine and the main Sn-W vein system simplified after Bender, 1983.

The mined area includes five granite hill-stocks protruding from a pluton that intruded into phyllites, schists, and quartzites of Mergui Group. Tin Hill (Yinthan Taung) in the north is Sn-rich, and Big Hill (Gadin Taung) produce mixed Sn-W concentrates, and Banyan Tree Hill (Naungbin Taung) in the south is W-rich. Quartz veins and marginal greisens contain cassiterite and wolframite in varying proportions and cut the granite and the surrounding envelope rocks. The quartz veins vary in thickness and they have been mined using monitoring, hill sluicing, and aditing.

2.3 Mining Method at Hermyingyi (Sn-W) Deposit Mine

2.3.1 Outcrop mine planning

Figure 2.3 shows the mine plan that represents tin-tungsten metal in this area that is currently being extracted.

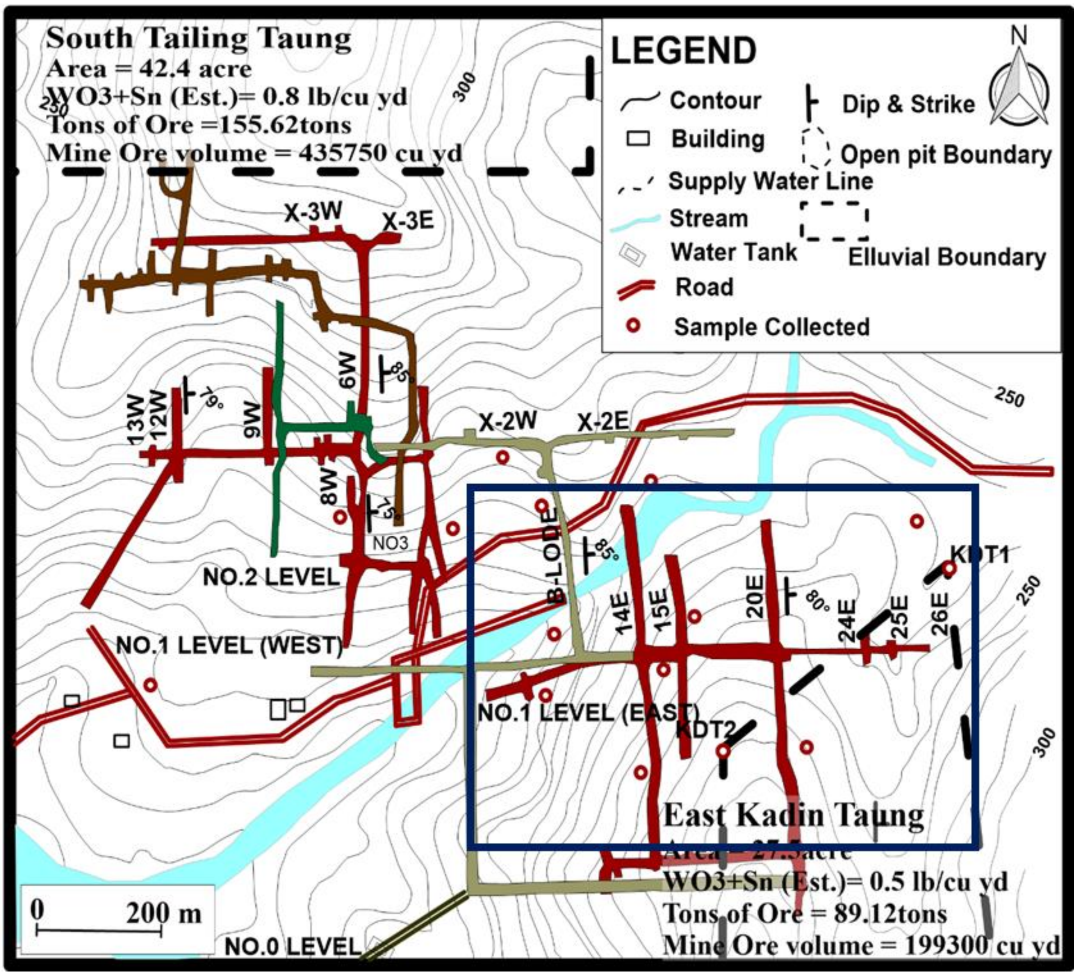


Figure 2.3. Illustrated the planning mine site at Hermingyi Mine (modified after Aung Tun Oo, 2018, unpublished).

Mine planning and methods of excavation are the most important aspects at the ongoing mine site. The deposit has been developed by main crosscuts in two levels, but current mining levels are concentrated on 154 m and 100 m because the adits above 154 m level have all collapsed and become inaccessible (Zaw, 1990). These productive veins strike N-S or NNW-SSE with steep easterly dips of 70°-85°, although some west-dipping veins are occasionally noted as shown in Figure 2.4. The veins vary from several centimeters to 2 m in thickness and some veins can be over 200 m in length. Therefore, the vein trend is associated with the safety of mine excavation in this area. According to the map, the mined-out area is located near the surface. Nevertheless, the plan of excavation is expected to expand to deep underground and width of the mountain since ore reserve is great in this research area.

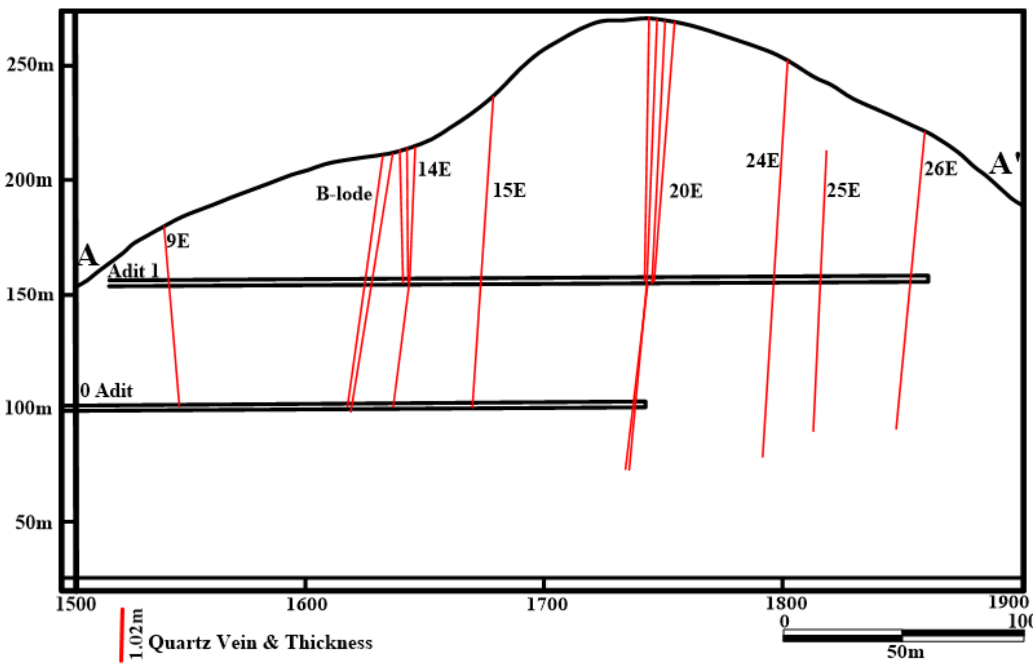


Figure 2.4. Illustrated the cross-section map along A-A' at Hermyingyi (S-W deposit) Mine (modified after Aung Tun Oo, 2018, unpublished).

2.4 Current Mining Method in Study Area

In the meantime, the open stoping method is well developed in the Sn-W Deposit Mine due to economical issues. However, the open-cut method, the open-pit method, the cut and fill method, the room and pillar method, sub-level method, and the shrinkage stoping method has been used somewhere in the underground Sn-W Deposit mine.

Figure 2.5. Configuration of the mining method at study area.

Figure 2.5 shows the configuration of the mining method at the mine site. Among these methods, the open stoping method has been developed in this area. Because of open stoping is easy to extract the metal from the underground. On the other hand, the cut and fill method has been used in some parts of the mine site because the rock condition needs to fill with material for the safety of the mine workers and the transportation of ore.

2.5 Problem Statement of Research Area

The main problem at the study was observed surrounding the excavated area which is related to varying conditions structure like geology, complex structural network, and environmental stress distribution as shown in Figure 2.6. Because the mining area is located at shallow underground surface where high vertical stress can affect the mine. Moreover, the open stoping is applied to extract the metal and there is no support for safe mining operations. The extraction of ore without support is evidently one of the reasons for the instability of the excavated areas. In another section of the mine the support systems have been used to control the collapse but this system is very weak as shown in Figure 2.7 (a).

The other problem is that the in-situ rock strength of the host rock is weaker than the vein (Sn-W deposit) especially in the upper section. Figure 2.7 (b) shows the collapsed rock from the side of the wall at excavated area due to the weak rock condition. This weak rock geological condition causes a big challenge in developing the tin-tungsten deposit mines.

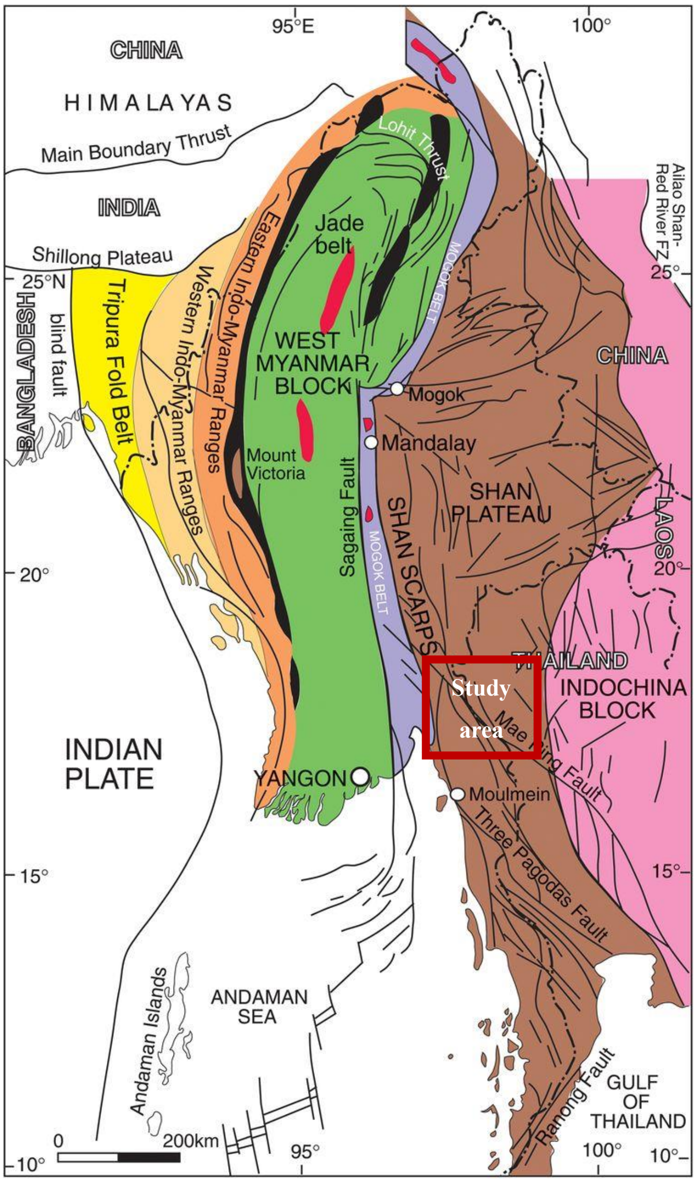
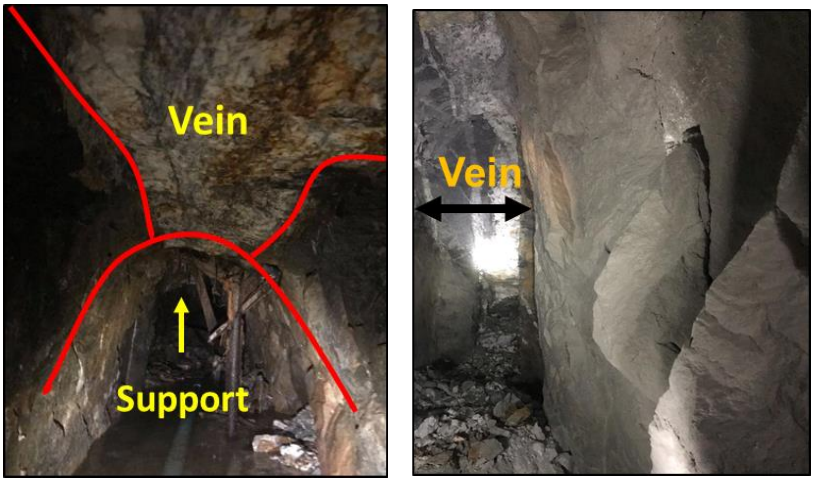


Figure 2.6. Tectonic features of Myanmar and surrounding area (Mitchell, et al., 2007).



(a) (b)

Figure 2.7. The current problem of the mined-out area at the tin-tungsten deposits mine; (a) Poor support system and (b) Collapse of the rock from the side-wall of excavated mine.

2.6 Research Methodology

Figure 2.8 shows the structure of the methodology for this dissertation. The structure of the study is divided into three main parts viz. field study, laboratory test, and numerical simulation.

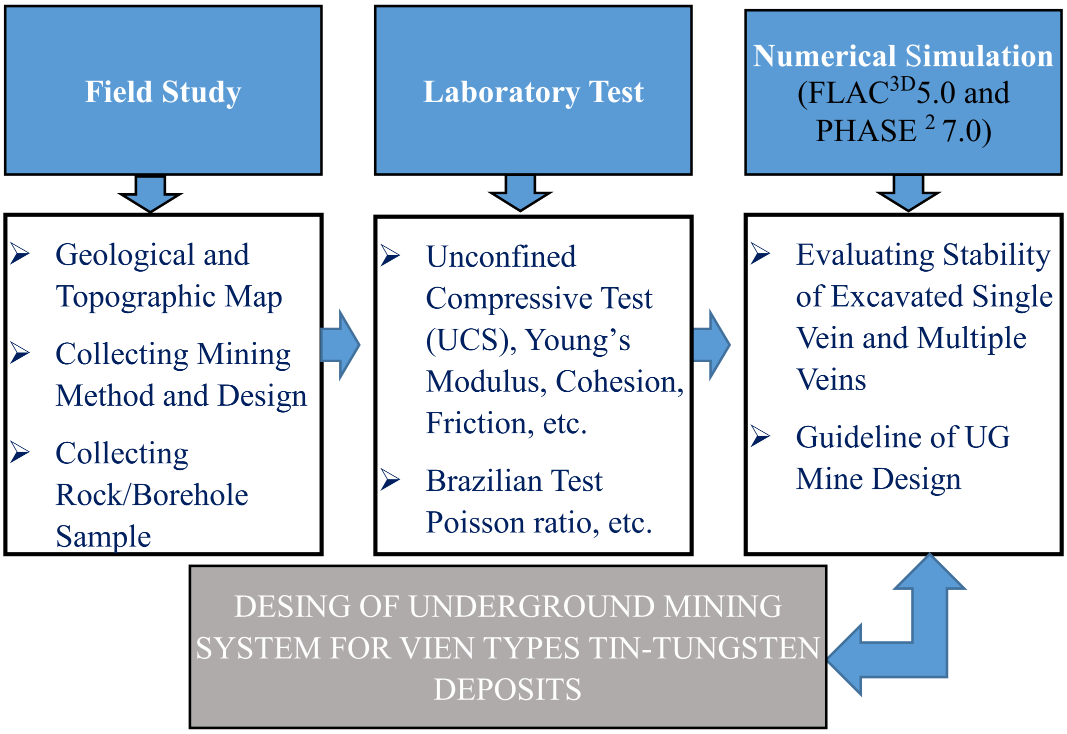


Figure 2.8. Configuration of the flow chart for the research methodology.

2.6.1 Field work

Field study is important for understanding the problem of the mine and actual mine condition such as structural, geological conditions, geotechnical conditions, and mining methods. Therefore, the field study was conducted to comprehend the actual problems encountered at the mine.

2.6.1.1 Field procedure

The field procedures undertake are as below;

1. Site investigation of the current mining condition and its related problems at the mine shown in Figure 2.9 (a): The presence of geological structures such as faults, joints, and bedding planes have been observed near the excavation of the mine. On the other hand, the mining methods were studied in the Sn-W Deposit Mine. The mine method and design at the research area such as the open stoping developed without supporting in weak rock condition zone in underground Sn-W Deposit Mine experienced instability problem.

2. Measurement of the joint sets from the core log box shown in Figure 2.9 (b). Based on the measurements, the Rock Mass Rating (RMR) is calculated to understand the rock properties as shown in Figure 2.12. Depending on these conditions, the rock strength properties have been calculated as Rock Quality Designation (RQD), the Geological Strength Index (GSI), etc. As a result, GSI values have been estimated around 30 surrounding the study area. Therefore, the geotechnical properties are based on GSI value of 30 for all simulation results except for simulations comparing various values of GSI.

3. Measurement of the strength of the in-situ rock in the mine site as shown in Figures 2.9 (c), the initial rock strength test were measured in an underground mine with a potable hand load test. The joints and bedding planes orientations were also measured (see Figures 2.9 (d)).

Figure 2.9. Illustration of the field study in Sn-W deposit mine, (a) Current mining method, (b) Measured the drilling sample, (c) Measured the in-situ strength test, and (d) Measured join and bedding plane in the underground mine.

4. Collecting the rock samples, drill core samples, and the reference of previous work such as the mining plan map, the topographic map, the geological map, and previous work references.

After field study, the rock samples are carried out to the laboratory to obtain the geotechnical properties for the research area.

2.6.2 Laboratory work

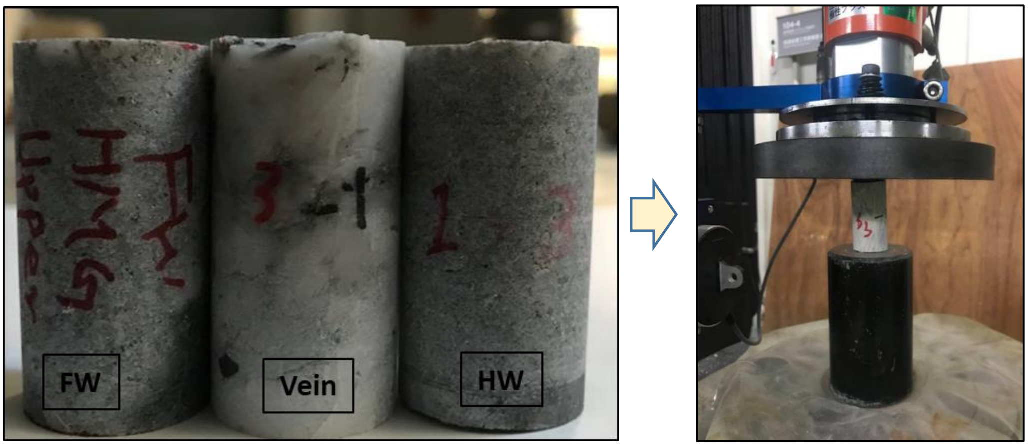
As a previous field study, the laboratory test was performed to determine the geological strength such as compressive strength, tensile strength, compressive strain, cohesion (c), friction (Ø), Young's modulus, unit weight of a collected rock samples, and Poisson's ratio.

2.6.2.1 Test procedure

The Unconfined Compressive Strength Test (UCS) is one of the oldest strength tests for rock mass. In the laboratory, the UCS was used to determine the compressive strength and compressive strain of rock by the Mohr-Coulomb failure criterion as shown in Figure 2.11. The values of strength and strain are high for footwall rock mass than the vein and the hanging wall rock mass. Therefore, the rock failure could be expected to occur above of excavated area at the inclined vein orientation. The rock sample specimen were prepared like logging samples with cutting machine as shown in Figure 2.10 (a). After that specimens were tested for various strengths and a plot of the maximum shear stresses versus the vertical (normal) confining stresses for each of the tests is produced in Figure 2.10. From the plot, a strain-line approximation of the Mohr-Coulomb failure envelope curve can be drawn.

The test was carried out on at least three specimens from a relatively undisturbed rock sample. A specimen is placed vertically under the UCS machine in Figure 2.10 (b) after the load is applied and the strain-induced is recorded at frequent intervals to determine a stress-strain curve for each confining stress. Several specimens are tested at varying confining stresses to determine the rock strength. The test results of each specimen are plotted on a graph with the peak stress on the y-direction and the confining stress on the x-direction.

These data have been essential for estimating the geotechnical engineering parameters for the research. Based laboratory test results, these data were applied in the simulations carried out for investigating the instability of Hermyingyi (Sn-W) Deposits Mine.



(a) (b)

Figure 2.10. (a) Prepared rock specimens after cutting with the machine, and (b) The laboratory tensile strength test.

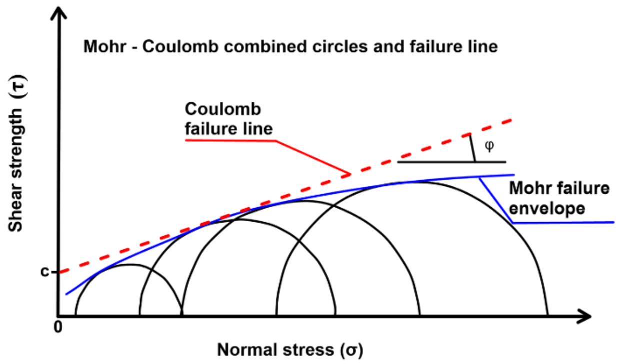


Figure 2.11. The laboratory test results with a plot of strain-line approximation of the Mohr-Coulomb failure comparing maximum shear stresses versus the vertical (normal) confining stresses for each of the tests.

Figure 2.12 shows the laboratory test results compared with compressive strain and compressive strength in the research area. Based on this result, the strong rock mass exists in the footwall which is abnormal condition in the underground mine because it is harder than ore body. This geological condition can be attributed to vein rock samples being affected by microstructures and intact rock degradation. Nevertheless, the strength of hanging wall rock is weaker than the strength of both type the vein and the footwall.

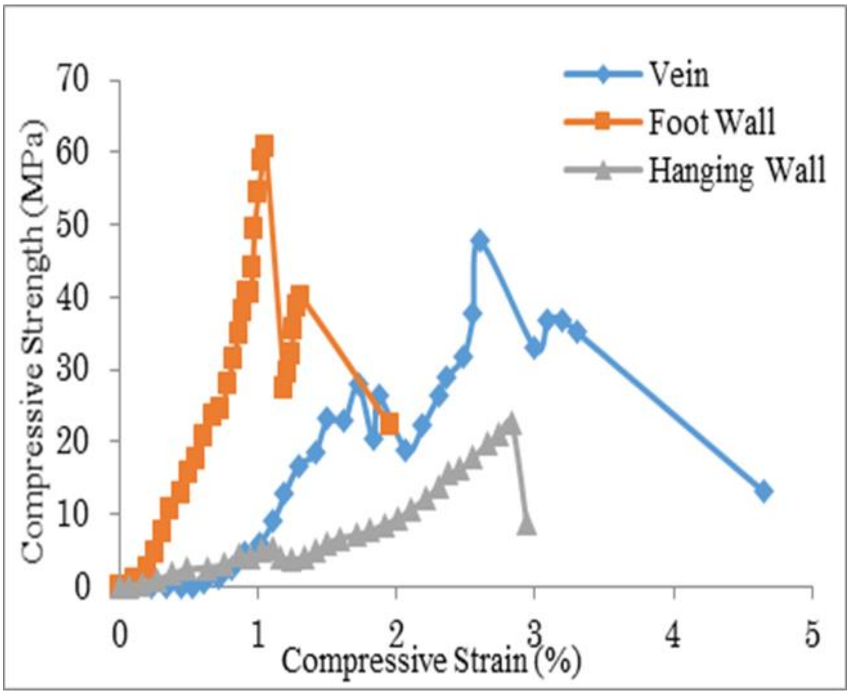


Figure 2.12. Laboratory test results comparing compressive strain and compressive strength in the research area.

2.6.3 Classification of rock mass in research area

2.6.3.1 Rock quality designation, RQD

Figure 2.13 shows the site investigation of rock failure observed near the intersection of the intact rock and joint zone with bore log sample. The result is carried out for evaluating the ROD by using Equations (2.1) and (2.2). The RQD is a standard technique in the mining industry for the quantitative assessment of rock quality and degree of jointing, fracturing, and shearing in a rock mass. It is defined as the percentage of intact drill core pieces as longer than 100 mm recovered during a single core run (Abzalov, 2017). The general equations are as follows;

$$\text{Core Recovery, CR} = \frac{\text{Total length of rock recovery}}{\text{Total core run lenth}} \text{ x } 100\% \qquad (2.1)$$

$$\text{RQD} = \frac{\sum \text{Length of sound pieces} > 100\text{mm}}{\text{Total core run lenth}} \text{ x } 100\% \qquad (2.2)$$

In the mine site, because of highly jointed condition and unconformity in the underground collapse and float of rock mass could. Based on the borehole samples and visual structural analysis, the rock mass condition is weak zone and structures complex. According to Table (2.1), the result of RQD indicate that the rock mas has a range of 20-50 % which means the rock mass at Hermyingyi Mine is fair or poor rock, but rock mass at greater depth the rock mass is strong with RQD values of ≥ 90 % as shown in Figure 2.14.

Figure 2.13. Core log sample box for calculating the RQD at the research area.

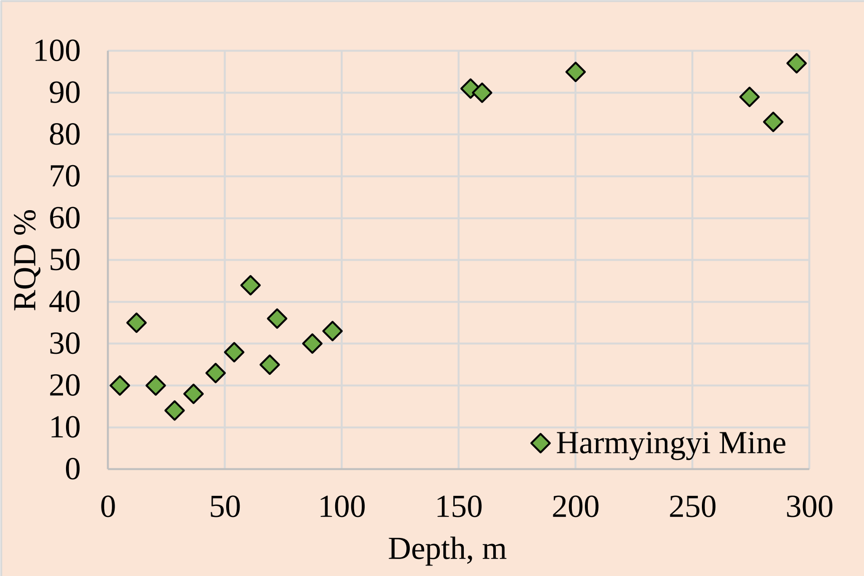


Figure 2.14. Calculated the RQD in the research area.

Table 2.1. Rock Quality and RQD by (Carmichael, 1989).

Rock Quality	**RQD (%)**
Very poor (Completely weathered rock)	<25%
Poor (weathered rocks)	25 to 50%
Fair (Moderately weathered rocks)	51 to 75%
Good (Hard Rock)	76 to 90%
Very Good (Fresh rocks)	91 to 100%

2.6.3.2 Q-system and rock mass rating, RMR

A more complex rock quality characterization of Q-system is defined by Barton et al. (1974) as follows;

$$Q = \mathrm{RQD}\frac{J_r \times J_w}{J_a \times J_n}\frac{1}{\mathrm{SRF}} \tag{2.3}$$

where;

RQD is the rock quality designation

J_a is a parameter depending on degree of joint alteration and clay filling

J_n is a parameter depending on the number of joint sets

J_r is a parameter on joint roughness

J_w is parameter depending on the amount of water flow or pressure

SRF is the stress reducing factor (e.g., due to faulting).

Bieniawski (1989) developed a rock classification system called that rock mass rating, RMR. Six parameters are used to classify a rock mass: uniaxial compression strength, RQD, spacing of discontinuities, condition of discontinuities, orientation of discontinuities, and groundwater condition.

2.6.3.3 Geological strength index, GSI

In this research, the GSI values are necessary for identifying the properties of rock mass to be applied in numerical simulation. The strength of the jointed rock mass depends on the properties of the intact rock pieces and also upon the freedom of the broken pieces to the side and rotate under different stress conditions. This freedom is controlled by the geometrical shape of the intact rock pieces as well as the condition of the surface separating the pieces. Angular rock pieces with clean, rough discontinuity surfaces will result in a much stronger rock mass than one which contains rounded particles surrounded by weathered and altered materials.

The influence of blast damage on the near-surface rock mass properties has also been taken into account in the 2002 version of the Hoek-Brown criterion discussed by Hoek et al. (2002) as follows;

$$m_b = m_i \exp\left(\frac{\text{GSI-100}}{\text{28-14D}}\right) \tag{2.4}$$

$$s = \exp\left(\frac{\text{GSI}-100}{9-3\text{D}}\right) \tag{2.5}$$

$$a = \frac{1}{2} + \frac{1}{2}\left(e^{-\text{GSI}/15} - e^{-20/3}\right) \tag{2.6}$$

The GSI, introduced by Hoek (1994) and Hoek et al. (1995) provides a number of different geological conditions by reduction in rock mass strength. This system is shown in Table 2.2, for blocky masses, and Table 2.3, for heterogeneous rock masses. During the early year of application of the GSI system, the value of GSI was estimated directly from RMR. However, this correlation has proved to be unreliable, particularly for poor quality rock masses and for rocks with lithological peculiarities that cannot be accommodated in the RMR classification. Consequently, it is recommended that GSI should be estimated directly by means of the charts presented in Table 2.2 and Table 2.3 and not from the RMR classification system.

Table 2.2. Characterization of blocky rock masses on the basis of interlocking and joint conditions.

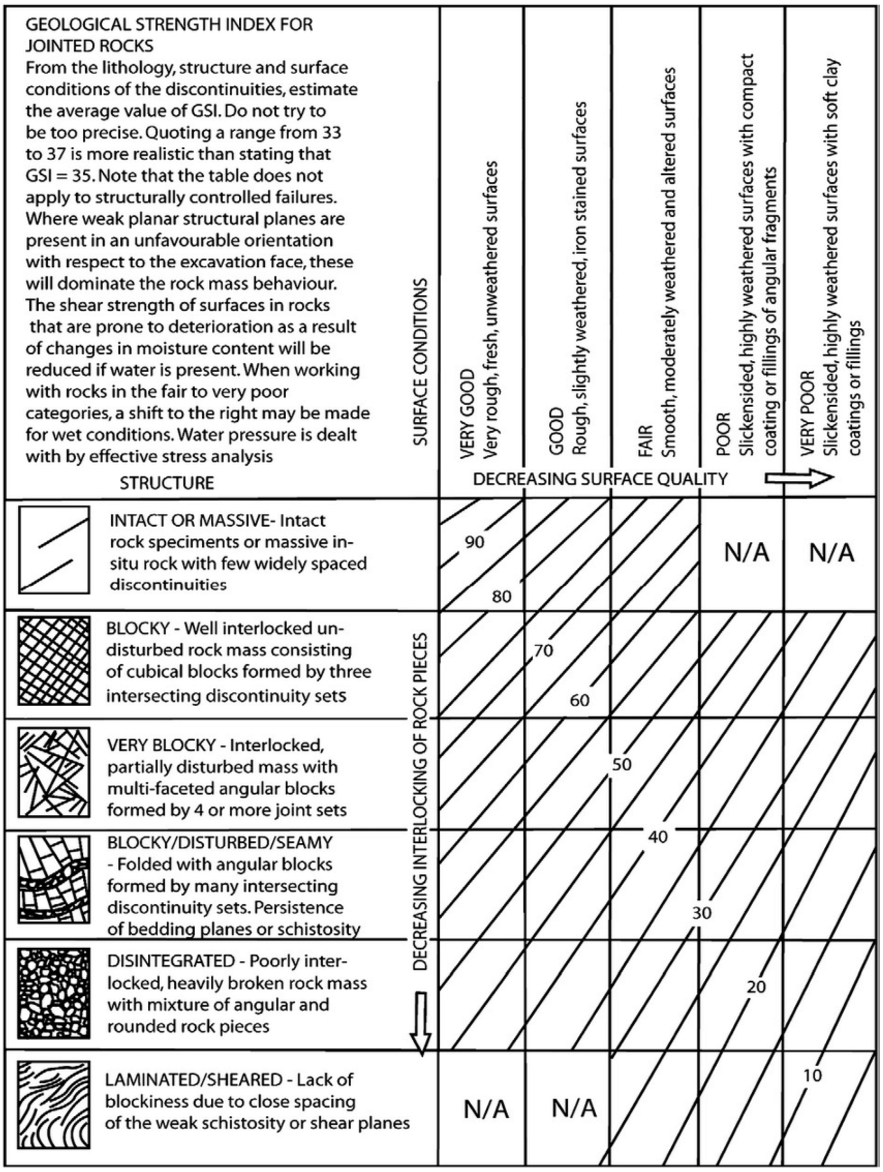
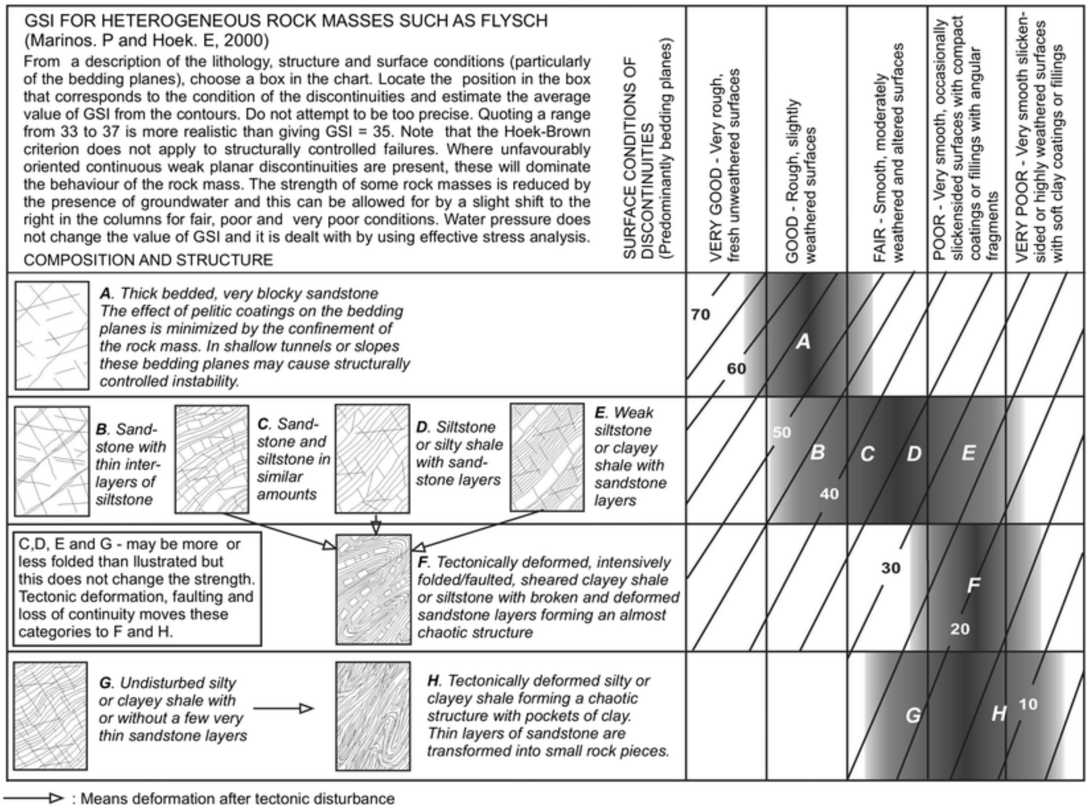


Table 2.3. Estimate of Geological Strength Index GSI for heterogeneous rock masses such as flysch. (After (Marinos & Hoek, 2021)).



D is a factor that depends upon the degree of disturbance due to blast damage and stress relaxation. It varies from 0 for undisturbed in-situ rock masse to 1 for very disturbed rock masses. Guidelines for the selection of D are presented in Table 2.4. Note that factor D applies only to the blast-damaged zone and should not be applied to the entire rock mass. For example, in tunnels, the blast damage is generally limited to a 1 to 2 m thick zone around the tunnel and this should be incorporated into numerical models as different and not the entire surrounding rock mass. Applying the blast damage factor D to the entire rock mass is inappropriate and can result in misleading unnecessarily pessimistic results.

Table 2.4. Guidelines for estimating disturbance factor D.

Appearance of rock mass	Description of rock mass	Suggested value of D
	Excellent quality controlled blasting or excavation by Tunnel Boring Machine results in minimal disturbance to the confined rock mass surrounding a tunnel.	$D = 0$
	Mechanical or hand excavation in poor quality rock masses (no blasting) results in minimal disturbance to the surrounding rock mass. Where squeezing problems result in significant floor heave, disturbance can be severe unless a temporary invert, as shown in the photograph, is placed.	$D = 0$ $D = 0.5$ No invert
	Very poor quality blasting in a hard rock tunnel results in severe local damage, extending 2 or 3 m, in the surrounding rock mass.	$D = 0.8$
	Small scale blasting in civil engineering slopes results in modest rock mass damage, particularly if controlled blasting is used as shown on the left hand side of the photograph. However, stress relief results in some disturbance.	$D = 0.7$ Good blasting $D = 1.0$ Poor blasting
	Very large open pit mine slopes suffer significant disturbance due to heavy production blasting and also due to stress relief from overburden removal. In some softer rocks excavation can be carried out by ripping and dozing and the degree of damage to the slopes is less.	$D = 1.0$ Production blasting $D = 0.7$ Mechanical excavation

Finally, according the evaluation of RQD result and RMR classification of mine, the GSI values is 30, it can be assumed in this mine area. Nevertheless, the rock mass condition is very essential for mine stability.

2.6.4 Numerical simulation

To meet the objective of this research, two kinds of simulations were used for the instability of underground mines. In which, FLAC^{3D}5.0, and Phase^{2} 7.0 simulations were applied to obtain the research results. FLAC^{3D} applies the finite difference method code which is based on zone elements characterized by the three-dimensional grid. Linear and nonlinear law of stress/strain are driven to allow each element behave responding to the boundary condition. This simulation is popular among mining companies because the simulation is useful for understanding the underground mine stability in the world. The simulation was built by Itasca company for the mining industry. The other simulation is Phase^{2} simulation, which is a two-dimensional plastic finite element grid for calculating stresses and displacements around underground openings, and can be used to solve a wide range of mining and civil engineering problems. In this dissertation, both simulations were applied for the stability of stope at underground Sn-W Deposits Mine.

2.7 Dissertation Objectives

According to the perspective given in problems statement, the objectives of the dissertation are listed below;

1. Understanding stress conditions around the stope, the crown pillar, and the sill pillar area of the open stoping method comparing with the cut and fill method at shallow depth under various conditions like geology, excavation sequence, vein geometry, and stress ratio.
2. Investigating effectiveness approach of single vein excavation due to several conditions of mine environment such as stress ratio, geology, the step of excavation, vein dipping, sill pillar thickness, and vein width by comparing the open stoping and the cut and fill method.
3. Controlling the induced stresses of the crown pillar at the shallow

underground by using countermeasure and adjusting the thichness of the crown pillar under different geological condition and vein geometry.

4. Evaluating the instability of the open stoping and the cut and fill methods at multiple veins excavation and optimizing of the induce stresses with countermeasure method due to geological condition, stress ratio, vein inclination, the width of vein, location of the mined-out area, order of vein excavation, and sequences of stope excavation.
5. Developing guidelines for the underground mine design for Sn-W deposit after analyzing the several results of the research study.

2.8 Conclusions

In this chapter, the overall research background and plan has been briefly explained. The Hermyingyi (Sn-W) Deposits Mine was discovered since 1890 before World War II by the British. The mine has a high grade and largest tin-tungsten deposit in the world which is important to Myanmar's economic development. Therefore, a good mining method is important for the stability of the excavated area. Currently, the occurrence of the instability problems in the method of open stoping being applied is related to regional geological condition and structures. Therefore, the Rock Mass Classification (RQD) is important to understand the condition of rock in which RQD, Q-system, and RMR were used to evaluate the geological properties. Ultimately, numerical simulation will be conducted to developing guidelines for the underground mine design for Sn-W deposit.

2.9 References

Abzalov, M. (2017). *Applied Mining Geology*. https://doi.org/10.5382/econgeo.112.6.br02: Economic Geology 112(6): 1543-1544.

Barton, N. R., & et al. (1974). Engineering Classificaion of Rock Masses for the Desing of Tunnel Support. *Rock Mechanics and Rock Engineering 6(4)*, 189-236.

Bender, F. (1983). Geology of Burma; Gebruder Borntraeger. *Berlin/Stuttgart, Germany*, 293.

Bieniawski, Z. T. (1989). *Engineering rock mass classifications: a complete manual for engineers and geologists in mining, civil, and petroleum engineering.* New Jersey: Wiley.

Carmichael. (1989). *Physical Properties of Rocks and Minerals.* Milton Park: Routledge.

Cobbing, E., Mallick, D., Pitfield, P., & et al. (1986). The Granites of the Southeast Asia Tin Belt. *Journal of the Geological Society, 143(3)*, 537-550.

Hoek, E. (1994). Strength of Rock Mass and Rock Masses. *ISRM News Journal, 2*, 4-16.

Hoek, E., Carranza-Torres, C. T., & Corkum, B. (2002). Hoek-Brown Failure Criterion - 2002 Edition. *Proceedings of the 5th North American Rock Mechanics Symposium, 7-10 July*, (pp. 267-273). Toronto.

Hoek, E., Kaiser, P. K., & Bawden, W. F. (1995). *(1995) Support of Underground Excavation in Hard Rock. Balkema, Rotterdam.* Rotterdam: Balkema.

Marinos , P., & Hoek, E. (2021). Estimating the geotechnical properties of heterogeneous rock masses such as flysch. *Bull Eng Geol Environ 60*, 85-92.

Mitchell, A., Htay, M. T., Htun, K. M., Win, M. N., Oo, T., & Hlaing, T. (2007). Rock relationships in the Mogok Metamorphic belt, Tatkon to Mandalay, central Myanmar. Journal of Asian Earth Sciences, 29, 891–910.

Zaw, K. (1990). Geological, Petrological and Geochemical Characteristics of Granitoid Rocks in Burma: With Special Refernce to the Associated W-Sn mineralization and Their Tectonic Setting. *Journal of Southeast Asia Earth Sciences, 4(4)*, 293-335.

CHAPTER THREE

3 Assessment of the Stability of Underground Mine at (Sn-W) Deposits on Single Vein

3.1 Introduction

The rock mass strength has been evaluated in the previous chapter by using rock mass classification such as Rock Quality Designation (RQD) and Geological Strength Index (GSI). The results showed that the geological strength index of the rock mass is characterized by the GSI value of 30. The ore, which hosted in the vein lies almost vertically inclined in Hermyingyi (Sn-W) Deposit Mine as shown in Figure 3.1.The mineralized veins strike N-S or NNW-SSE with dips ranging between 70°-85°. Occasionally some west-dipping veins are found. The veins vary from several centimeters to 2 m in thickness and some veins can be over 200 m in length. The main focus in this chapter is on the excavated vein highlighted in blue as presented in Figure 3.1. As previously presented in Figure 2.11, the results of geological strength by a laboratory test demonstrated that the rock strength is weak in the hanging wall domain. Additionally, the results indicated that the vein is stronger than the hanging wall rock but it is weaker than the footwall. According to the geological conditions, the hanging wall is likely to be exposed to environmental impacts like ground subsidence if the underground mine is developed at shallow depth. Figure 3.2. shows that failures occurred surrounding the excavated stope in the study area. This is because the veins emplaced are narrow and the surrounding rock mass is weak due to geological structures, thus the ground and rock collapse occurs when the vein is being excavated. Currently, the Hermyingyi (Sn-W) Deposits Mine is being developed not only by single vein excavation but rather the multiple vein excavation with open stoping method. Before discussing the multiple vein extraction, this chapter explores the stability conditions of single vein stope excavation under the various regimes of regional stress, varying geological and structural conditions, and excavation sequences. According to the problems of instabilities currently being encountered in the mine, this chapter aims to provide pragmatic solutions by understanding the key parameters that cause such instabilities by using FLAC^{3D} V5.0 simulation.

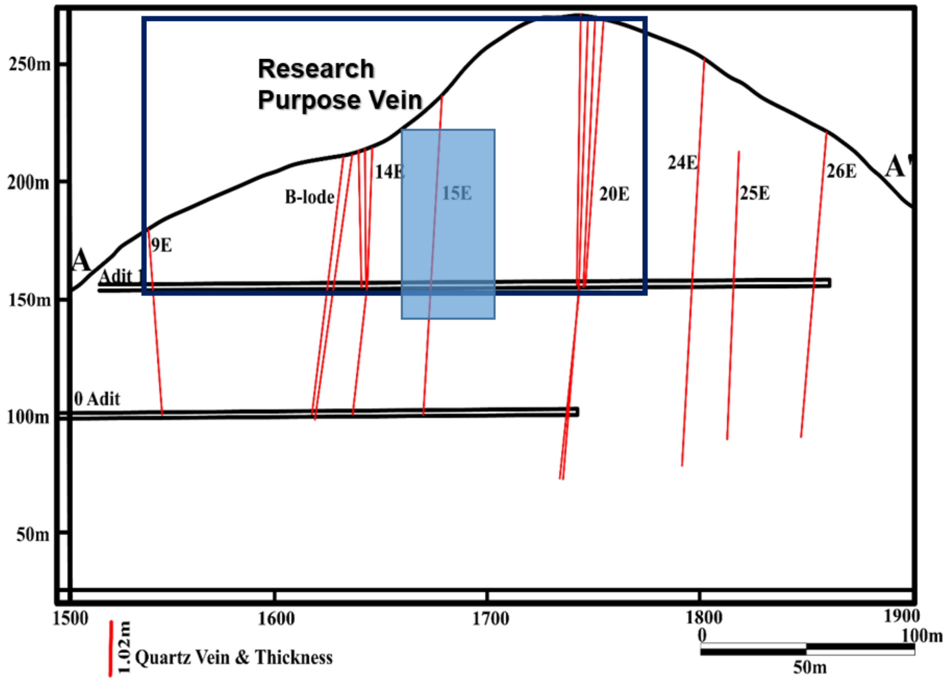


Figure 3.1. Shows the analyzed vein location of mine at the cross-section map along A-A´ the East-West direction.

Figure 3.2. Current problem statement of Hermyingyi (Sn-W) Deposit Mine: (a) The rock collapse surrounding of the excavated area and (b) Narrow inclined vein (nearly N-S direction) in the host rock.

3.2 Constructed Model

To effectively study the stope stability at Sn-W Deposit Mine, several numerical models were created in FLAC^{3D}V5.0 with different inclined vein angles, various excavated depths, and different mining sequence. The first model was constructed with dimensions of 200 m x 200 m where x-direction, y-direction, and z-direction are fixed at the bottom, x-direction and y-direction are fixed on the sides and the top part of the model is freed as shown in Figure 3.3. This model is created based on the current mine cross-section plan map in Figure 3.1. Stopes starting from 120 m, and 130 m depth are filled up by waste rock in the excavated zone between the in-situ stope pillars (5 m of sill pillars) left during the ore extraction. In another case, the excavated areas in the stopes starting from 120 m and 160 m are without fill by leaving 5 m sill pillars due to narrow ore body. For mining, this section focuses on a single vein, with a circumference of 40 m x 50 m, which is divided into two excavated areas. Each area is excavated in eight steps, and the stopes were excavated by 5 m x 2 m stopes up to 40 m height. The first sequencing of mining starts from the upper part of the mine area and then the lower one. The monitoring points were installed at the top of each stope. The modified parameters were the geological conditions, mining sequence, the vein width, and stress ratio while comparing the currently applied open stoping method and the author's proposed cut and fill method. To optimize the stability of the mine, different supporting systems were also evaluated to judge their efficiency and economic viability. The rock failure mechanism, the factor of safety, and the displacement of the rock mass were determined for the mine stability guideline.

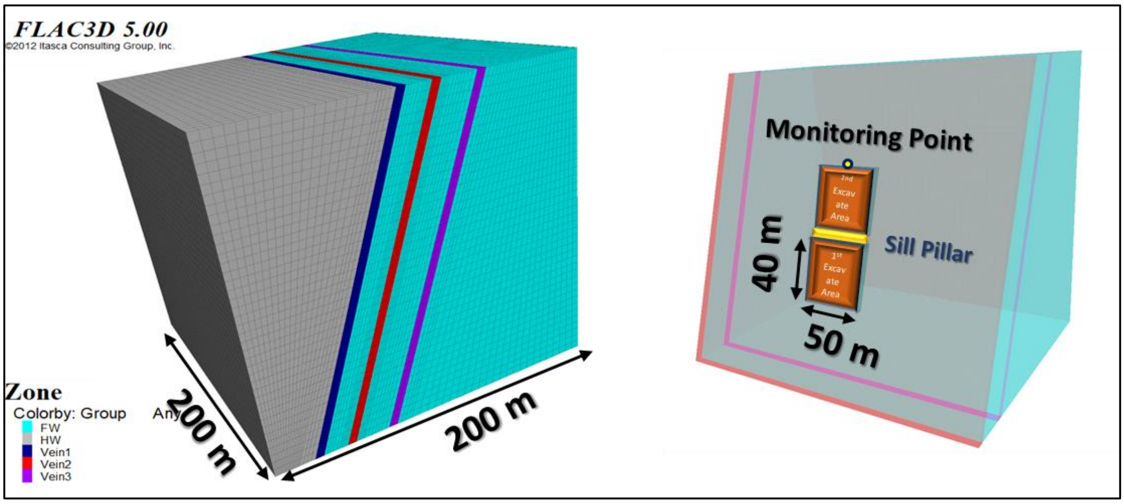


Figure 3.3. The constructed model for single vein by FLAC^{3D} V5.0 simulation.

The explanation of failure conditions namely **"none"** indicates no-failure zone, "**shear-n" indicates** the regions failed under shear load and failure process is still in progress, "**shear-p" indicates** the region failed under shear loading and failure process is stopped due to reduced amount of shear forces, "**tension-n"** means the region failed and tensile loading failure process is still in progress, and "**tension-p"** explains the regions failed under tensile loading and failure process is stopped due to reduced amount of tensile forces (Yasitli & Unver, 2005).

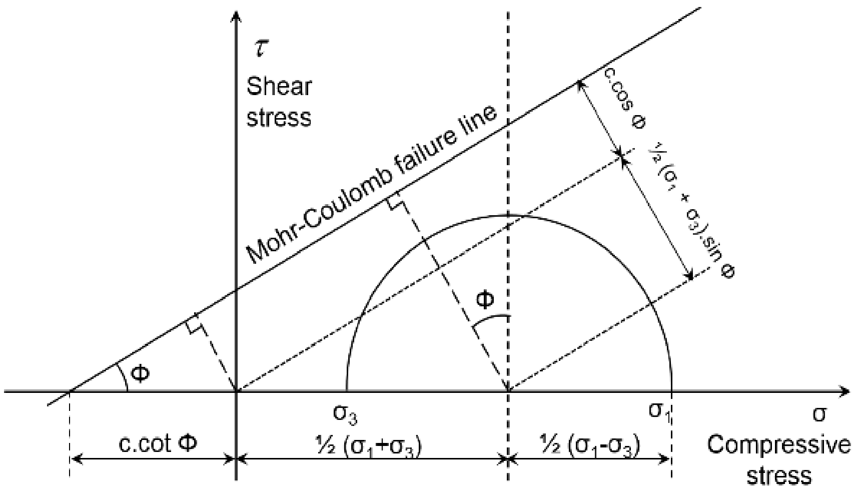


Figure 3.4. The factor of safety by Mohr-Coulomb failure envelope.

The factor of safety of a certain point in the rock can be estimated using the Mohr-Coulomb failure criterion with the availability of major and minor principal stress at that point (Kwon, Cho , & Lee, 2013). Figure 3.4 depicts the system to calculate the factor of safety using the failure criterion presented in Equation 3.1.

$$\text{Factor of safety} = \frac{\text{Rock Strength}}{\text{Induced Stress}} = \frac{\left[\frac{\sigma_1+\sigma_3}{2} + \frac{C}{\tan\varphi}\right]\sin\varphi}{\frac{\sigma_1-\sigma_3}{2}} \tag{3.1}$$

Equation 3.1 shows the formula for estimating the factor of safety. The equation presents the ratio of strength divided by stress where σ_1 and σ_3 are major and minor principal stresses, C is cohesion and φ is friction angle. The factor of safety of ≥ 1.3 is applied as a benchmark for temporary mine opening (Hoek, Kaiser, & Bawden, 1995).

3.3 Effect of the Instability Single Vein Stope Under Various Mine Condition

When the stress risks to a project warrant it, in-situ stress must be measured. As the stress-induced risks increase, it can have an influence on the strength of the rock mass. Thus this section evaluated this phenomenon by FLAC^{3D}5.0 simulation. In this case, the stress ratios (0.5, 1.0, and 1.5), GSI values (ranging from 20 to 55), vein inclined (ranging from 60° to 90°), and stope width ($\geq$ 2 m) are applied in the model. However, in the actual mine condition the geological condition is characterized by a GSI value of 30 and the inclined vein dips at 80°.

3.3.1 Stress ratios

The relationships of vertical stress (σ_v) and horizontal stress(σ_h) is defined as the stress ratio (K). The magnitudes and direction of in-situ stress and induced stresses is an essential component of ground excavation design. In many cases, the strength of the rock is exceeded and the resulting instability can have serious consequences on the behavior of the excavation (Hoek & Brown, 1978). Because the rock at depth is subjected to stresses resulting from the weight of overlying strata and locked-in stresses of tectonic. Therefore, the stress ratio has been considered under different stress ratios such as K = 0.5, K = 1.0, and K = 1.5 for the instability of mine excavation at a single vein. The simulations have been calculated by comparing the open stoping and the cut and fill. Figure 3.5 shows the simulation result of the failure zone by comparing the various stress ratios and the mining methods.

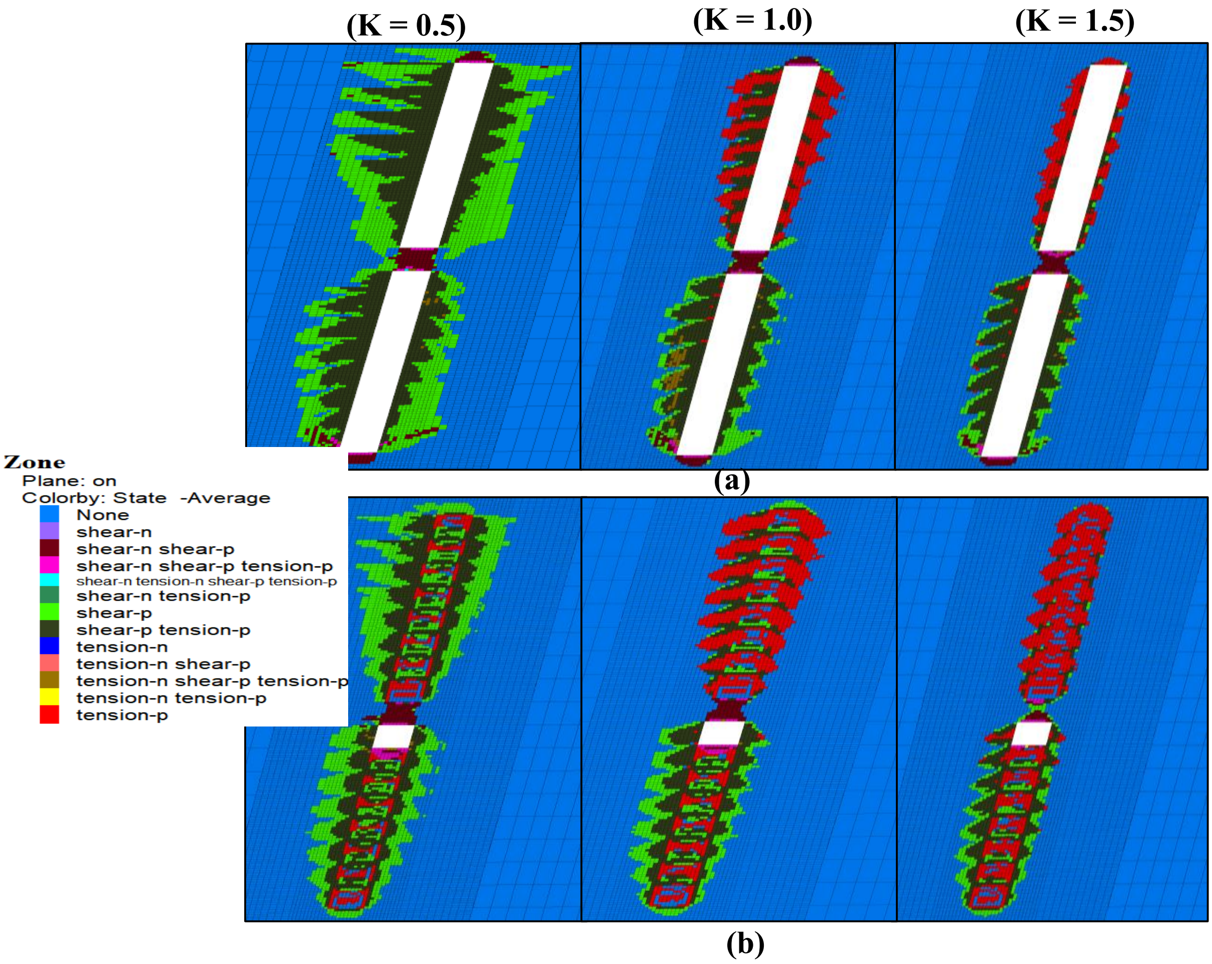


Figure 3.5. The failure result of the model with the application of various stress ratios: (a) The open stoping method and (b) The cut and fill method at single vein excavation.

The results show that in open stoping method the unstable states area of rock failure are dramatically increased at different stress ration as ≤ 10 m failure extent at K = 0.5, ≤ 6 m failure extent at K = 1.0, and ≤ 5 m failure extent at K = 1.5 as shown in Figure 3.5 (a) compared to ≤ 8 m failure extent at K =0.5, ≤ 5 m failure extent at K = 1.0, and ≤ 4 failure extent at K = 1.5 in cut and fill method as presented in Figure 3.5 (b). By comparing the K ratios, at K = 0.5 the potential failure extent is severer than the other rations namely K = 1.0 and K = 1.5. Furthermore, at K = 0.5 the instability is high in the upper mine excavated area while at K = 1.0 and K = 1.5 the instability is high in the lower mine excavated area, this is because the gravitational loading effect is high at shallow depth. On the contrary, at K = 1.0 and K = 1.5 the instability is high in the lower excavated area because of stress redistribution as the excavation extends to a greater depth.

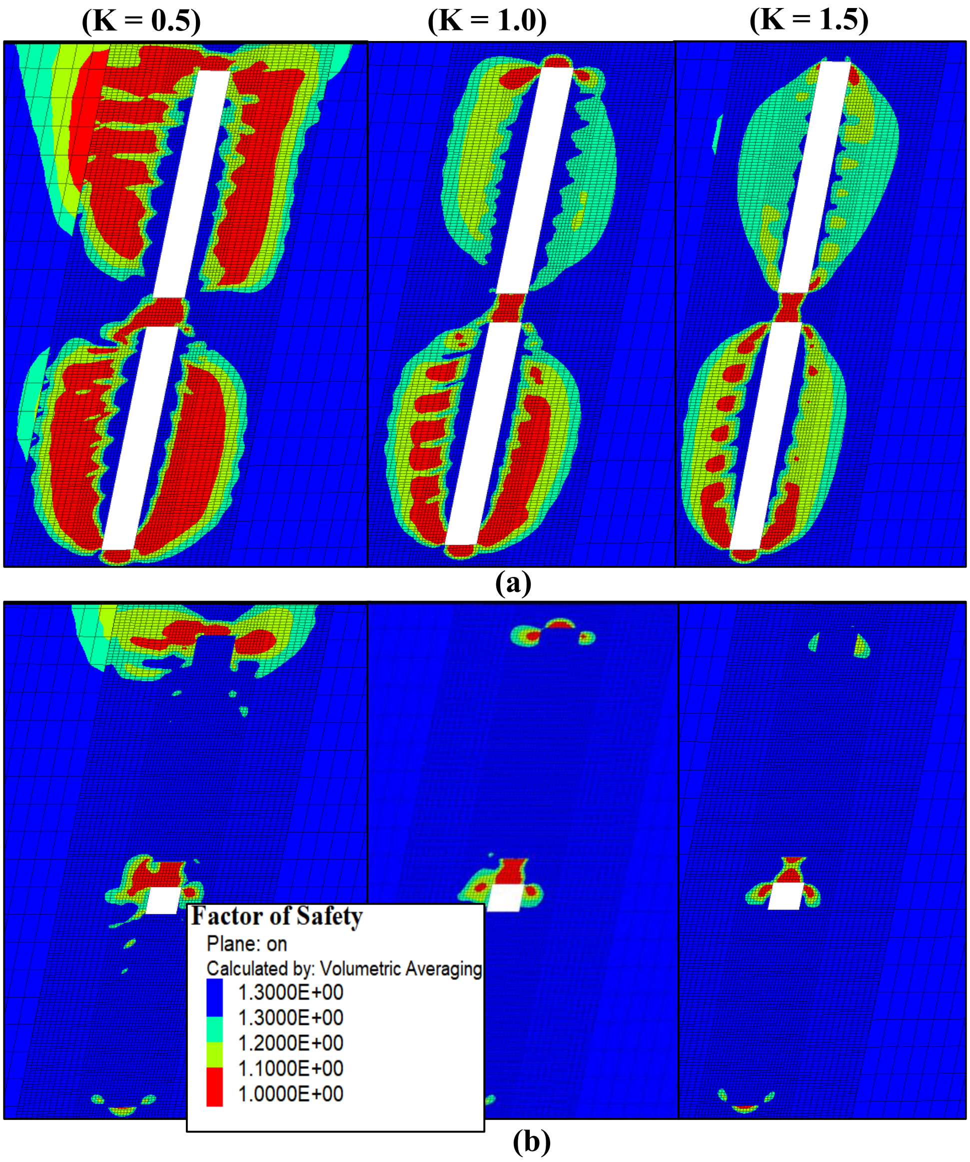


Figure 3.6. The Factor of safety result of the model with application of various stress ratios: (a) The open stoping method and (b) The cut and fill method at single vein excavation.

To further deterministically evaluate the stability conditions, the factor of safety has been calculated by comparing the open stoping method and the cut and fill method under various stress ratios as shown in Figure 3.6. The factor of safety has been identified by the Mohr-Coulomb failure criterion. The open stoping method results show that the instability is very high demonstrated by an expansive potential failure region compared to the cut and fill method where the instability has drastically reduced with the failure region concentrated around the last excavated stope and around the crown pillar as presented in Figure 3.6 (a) and Figure 3.6 (b) respectively. With respect to K ratio, the results reveal that K = 0.5 induces high instability in both mining methods than K = 1.0 and K = 1.5. The lowest instability condition is observed to be at K = 1.5. Additionally, in the open stoping method the factor of safety is low all around the excavated area, sill pillar and the crown pillar while in cut and fill method the factor of safety is low mainly in the crown pillar and the sill pillar.

3.3.2 Geological strength index

The performance of stope stability was also evaluated with respect to rock mass properties determined by the geological strength index parameters. Different GSI values were simulated ranging from 20, representing disintegrated rock mass with poor joint surface quality, to 55, representing blocky rock mass with good joint surface quality. The rock mass properties were determined for each GSI by using Hoek and Brown Failure Criterion. Table 3.1. presents summarized properties of the rock mass. The primary data set which included UCS, BTS were collected from the laboratory test results and then additional data set was generated with RocLab software using the GSI values. The parametric analyses were then carried out for different geological conditions and comparison of the open stoping method and the cut and fill method was done.

Figure 3.7 shows the result of the stability conditions of the excavated stope with different rock mass properties in both the open stoping method and the cut and fill method. The results of the failure state with respect to GSI demonstrate that GSI of 20 records very large potential failure region than the subsequent GSI values up to GSI of 55. As expected, the potential failure region is the smallest at the highest GSI value of

55. This indicates that the rock mass strength has a significant influence on the stability of stopes in underground mine excavation. Comparing the two methods, it is observed that open stoping method is more vulnerable to instability than the cut and fill method. In the open stoping method, the potential failure extent ranges between 10 m and 15 m both in the footwall and hanging at GSI values of 20, 25, 30, 38, and at GSI values of 46, 50, 55 the potential failure extent ranges between 5 m to 8 m, while in the cut and fill method, the potential failure extent ranges between 5 m and 10 m both in the footwall and hanging at GSI values of 20, 25, 30, 38, and at GSI values of 46, 50, 55 the potential failure extent ranges between 2 m to 5 m.

Table 3.1. Rock mass properties evaluated with geological conditions using RocLab software.

Mechanical Properties	GSI Values	Granite, FW	Limestone, HW	Vein
Uniaxial Compressive Stress (MPa)	GSI 20	0.018	0.010	0.398
	GSI 25	0.026	0.014	0.018
	GSI 30	0.012	0.010	0.013
	GSI 38	0.021	0.019	0.023
	GSI 46	0.039	0.034	0.043
	GSI 50	0.168	0.092	0.115
	GSI 55	0.245	0.134	0.168
Young's Modulus (MPa)	GSI 20	1520	1124	1257
	GSI 25	2026	1499	1676
	GSI 30	2702	2000	2236
	GSI 38	4282	3170	3543
	GSI 46	6787	5024	5617
	GSI 50	8544	6324	7071
	GSI 55	11394	8434	9429
Poisson's ratio	GSI 20	0.22	0.23	0.22
	GSI 25	0.22	0.23	0.22
	GSI 30	0.22	0.23	0.22
	GSI 38	0.22	0.23	0.22
	GSI 46	0.22	0.23	0.22
	GSI 50	0.22	0.23	0.22
	GSI 55	0.22	0.23	0.22
Friction angle (°)	GSI 20	37.97	33.58	35.20
	GSI 25	40.20	35.70	37.37
	GSI 30	51.96	43.67	45.34
	GSI 38	54.33	46.26	47.92
	GSI 46	56.44	48.61	50.24
	GSI 50	48.28	43.91	45.57
	GSI 55	49.47	45.25	46.86
Cohesion (MPa)	GSI 20	0.443	0.357	0.387
	GSI 25	0.520	0.414	0.449
	GSI 30	0.820	0.581	0.628
	GSI 38	0.955	0.679	0.736
	GSI 46	1.101	0.784	0.855
	GSI 50	0.990	0.730	0.818
	GSI 55	1.169	0.835	0.940

Due to the difference in the geological strength of the rock mass, the footwall side has a relatively small potential failure region than the hanging wall side (see Figure 3.7). As previously evaluated under rock mass characterization, the hanging wall rock mass is weak and may be affected by high number of cracks, and joints.

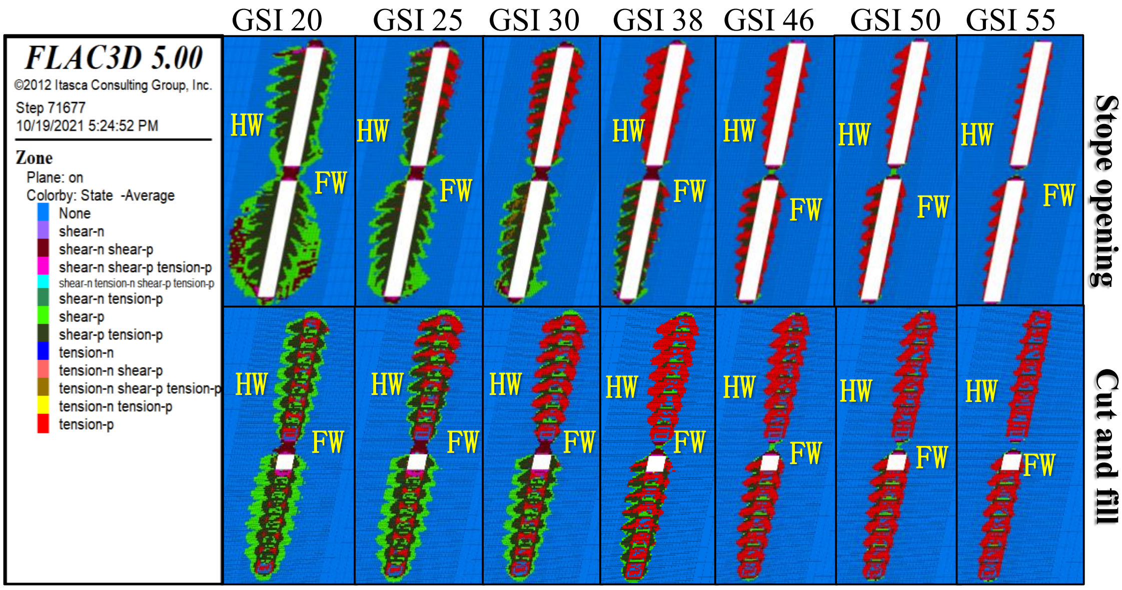


Figure 3.7. The failure result of the model with application of various GSI values for single vein excavation.

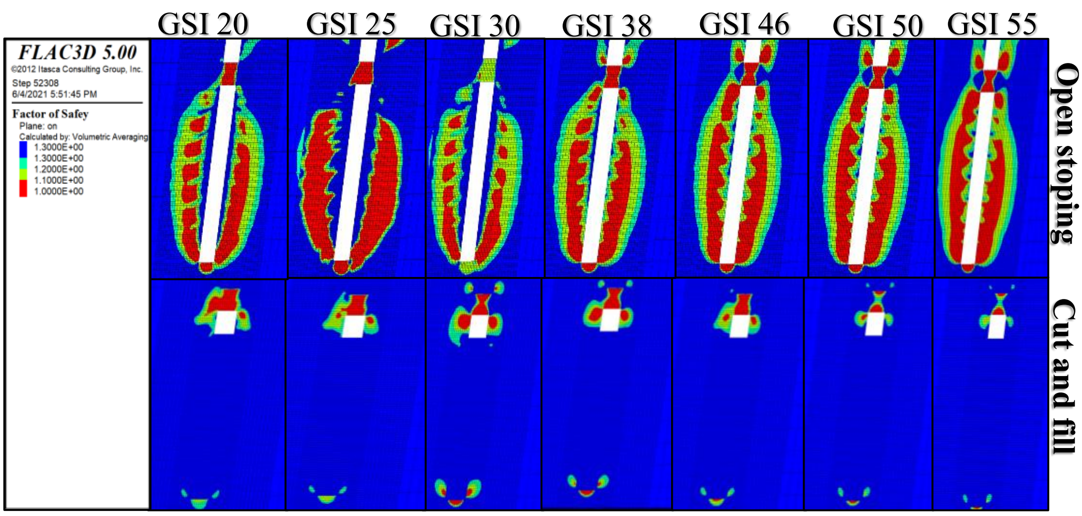
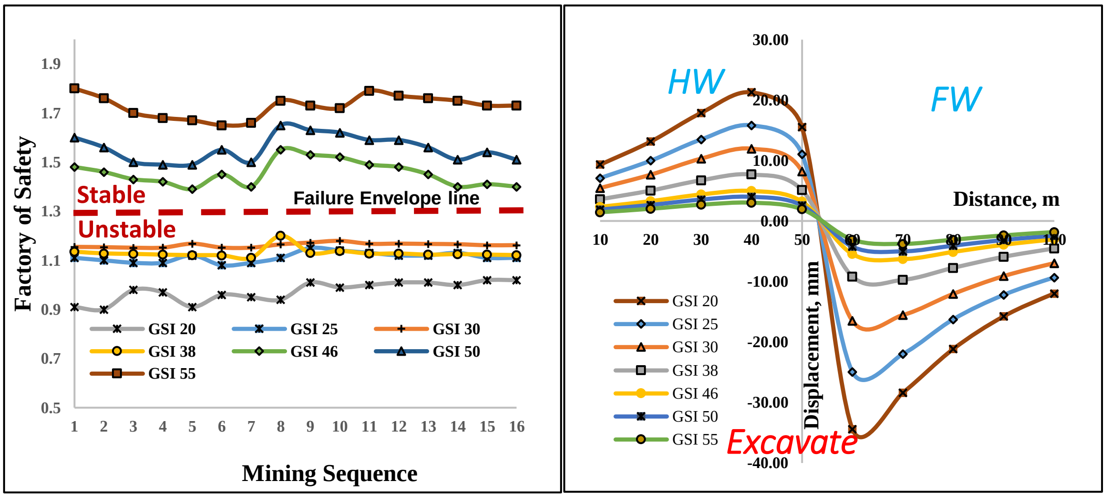


Figure 3.8. The factor of safety results of the model with application of various GSI values for single vein excavation.



(a) (b)

Figure 3.9. Shows the result of single vein excavated under various GSI values: (a) Comparison on mining sequence and factor of safety and (b) Comparison on the mine distance of cross-section along with A-A´ and horizontal displacement.

To verify the results of the failure state, factor of safety was computed to understand the instability condition. Figure 3.8 shows the results of factor of safety for a single vein excavated under various GSI values. It can be observed that in the open stoping method the factor of safety is low all around the excavated area, sill pillar and the crown pillar while in cut and fill method the factor of safety is low mainly in the crown pillar and the sill pillar. This demonstrates that the open stoping method has high instability risk than the cut and fill method at all GSI values.

In terms of mine sequencing, as shown in Figure 3.9 (a), at initial stage of excavation, the safety factor is higher than at any of the subsequent excavation stages but it can be noted that at stage 8 the stability is slightly high than at stage 7 and 9 because of the sill pillar that was left when moving from the first excavation sequence to the next excavation sequence. In general, it is recommended that as excavation progresses, careful assessment of the stability conditions is important in order to judge if artificial support is required or not in relation to the rock mass condition. From the case of Hermyingyi Mine, the geological conditions characterized by GSI values of 46, 50, and 55 are reliably stable and artificial support would not be required because the rock mass is hard enough to self-support but rock masses characterized by GSI values of ≤ 30, artificial support technologies would definitely be required.

The stability conditions were also evaluated using the displacement pattern. Figure 3.9 (b) shows the displacement extents in the two geological domains namely the footwall and the hanging wall. It is interesting to note that despite the weak rock mass condition of the hanging wall the displacement is high in the foot wall. This can be attributed to the vein orientation which is inclined and also the gravitation weight since the foot wall sustains the mine base. On another note, as anticipated, the displacement is high in weak rock mass conditions especially at GSI value of 20 and low in hard rock mass condition particularly at GSI value of 55.

3.3.3 Various inclined vein angles

After the analysis of the geological strength index (GSI), a parametric study of the orientation of the inclined vein was carried out. The actual inclination of the veins at Hermyingyi Mine is near vertical ranging from 70° to 85° but the stability analysis evaluated the stability under various vein angles even less and more than the range aforementioned to understand its influence. Finite difference models of the stopes and sill pillars were developed below 120 m from the surface at various dipping angles of 50°, 60°, 70°, 80°, and 90°. Comparison of the open stoping method and cut and fill method was also done. Figure 3.10 presents the simulation results of the stope stability under various vein dipping angles. According to the results, high potential failure state is observed at 50° inclined vein than at all other inclination of the vein in both the open stoping and cut and fill methods. In the open stoping method, the potential failure zone is approximately 10 m in the upper excavation section and 15 m in the lower excavation section at 50° with the hanging wall highly affected, at 70° the potential failure zone is approximately 7 m in the upper excavation section and 10 m in the lower excavation section with the hanging wall slightly more affected, and at 90° the potential failure zone is symmetric around the excavated stopes at just about 7 m both in the upper and lower excavation sections as shown in Figure 3.10 (a).

In the cut and fill method, the potential failure zone is approximately 7 m in the upper excavation section and 11 m in the lower excavation section at 50° with the hanging wall highly affected, at 70° the potential failure zone is approximately 5 m in the upper excavation section and 6 m in the lower excavation section with the hanging wall slightly more affected and at 90° the potential failure zone is symmetric around the excavated stopes at just about 5 m both in the upper and lower excavation section as shown in Figure 3.10 (b). Thus, in terms of the mining method, the open stoping method has high instability potential than the cut and fill method but in both cases the lower excavation section is more dangerous than the upper section because of the large extents of shear-n and tension-p, and shear-p recorded.

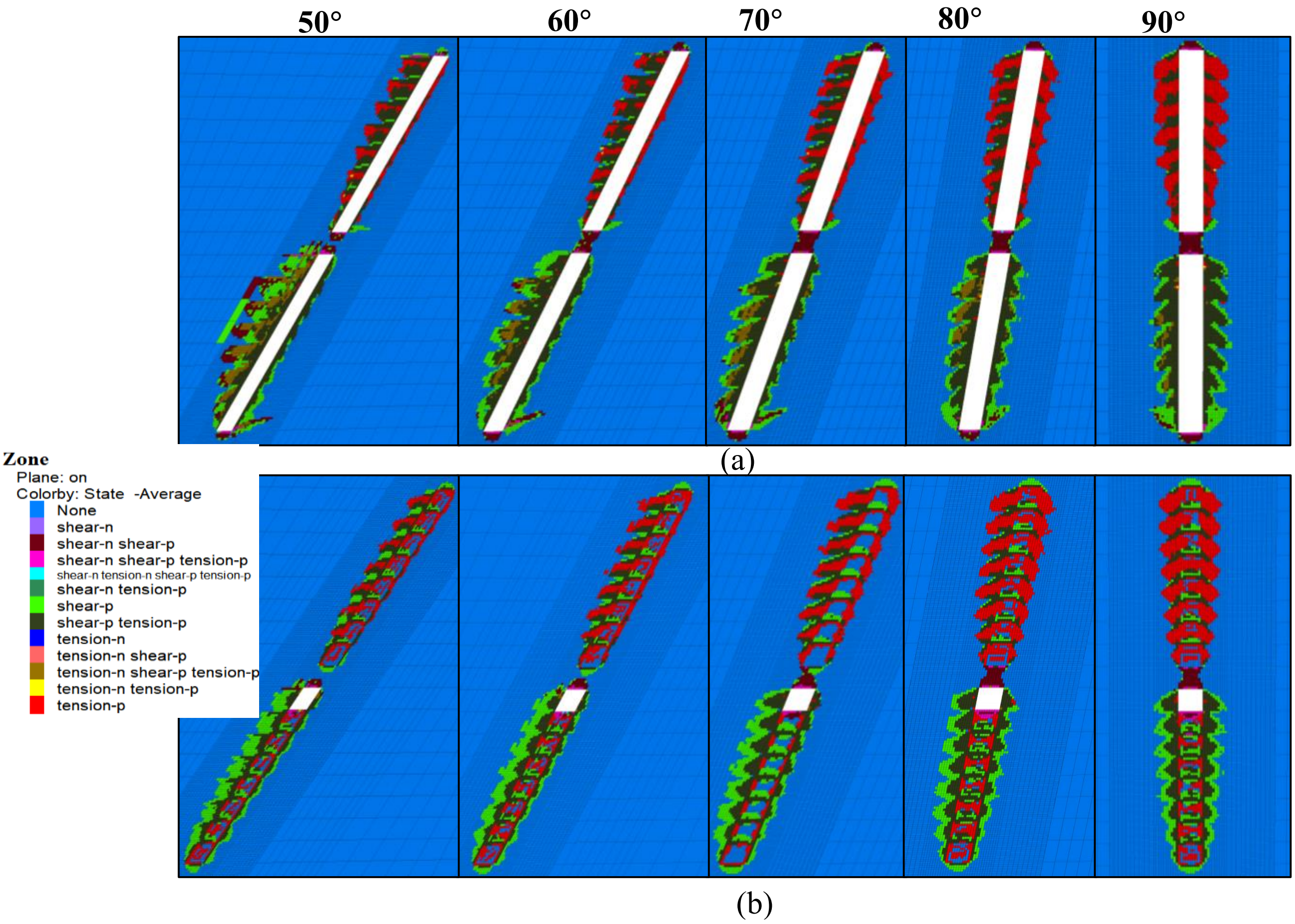
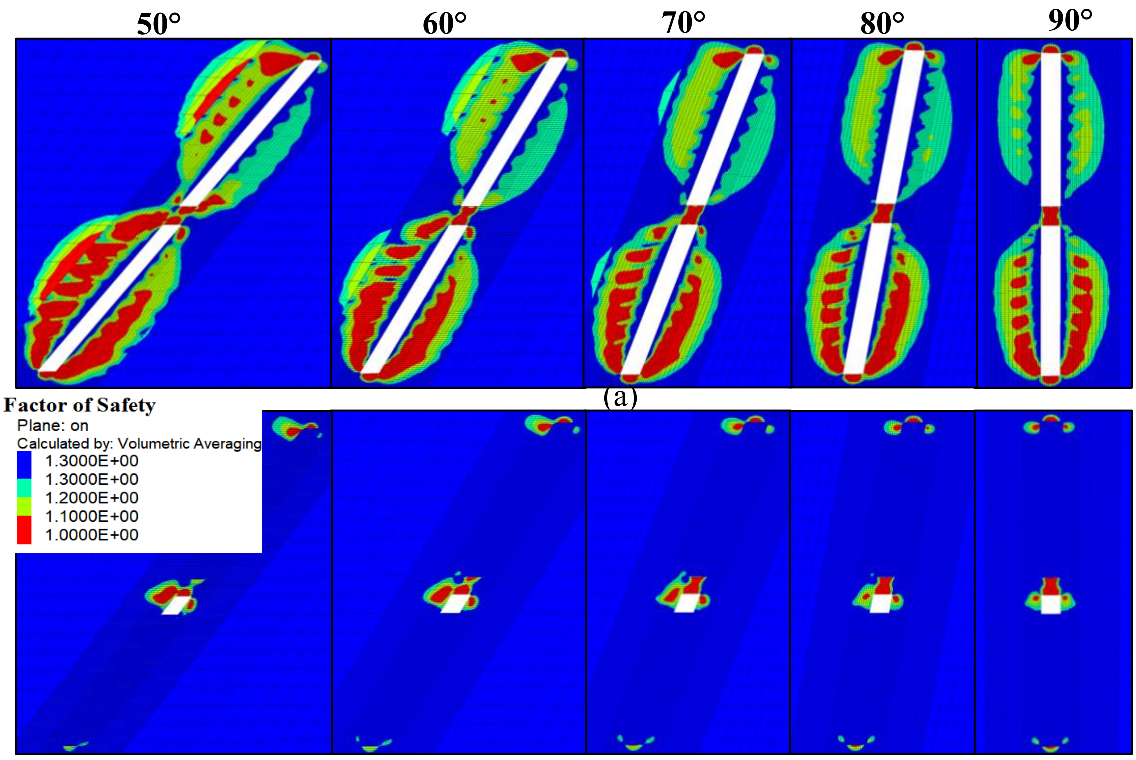


Figure 3.10. Simulation result of the failure of stope stability under various vein angles: (a) The open stoping method and (b) The cut and fill method.

After interpretation of the potential failure zones, the FoS was also calculated under different vein dipping angles in the similar fashion. Figure 3.11 gives the result of FoS of both methods at different inclined vein dipping angles.



(b)

Figure 3.11. Simulation result of FoS of stope stability under various vein angles: (a) The open stoping method and (b) The cut and fill method.

The outcome of the analysis indicates that in the open stoping method the factor of safety is low all around the excavated area, the sill pillar and the crown pillar (see Figure 3.11 (a)) while in cut and fill method the factor of safety is low mainly in the crown pillar and the sill pillar (see Figure 3.11 (b)) and the open stoping method has high instability risk than the cut and fill method at all dipping angles. Regarding comparison of the vein inclination, the gentle inclined vein at 50° has high instability risk than steep dipping angles of the vein which is in concordance with the failure state results.

3.3.4 Different stope width

Considering the fact of safety in mining operations, the study evaluated the stability conditions of the excavated stope dimensions. It is critical to carefully consider the width of stopes because the stress distribution has an influence in excavated stope dimensions. Currently, at Hermyingyi Mine, the Sn-W deposit is being extracted at ≥ 2 m stope width. In this study, the stope widths in the vein were modeled and evaluated at 2 m, 3m, 5 m, and 10m and the sill pillar was kept at 5m thickness between the two excavation sequences. Figure 3.12 shows the results of the instability of stope under various vein widths of 2 m, 3m, 5 m, and 10 m. From Figure 3.12, it is observed that larger excavated stope dimensions expose the mine to a higher risk of failure than smaller excavated stope dimensions. At 2 m and 3 m widths, the potential failure zone is at acceptable levels and the sill pillar is also stable but at the width of ≥ 5 m the potential failure zone is quite unacceptably large for safe mining operations and the sill pillar becomes unstable. Hence, the ore recovery plan from the deposit has to cautiously consider the dimensions of the stope by keeping them as narrow as possible to ensure high geotechnical safety of the mine and evade unnecessary mine failure or collapse.

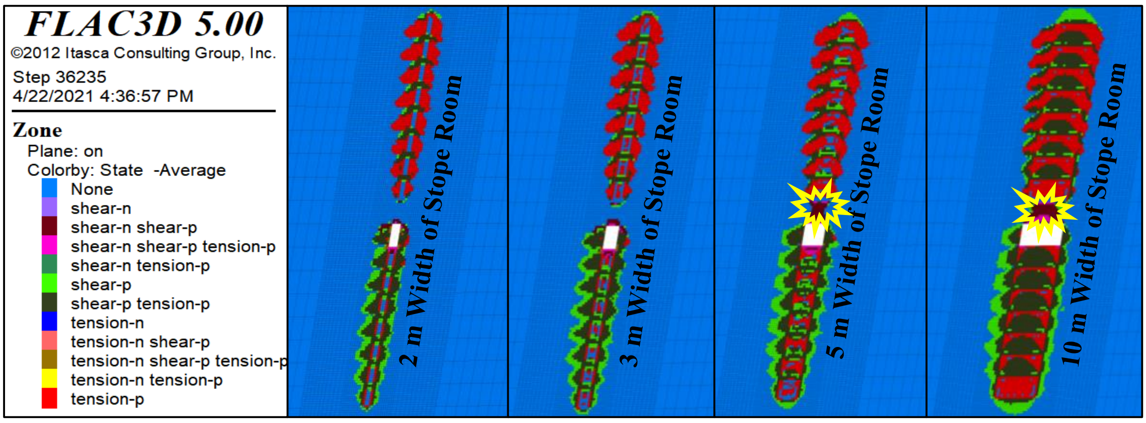


Figure 3.12. Simulation results in the instability of stope under different widths of stope.

3.4 Assessment of the Stability of Sill Pillar

The sill pillar is one of effective features to stabilize the underground mine with crown pillar and no-crown pillar. Sill pillar is defined as a portion of deposit underlying an excavation and left in place as a pillar (Tri & et al., 2016). In the underground mine excavation, the sill pillar should be kept for the safety of the mine workers and transportation of ore metal from the mined-out area as shown in Figure 3.13. In the previous simulation results, the still pillar was kept at 5 m thickness for the excavations with different stope dimensions. The results showed the sill pillar would be at risk of collapse if the stope width is ≥ 5 m. Therefore, in this section various thickness of the sill pillar were evaluated at the 5 m width of the stope to propose the optimal thickness that can be left if the recovery plan is adopted for large dimensions of stope. To achieve the objective, simulation model was created by comparing the sill pillar thickness of 3 m, 5 m, and 10 m. The results of the analysis are presented in Figure 3.14.

Figure 3.13. Systematic diagram of an underground mine with the sill pillar.

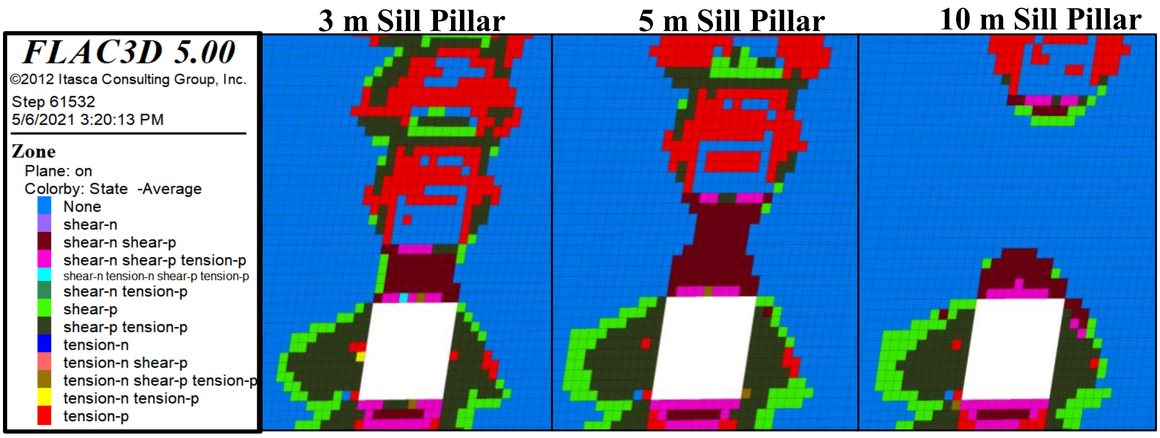


Figure 3.14. Simulation result of the instability of the sill pillar under different lengths of the sill pillar.

The results show that the size of sill pillar thickness is critical to the stability and safety of the underground mines that intend to adopt large stope dimensions. As shown in Figure 3.14 the failure state of the sill pillar is high linking the upper and lower excavation sections with active shearing characterized by shear-n and shear-p at 3 m and 5 m while at 10 m sill pillar thickness the stability of the pillar is restored with minimal active shearing and the linkage of the failure state between the two excavation sections is disjointed. Furthermore, the stress distribution as shown in Figure 3.15 indicate that the principal stresses are severer surrounding the sill pillar region at 3 m and 5 m pillar thickness than at 10 m thickness because of the induced stress affecting the sill pillar. This shows a good stability condition for the underground mine adopting large stope dimensions with ≥ 10 m sill pillar thickness. The minimal active shearing observed a 10 m thickness of the sill pillar can be fully eliminated with artificial support if required. However, a large thickness of the sill pillar has to be carefully considered in economic terms since a lot of ore would be left unrecovered in an open stoping mining method.

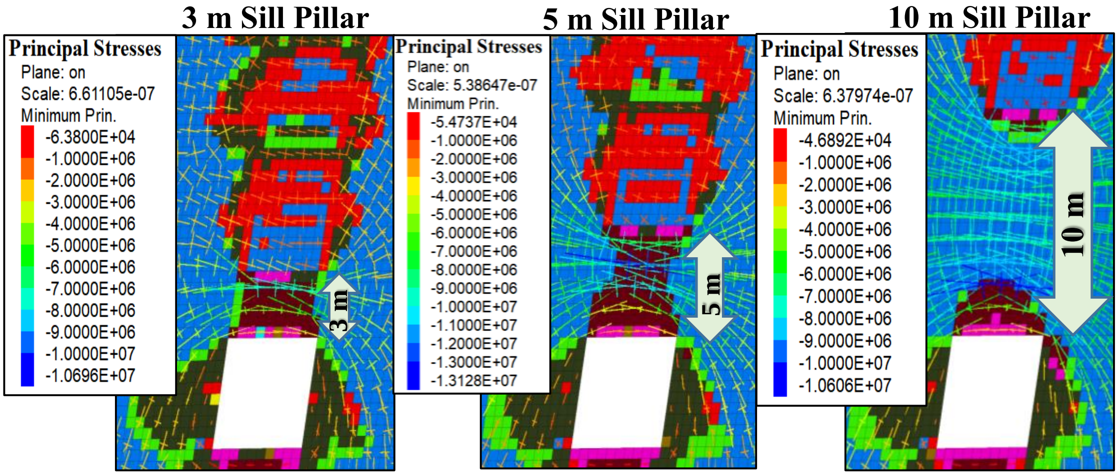


Figure 3.15. Simulation result of the stress distribution of sill pillar under different lengths of sill pillar.

3.5 Evaluation of Stope Stability with Various Excavated Sequences

According to the previous results, the sequences of excavation of mining stopes appeared to control the stability of stope. Figure 3.16 shows the systematic diagram of stoping sequences represented by a red line colors in the block model. Excavation sequencing is one of the important aspect for underground mines. Hence, three kinds of mining sequences were evaluated in the simulation by FLAC^{3D}.

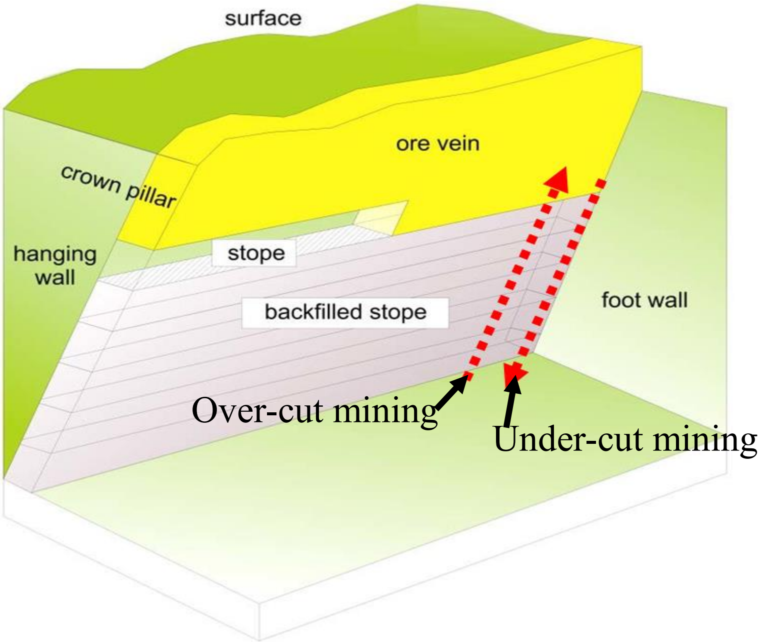


Figure 3.16. Systematic diagram of stoping sequences line by red color.

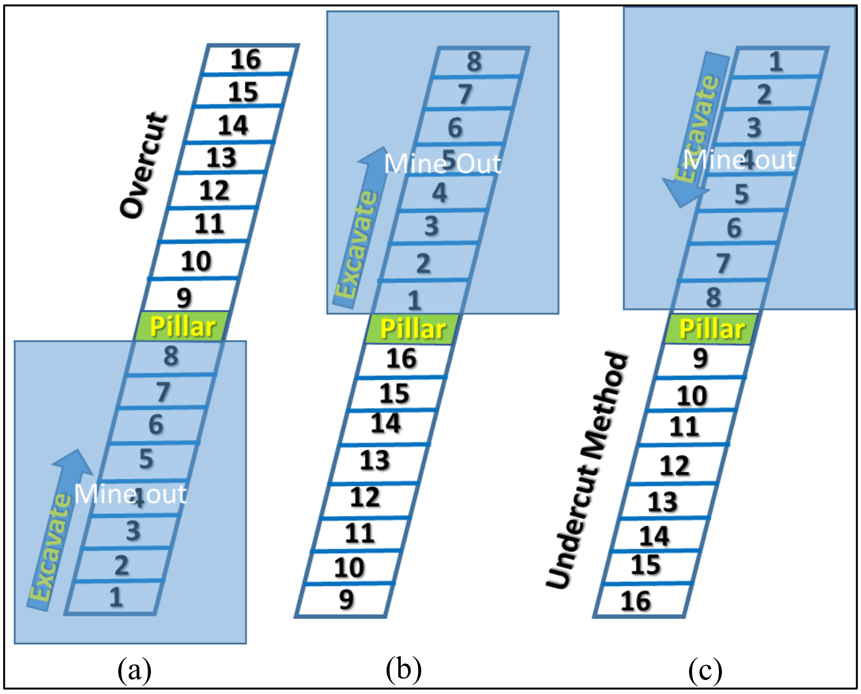
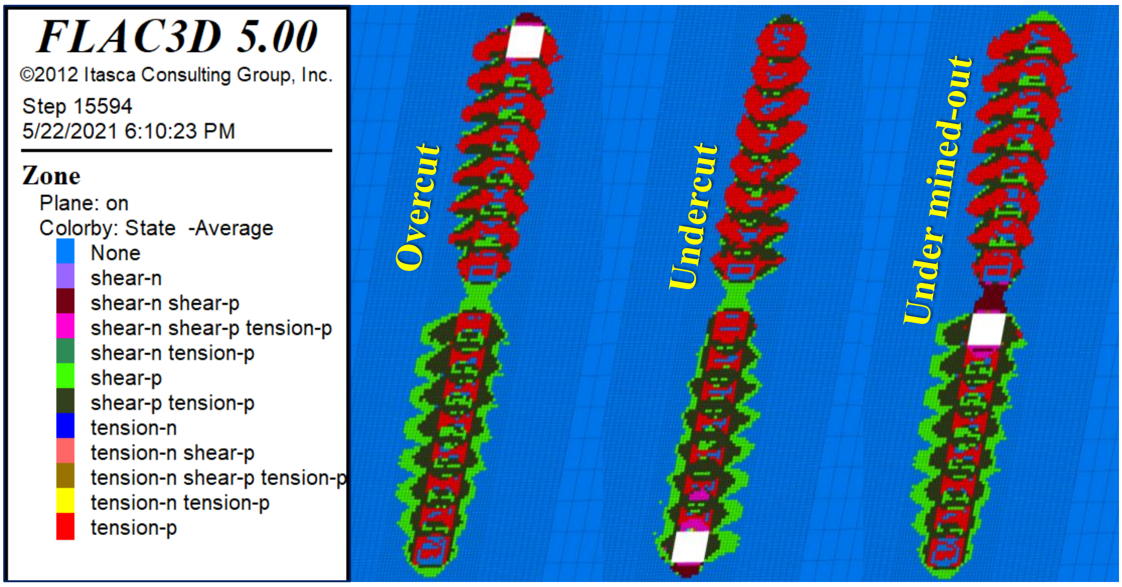


Figure 3.17. Systematic diagram of the sequence of the mine steps: (a) Over-cut sequences stope, (b) Under mined-out area excavation sequences stope, and (c) Under-cut sequences stope.

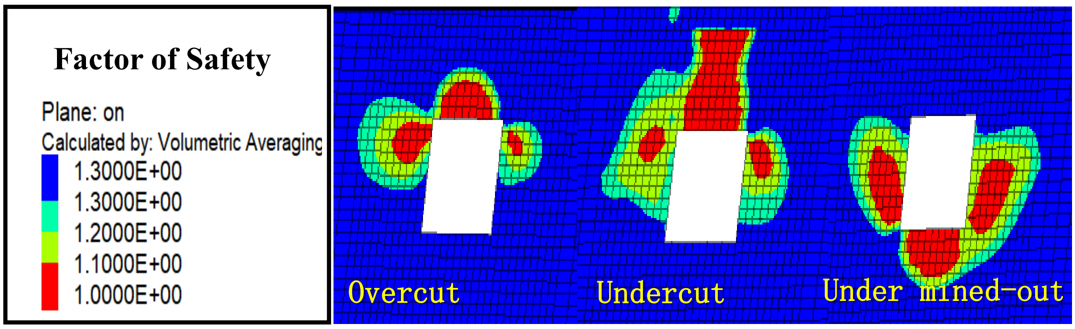
The sequences of mine excavation are as follows; over-cut mining sequencing, under mined-out area excavation sequencing, and under-cut mining sequencing as shown in Figure 3.17. The detailed explanation of mining sequences is as follows (i) over-cut mining: the first stope excavation is started from the bottom going up top, (ii) under mined-out excavation: the first excavation stope is started above the sill pillar at the top of mine stopes and then excavation returns back to the bottom of the lower mine stopes below the mined-out area, (iii) under-cut mining: the first excavation of stope is started from top to bottom. A series of simulations for the model with GSI value of 30 and 80° inclined vein were carried out. The mining excavation was conducted in 16 stopes splited by a sill pillar in equal 8 stopes in the two sections. The sill pillar is important for controlling the instability of the underground mine as previously discussed.

Figure 3.18 and Figures 3.19 show the stability of stopes under different mining sequences. The results of rock failure evaluated in over-cut mining, under the mined-out mining, and the under-cut mining sequences show that the over-cut mining method offers relatively high stability for underground mining than the under-cut mining and the under the mined-out mining methods. As shown in Figure 3.18 the potential failure zone is slightly small than the other two methods. Comparing the under-cut mining and the under the mined-out mining sequencing the latter is more dangerous than the former because the sill pillar becomes unstable with active shearing characterized by shear-n and shear-p. In the over-cut and under-cut mining sequencing, the sill pillar is stable with shear-p showing restoration of stability. Regarding factor of safety, the over-cut mining sequencing shows high stability with small low instability section around the stope than in the under-cut mining and the under the mined-out mining sequencing as shown in Figures 3.19. This can be explained in terms of the redistribution of stress during the stoping process. Generally, the initial cut of the stope experiences high stress induced after the cut, as mining progresses the induced stress gets redistributed. In the over-cut mining sequencing the loading effect is minimized as the excavation progresses. On the other hand, in the under-cut and under the mined-out mining sequencing the loading effect on the cut stope increases.



(a) (b) (c)

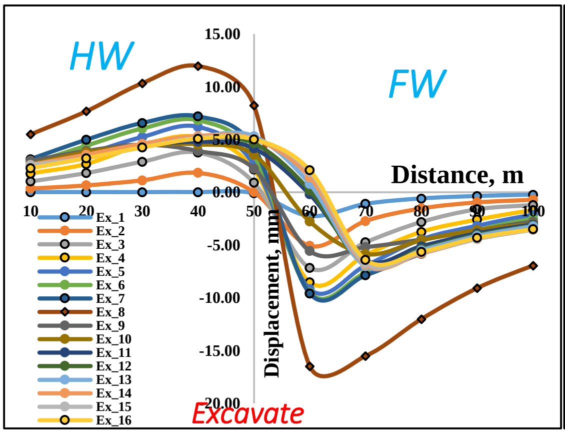
Figure 3.18. Simulation results of failure of stope under different mining sequences in underground mine (Sn-W) Deposit Mine: (a) Over-cut sequences stope, (b) Under-cut sequences stope, and (c) Under mined-out sequences stope.



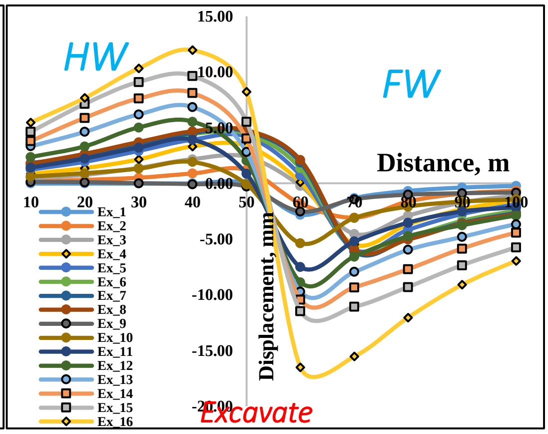
(a) (b) (c)

Figure 3.19. Simulation results of safety factor of stope under different mining sequences in underground in (Sn-W) Deposit Mine: (a) Over-cut sequences stope, (b) Under-cut sequences stope, and (c) Under mined-out sequences stope.

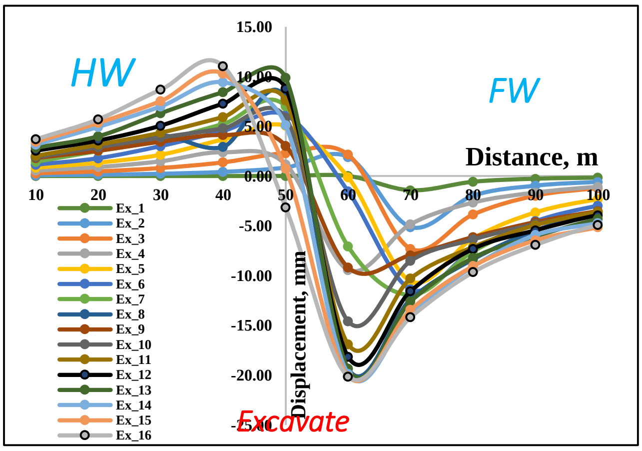
After evaluating the failure state and safety factor, the displacement was also evaluated to understand the extent of deformation in the underground excavation under various mining sequences in (Sn-W) Deposit Mine. The results of the displacement are shown in Figure 3.20. It can be observed that high displacement occurred at the eighth step of the excavated stope (Ex-8) for over-cut mining sequence which is the final excavation step in the lower excavation section (see Figure 3.20 (a)). In the case of under mined-out mining sequence, high displacement occurred at the sixteen steps of excavated stope (Ex-16), which is as well the final excavation step for stoping sequences Figure 3.20 (b). Finally, in the under-cut mining sequence, the high displacement occurred at the first step of excavated stope (Ex-1) which also happens to be the last excavated stope Figure 3.20 (c). Comparing the results of excavation in the last stoping, the under-cut mining sequence recorded high displacement of 20 mm than the under mined-out mining and the over-cut mining sequences as shown in Figure 3.21.



(a)



(b)



(c)

Figure 3.20. Simulation results of displacement of stope under different mine sequences of the stope: (a) The displacement results at each stope under-cut mining method (b) The displacement results under mined-out excavated mining method, and (c) The displacement in under-cut mining method.

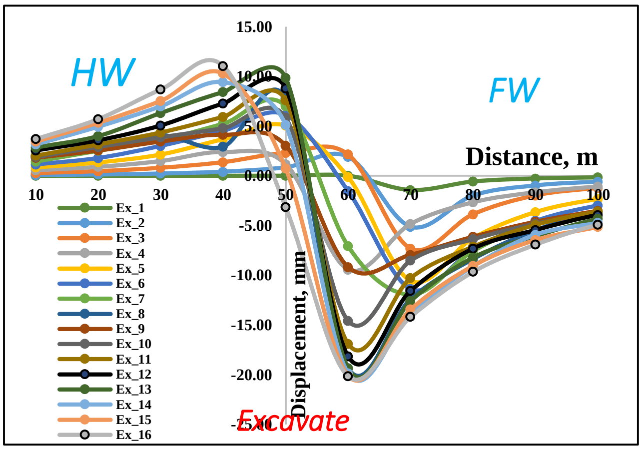
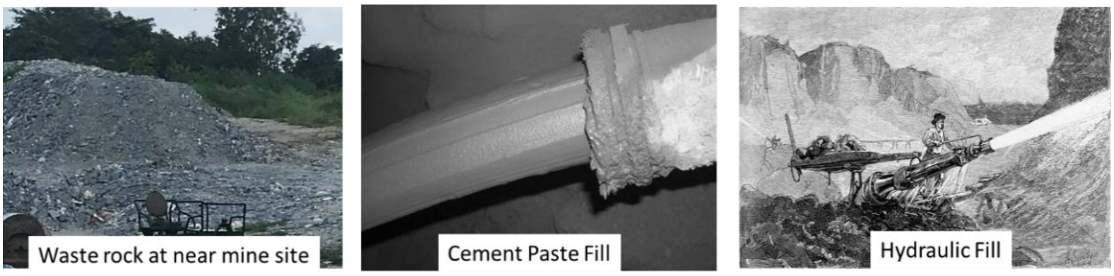


Figure 3.21. The instability of stope is compared due to the various mining sequences.

From the same Figure 3.21, as observed previously, the footwall recorded high displacements than the hanging wall. Thus, from the results, we can conclude that the over-cut excavation sequencing is good to control the instability of stopes for the mine safety in underground mine.

3.6 Instability of Optimized Stope with Various Types of Filling Materials

After analyzing the instability of mine under different conditions in the underground mine for Sn-W Deposits, the results showed that higher failures are observed in the open stoping method because the induced stresses from surrounding excavated areas affect the open excavated section. To minimize the risk of instability and collapse economical supporting solutions are evaluated for recommendation. In this section, the supporting solutions using filling materials are assessed for stope instability optimization. Figure 3.22 shows different types of filling materials applicable for stope instability optimization. In this study, the numerical models for simulation were constructed with the single vein stope and four different types of backfilling materials were introduced which include; waste rock fill (WRF), hydraulic fill (HF) (mixture of 60% - 80% solid in water), cement paste fill (CPF) (cement-tailing ratio was 1:8) and cemented rock fill (CRF) (mixture with 5% cement with waste rock) as shown in Table 3.2.



(a) (b) (c)

Figure 3.22. Different types of filling material applicable for stope instability optimization.

Table 3.2. The properties of the various types of the filling material.

Parameter	Cemented Paste Fill, CPF	Cemented Rock Fill, CRF	Hydraulic Fill	Waste Rock Filling
Uniaxial compressive stress (MPa)	1.01	0.7	-	-
Young's modulus (MPa)	1130	2850	181	153.1
Poisson's ratio	0.16	0.34	0.2	0.321
Friction angle (°)	48	25.4	6.9	20.5
Cohesion (MPa)	1.16	1.4	0.07	0.2

The simulation models with GSI value of 30 and 80° inclined vein were created and each of the filling material was analyzed to compare the efficacy of the various types of filling materials for the Hermyingyi (Sn-W) Deposits Mine. Figure 3.23 shows the results of instability of stope for various filing types. According to the results, the instability condition is critical prior to installation of support system. The failure region is extremely large and the factor of safety, which has also been quantitatively presented in Figure 3.24 (b), is low around the wall of the excavated stope indicating high instability around the mine excavated area especially in the hanging wall side. This implies a potential high risk of failure for the excavated stope area. After the introduction of different filling materials, the failure region is extremely reduced and the factor of safety increases around the mine excavated area indicating very improved stability.

In terms of displacement, the results are in harmony with the results of failure state and factor of safety. The CRF is clearly recognized to have the lowest displacement followed by the CPF and HF recorded highest displacements around the excavated and filled stopes as shown in Figure 3.24 (a). Hence, in terms of efficacy, the CPF, CRF and WRF types of filling material are more effective in reducing instability than the HF filling material. Comparing CPF, CRF and WRF, it can be observed that CPF and CRF have excellent stabilizing performance than the WRF with CRF being more excellent. However, the WRF is a better stabilizing option in the economic point of view because it is cheaper than the CPF and the CRF since the materials can be obtained from the dumping site near the mined-out area.

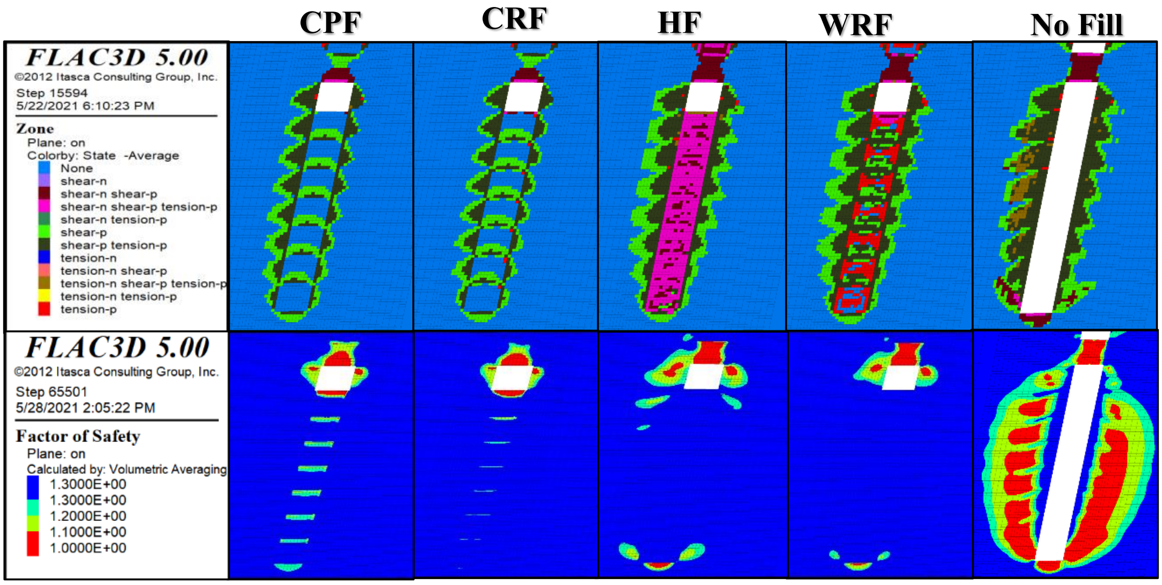
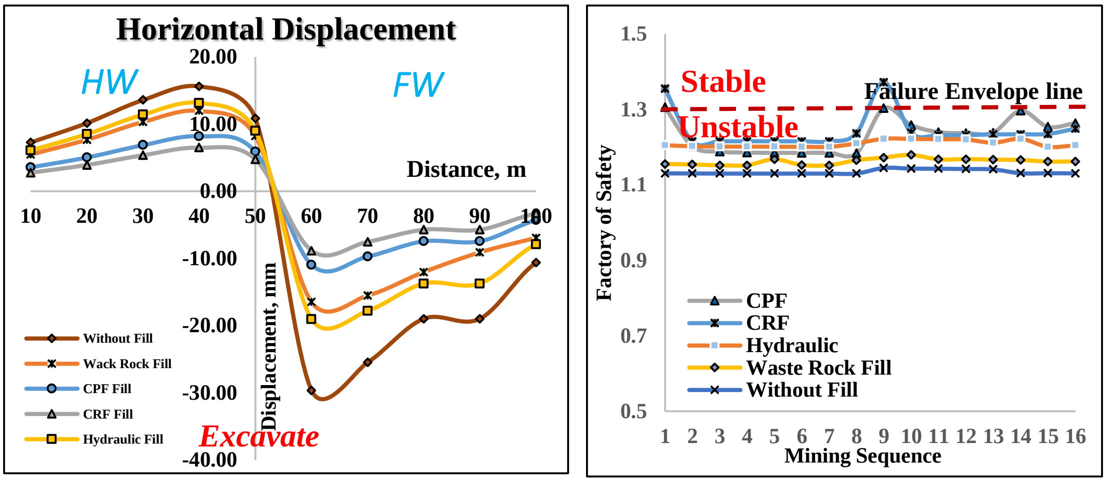


Figure 3.23. Simulation results for instability of stope with different filling material.



(a) (b)

Figure 3.24. Simulation results stope instability with different filling material.

3.7 Conclusion

In conclusion this chapter has evaluated the instability of stopes at a single excavated inclined vein under several conditions of the Hermyingyi (Sn-W) Deposits Mine. Currently, the mining method being used for the metal extracted from the ground at Herymingyi Mine is the open stoping method. After the investigation of the instability of stope in several conditions of the mine at the single vein a number of aspects have been made clear which could be applicable in different mining regions. Firstly, the geological condition of the rock mass has been confirmed to be an influential factor in mine stope stability. Generally, weak rock masses with GSI value of ≤ 38 are susceptible to instability and require artificial supporting mechanisms. Secondly, it has been revealed that high vertical stress in shallow underground mines like the Hermyingyi Mine greatly affect the stability of excavated stopes, however at a depth of ≥ 150 m the impactful stress is the horizontal stress. The other aspect is that the inclination of the veins influence greatly the stability of the excavated stopes especially in the hanging wall section. It has been demonstrated that inclined veins dipping at ≤ 70° in weak rock mass condition expose the excavated stopes to higher risk of instability and/or collapse particularly in the hanging wall section. The fourth aspect is that the sill pillar thickness has to be carefully designed for the stability of underground mines. The underground mines intending to develop large excavated stopes of ≥ 5 m width should keep a sill pillar thickness of ≥ 10 m but in excavated stopes of ≤ 3 m, a sill pillar thickness of 5 m is sufficient. Another interesting finding is that the mining sequencing determines the stability of the underground mine and that over-cut mining sequence approach is by far the best mining approach for underground mining. Finally, in the case of higher instability in the mine, cheap supporting mechanism for optimizing the instability are encouraged. The study, has shown that among the economical supporting solutions, the CPF and the CRF types of filling material have excellent stabilizing performance in excavated stopes with CRF being the most excellent one. However, the WRF, which has equally an excellent stabilizing performance close to the CPF and CRF, is a better stabilizing option in the economic perspective because the materials can be obtained easily from the dumping site near the mined-out area.

3.8 References

Hoek, E., & Brown, E. T. (1978). Trends in Relationship Between In Situ Rock Stresses and Depth. *Int. J. Rock Mech. Sci & Geomech*, 211-215.

Hoek, E., & Brown, E. T. (1988). The Hoek-Brown Failure Criterion - A 1988 Update. *Proceedings of the 15th Canadian Rock Mechanics Symposium* (pp. 31-38). Toronto: Civil Engineering Department, University of Toronto.

Hoek, E., Kaiser, P. K., & Bawden, W. F. (1995). *(1995) Support of Underground Excavation in Hard Rock. Balkema, Rotterdam.* Rotterdam: Balkema.

Kwon, S., Cho , W. J., & Lee, J. O. (2013). An analysis of the thermal and mechanical behavior of engineering barriers in a high-level radioactive waste repository. *Nuclear Engineering and Technology, 45(1),* 41-52.

Tri , K., & et al. (2016). Countermeasure Method for Stope Instablity in Crown Pillar Area of Cut and Fill Underground Mine. *International Journal of Geosciences, 7*, 1-21.

Yasitli, N. E., & Unver, B. (2005). 3D numerical modeling of longwall mining with top-coal caving. *Journal of Rock Mechanics and Mining Sciences*, 219-235.

CHAPTER FOUR

4 Stability of Crown Pillar in Shallow Underground Tin-Tungsten Deposits Mine

4.1 Introduction

In the previous chapter the instability of the stope of a single vein by comparing the open stoping method and the cut and fill stope method was discussed. According to the results, the open stoping method has an obvious huge impact on the stability of surrounding rock mass, especially not only the crown pillar but also surrounding excavated stope area. Therefore, the safety of the crown pillar in open stoping needs to be evaluated in order to identify the key parameters that would induce failure such as rock strength, the instability due to geological condition, and the ratio of the span of the crown pillar and so on. This is because the tin-tungsten metal extraction is extending to deep levels due to the depletion of resources in the shallow ground as shown in Figure 4.1. Additionally, the condition of the crown pillar in the mine could be affected by the weathering condition such as weak rock, joint spacing, the impact of the environment and orientation of major discontinuities, and other properties of the rock mass surrounding the excavation (Tavakoi, 1994). It is clear that in-situ stresses, structural geology and the geometry of the crown pillar all play significant roles in the stability condition, and all of the parameters can be used to help determine the possible mode of failure. Figure 4.2. shows the example of crown pillar failures which have been derived from some case studies in the research area, in a potential low-stress environment such as crown pillars are expected to be controlled by structure rather than stress. Faults, shear zone, and schistosity may be occurred to affect the stability. Figures 4.2 (a) and (b) show that in altered rock, localized shear failure such as chimneying and crown degradation, where a sound rock environment discontinuities are the most critical parameter, large scale movement is expected. In both cases, it is critical to know the stress redistribution around the excavation and to know if this stress is enough to prevent direct gravity failure or sliding block failure (Betournay, 1989). Most crown pillar failures have occurred as a result of sliding along adversely oriented joints in the rock mass at the hanging wall or footwall contact (Carter, 1989). In shallow underground mines, the high vertical stress may cause collapse of hanging wall and footwall rock, hence designing of the suitable crown pillar is required for determining overall safety of the stope. Furthermore, the need for suitable supports and ground control mechanisms for safe mining necessitates the selection of proper pillar supporting technologies in the stope opening. This analysis is used to understand the behavior of the crown pillar under different thickness, depth, span, and width of the crown. To achieve the purpose of understanding the crown pillar failure at the open stoping method, simulations using Phase^{2}ver. 7.0 software were conducted to identify the yielding element, displacement extents, and safety factor.

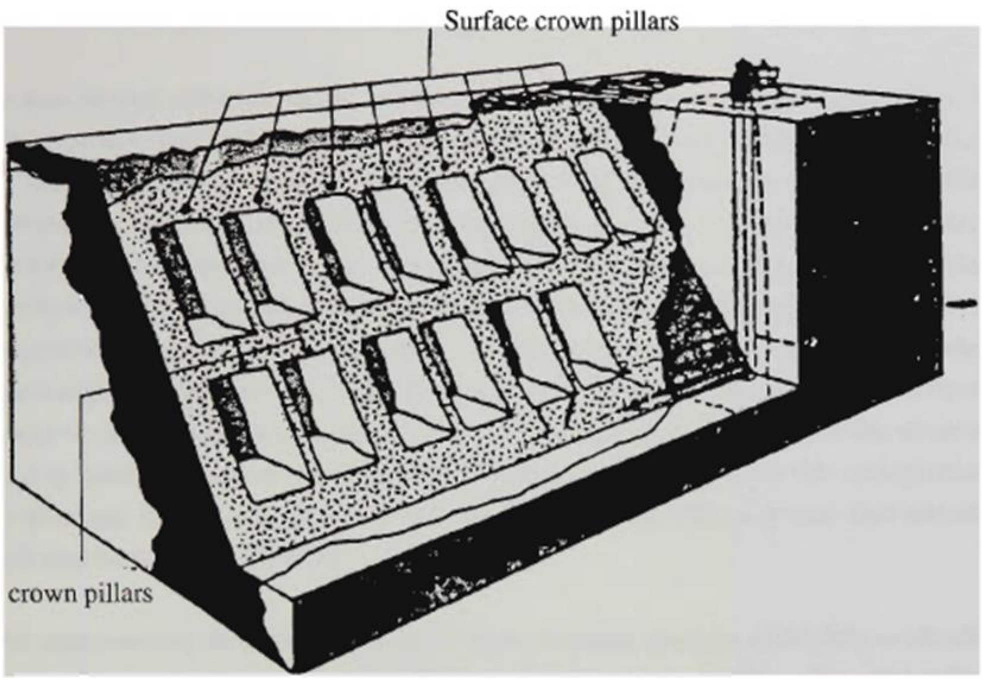


Figure 4.1. Research location of the crown pillar in the underground (Sn-W) Deposit Mine.

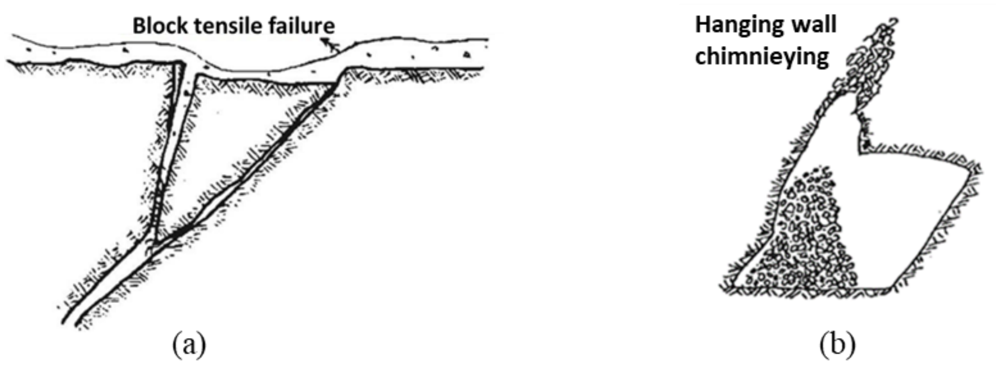


Figure 4.2. Types of failure at the crown pillar in an underground mine.

4.2 Model Simulation

In order to evaluate the failure of the crown pillar, a series of numerical analyses are carried out by two-dimensions finite element of Phase^{2}ver. 7.0 simulation. The model dimensions constructed are 200 x 200 m with 5 m x 5 m stoping, and the excavated area is located at 115 m depth from the surface as shown in Figure 4.3.

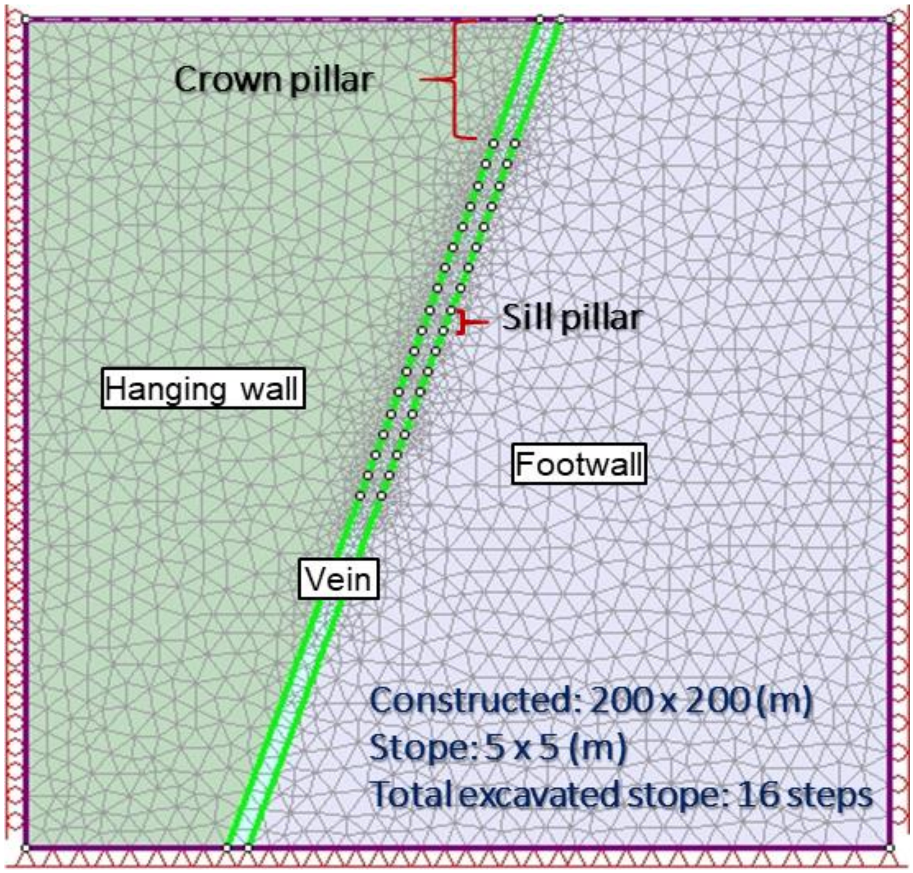


Figure 4.3. Illustration of the constructed model of the crown pillar.

The boundary conditions are fixed in x, y-direction at the bottom, fixed in the x-direction and free at the top. The excavation steps are a total of 16, but after 8 steps of excavation the sill pillar of 5 m is left and the subsequent steps were performed. The stope is based on the actual condition in the mine site of the case study. Because the mining area was developed at shallow underground depth from the where the orientation of ore body could be affected by ground landslides, it is important to be careful design the shape and thickness of the upper part of the mining area when looking at the necessary precautions.

4.3 Stability of Crown Pillar at Shallow Underground Mine

To judge the stable conditions of the crown pillar the FoS of $\geq$ 1.3 is used as a reference after extensive empirical studies using Mohr-Coulomb failure criterion as presented in (Kwon, Cho , & Lee, 2013) and (Hoek & Brown, 1988). Moreover, the yielding elements are calculated for the instability of the crown pillar in which the different parameters of mine conditions like stress ratios, structural and geological conditions are evaluated. To control the instability of the stope, different supporting technologies are installed in order to suggest the effective approach to stabilize the crown pillar in the open stoping area.

4.3.1 Stress ratio

The stress ratio (K) refers to ratio of the horizontal stress by vertical stress (Brown & Hoek, 1978). The parametric study of the stress ratio is carried out to understand the influence of the environmental stress conditions for the instability of the stope area and the crown pillar. The various stress ratios ranging from 0.5 to 2.0 were applied to evaluate the condition of the instability of stope and the crown pillar under high horizontal stress and vertical stress conditions. To achieve the objective of this section, the rock mass properties used were for the GSI range of 30 to 55 with a 70° inclination of the vein.

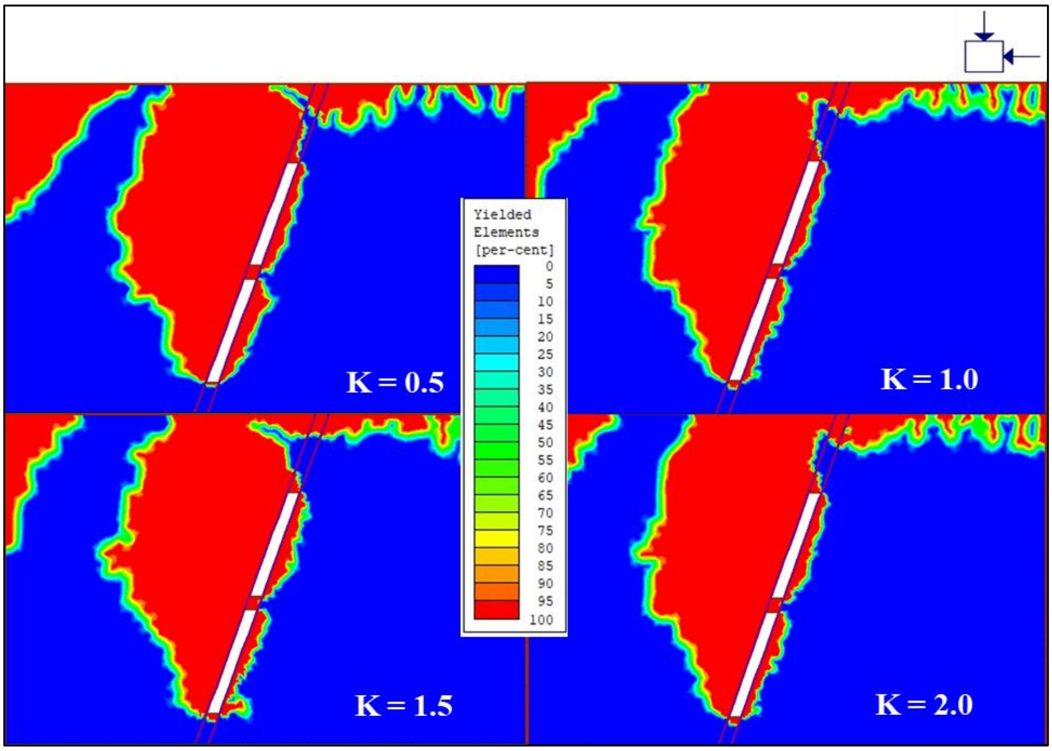


Figure 4.4. Simulation result of yielding elements of the stopes and crown pillar under different K-ratios with GSI 30.

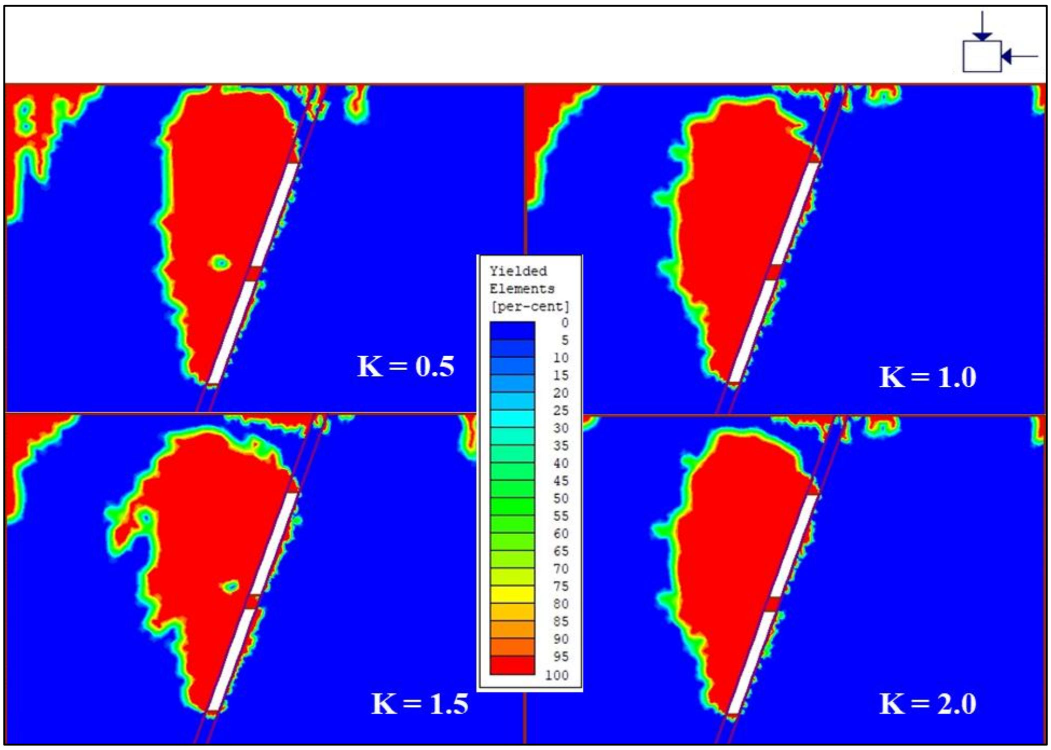


Figure 4.5. Simulation result of yielding elements of the stopes and crown pillar under different K-ratios with GSI 46.

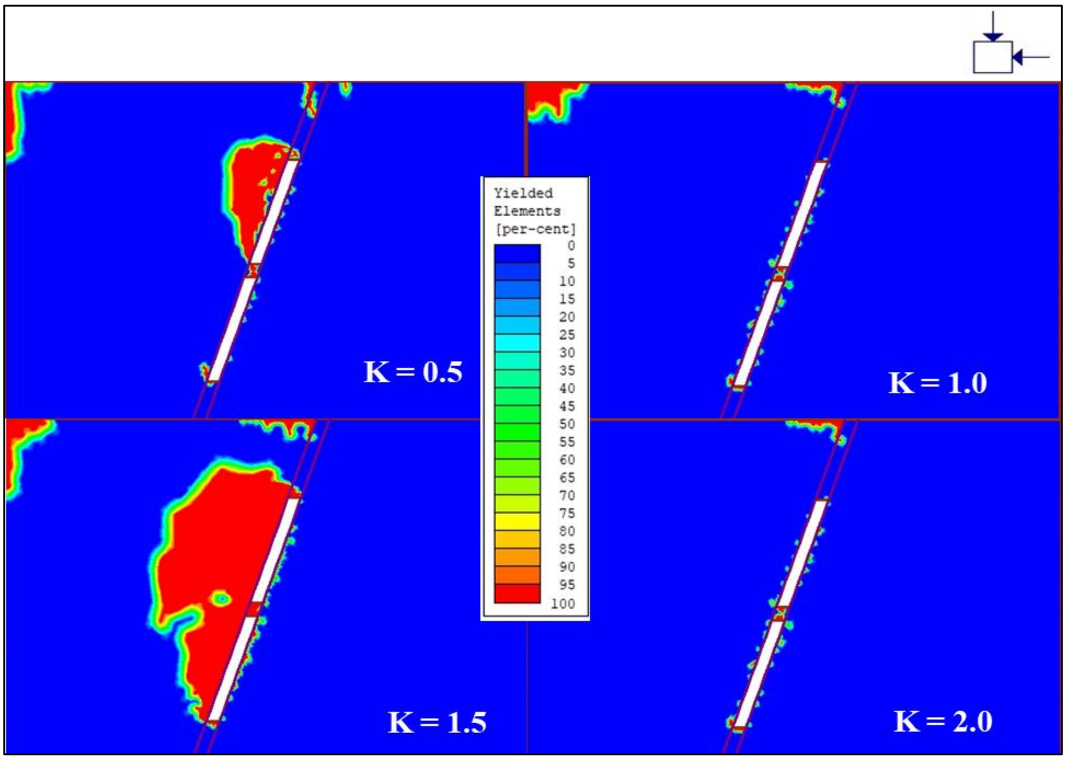


Figure 4.6. Simulation result of yielding elements of the stopes and crown pillar under different K-ratios with GSI 55.

Figure 4.4 shows the simulation results of yielded elements of the stopes and crown pillar under various K ratios for GSI 30. The induced stresses have been observed surrounding the stope and the crown pillar. Based on the results, the yielded element is relatively high at K = 0.5 than at the other K ratios. The representation of this result means vertical stress are more influential especially between the stope and crown pillar regions, where it is located at a shallow area of Sn-W Deposit Mine. Meanwhile, Figure 4.5, which shows the simulation results of yielded elements of the stopes and crown pillar under various K ratios for GSI 46, shows that the induced stress has decreased at stoping area even though the potential failure is still occurring. Therefore, a high value of GSI is evaluated to understand the failure of stope under geology conditions.

Figure 4.6 presents the simulation results of yielded element of the stopes and crown pillar under various K ratios with GSI 55. It can be observed that the induced stress has dramatically reduced irrespective of the K ratio when compared to the GSI of 30 and 46. However, high induced stresses occurred at K = 0.5 and K = 1.5 in the stoping area. This trend at K = 0.5 can be explained with respect to mine depth. Since the mining is still at shallow underground, the vertical stress tends to be high than the horizontal stress which directly affect the stopes of the gentle dipping inclined vein and the crown pillar due to gravitational loading. For K = 1.5, redistribution of the induced stress due to the differential stress condition could account for the development of the large yielding elements especially at the lower initial sequence of excavation.

4.3.2 Geological strength index

After studying the stress condition of the Sn-W Deposits Mine, similarly, the rock mass condition was evaluated to understand the behavior of the crown pillar in relation geological conditions using geological strength index properties. Different GSI values were simulated ranging from 25 representing disintegrated rock mass with poor joint surface quality, to 55 representing blocky rock mass with good joint surface quality. The geology is an essential aspect for mine safety not only stoping area but also crown pillar due to the controlling of ground condition and structural effect of the environment. Therefore, the stability of the crown pillar has been evaluated with respect to geological strength index.

Figure 4.7 shows the simulation results of stope and the crown pillar under various GSI values. Based on the results, a large yielded zone is observed at the open stope area and the crown pillar under GSI values 25, comparing the other GSI values. Thus, open stoping method is not suitable in weak rock mass. However, in good geological conditions like of GSI 55, the open stoping method could be applicable minimal support requirement around the crown pillar.

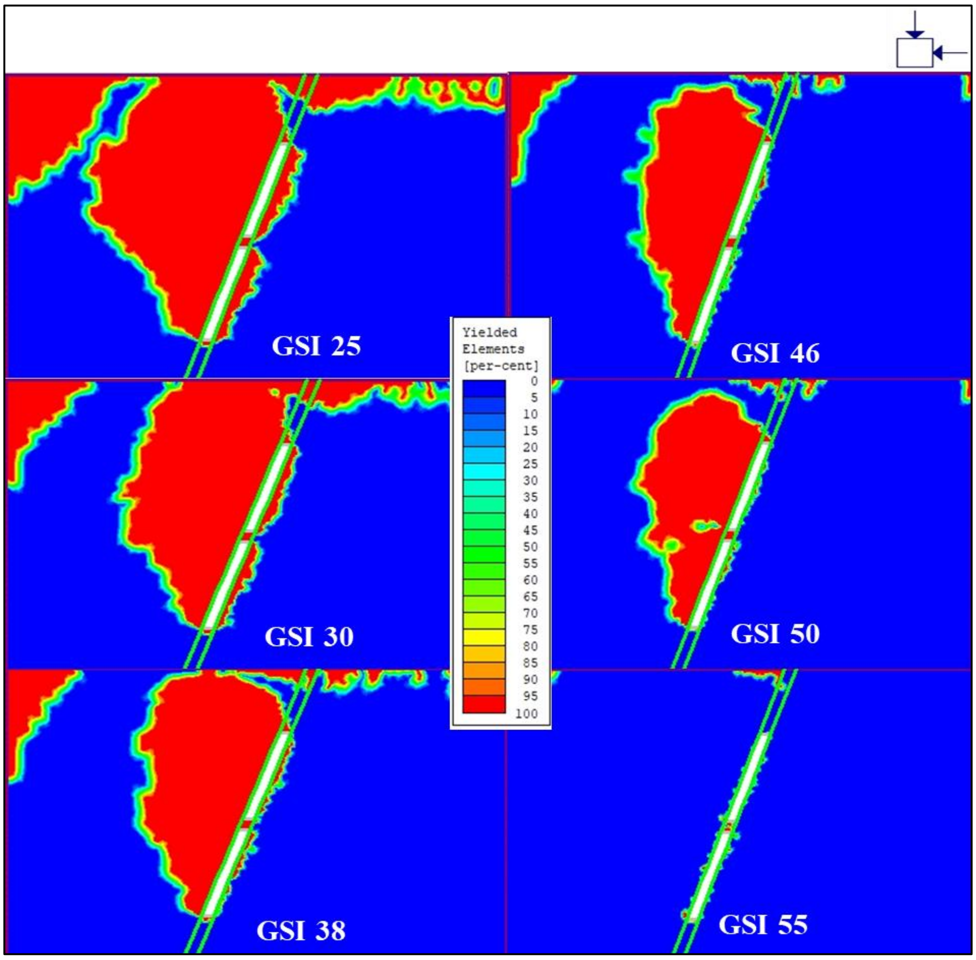


Figure 4.7. Simulation result of yielding elements of the stopes and crown pillar under different GSI values.

4.3.3 Various ore dipping

To understand the stability of stoping area and crown pillar due to various inclined veins, simulations were undertaken with different vein angles ranging from 50° to 90° inclined veins with 30, 46, and 55 GSI. As shown in Figure 4.8, as the vein angles become more gentle, the induced stress becomes higher and the yielding zone is expansive especially at 50° inclined vein because the rock strength in the hanging is weaker than the vein and footwall rock strength. On the other hand, the overburden load of mine and environmental stresses are applied to the excavated area in the mined-out area. But the lower part of the footwall of stope stability is reduced than the upper side of the hanging wall. At the 90 ° inclined vein a systemic failure around the stope area is observed but it is extremely reduced compared to other angles of vein emplacement.

From the results, the development of mine in gentle inclined veins would require comprehensive supporting systems to control the instability around the crown pillar.

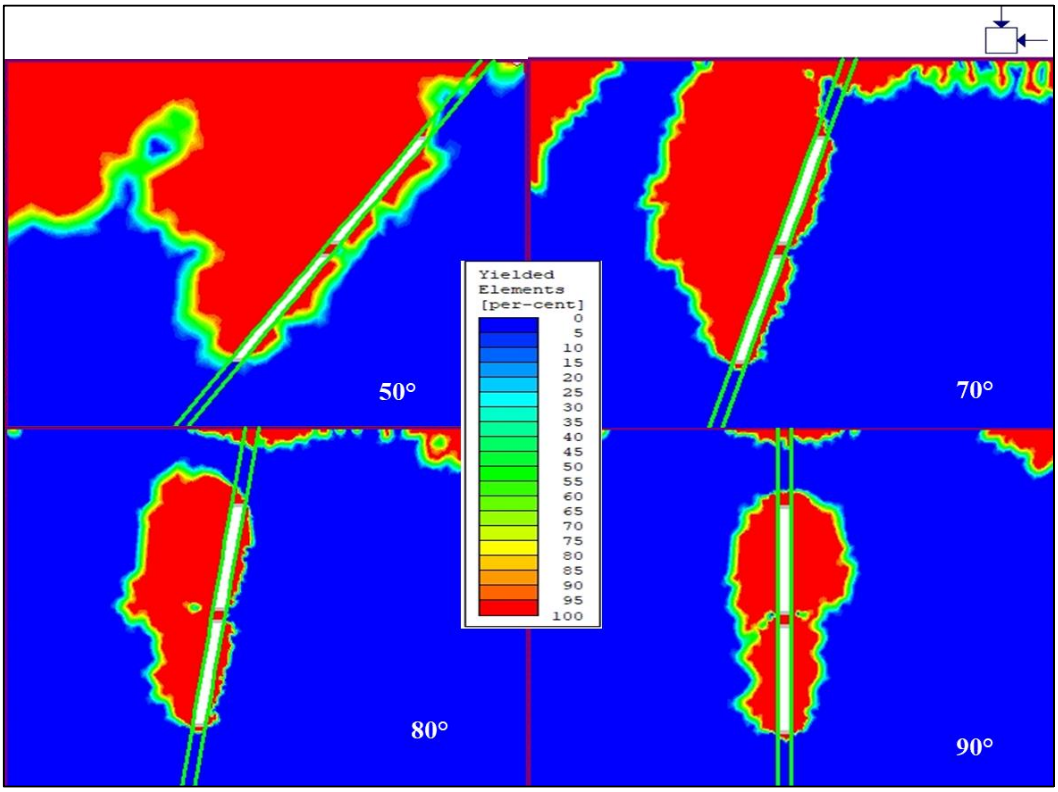


Figure 4.8. Simulation result of yielding elements at the crown pillar under various inclined veins with GSI 30.

Figure 4.9 shows the simulation results of yielding elements at crown pillar under different dip of the vein with GSI 46 and Figure 4.10 shows the simulation result of yielding elements at crown pillar under different dip of the vein with GSI 55. In general, the increase in the rock strength reduces the effect of the induced stress and the yielding zone becomes small. In the case of GSI 46, the yielding zone is relatively small compared to GSI 30 and relatively high compared to GSI 55. In Figure 4.10, only the 50° dipping vein has a large yielded zone around the crown pillar. In the other cases of steep dipping inclined vein the yielding is systemic around the stopes with minimal effect around the crown pillar.

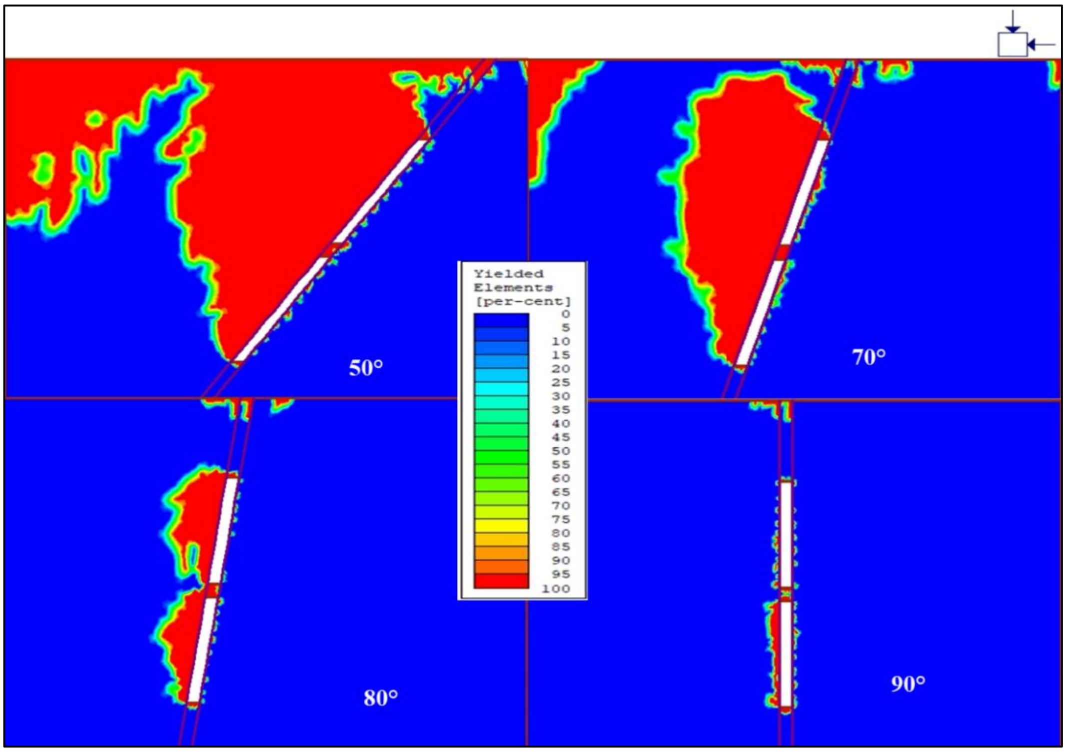


Figure 4.9. Simulation result of yielding elements at the crown pillar under various inclined veins with GSI 46.

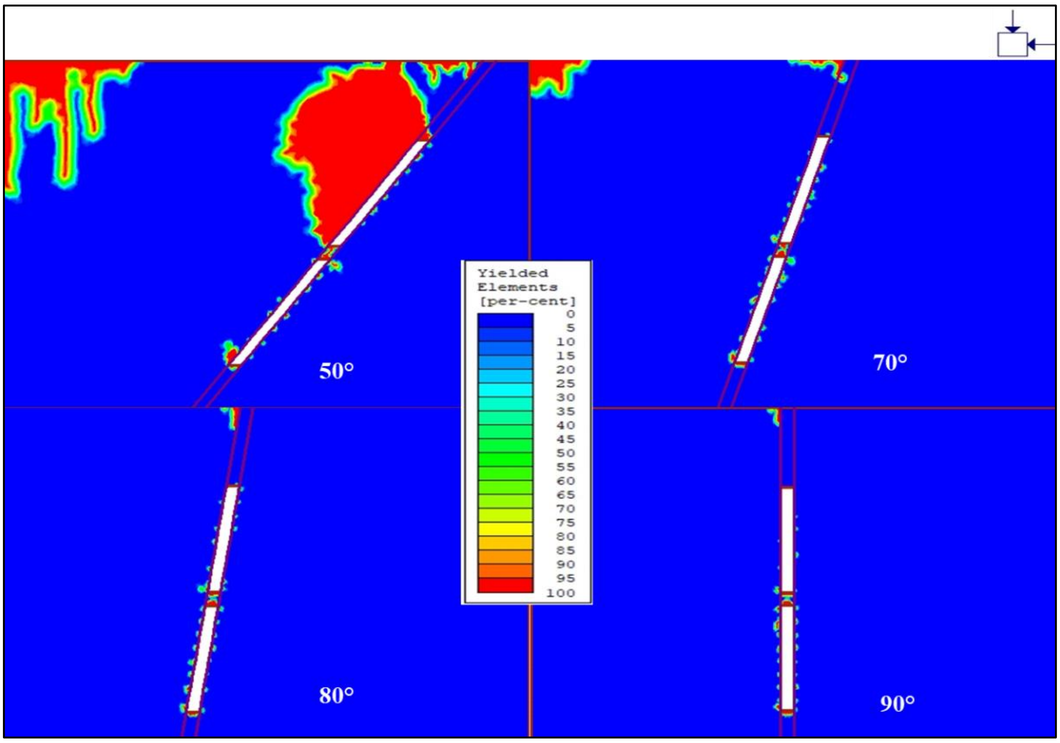


Figure 4.10. Simulation result of yielding elements at the crown pillar under various inclined veins with GSI 55.

4.3.4 Various thicknesses of the crown pillar

To propose the guidelines of how much ore should be left in the crown pillar, a parametric analysis was undertaken to evaluate the deformation behavior of the crown pillar under different thickness. All the analyses were carried out at 80° dip of the inclined vein with GSI value of 30. Prior to the parametric analysis empirical values of the crown pillar were determined using Equation 4.1, which was developed by (Carter, 2014) and is widely applied.

$$Cs = \left[\frac{\gamma}{T\left(1+\frac{S}{L}\right)(1-0.4\cos\theta)} \right]^{\frac{1}{2}} \tag{4.1}$$

Where Cs represents the scaled crown span, S represents the span, T is the thickness of crown pillar, L is represents length, Ɵ is the dip of foliation zone and ore body, and γ is the crown pillar rock mass unit weight. Therefore, the simulations were created with different thicknesses of crown pillars viz. 5 m, 10 m, 15m, 20 m, 25 m, and 30 m in height. The results of the analysis are presented Figure 4.11, Figure 4.12, and Figure 4.13. In Figure 4.11 the results of yielded zone is large compared to cases when the GSI is 46 and 55. But with respect to the crown thickness, the 5 m thick crown pillar has a larger yielded zone, which gradually reduces as the thickness of the crown pillar sequentially increases to 10 m up to 30 m. Based on simulation results, the instability of around the crown pillar is dramatically decreased at ≥ 20 m thickness with GSI 30. Nevertheless, the stope area and the crown pillar still experience significant yielding potential which may need artificial support.

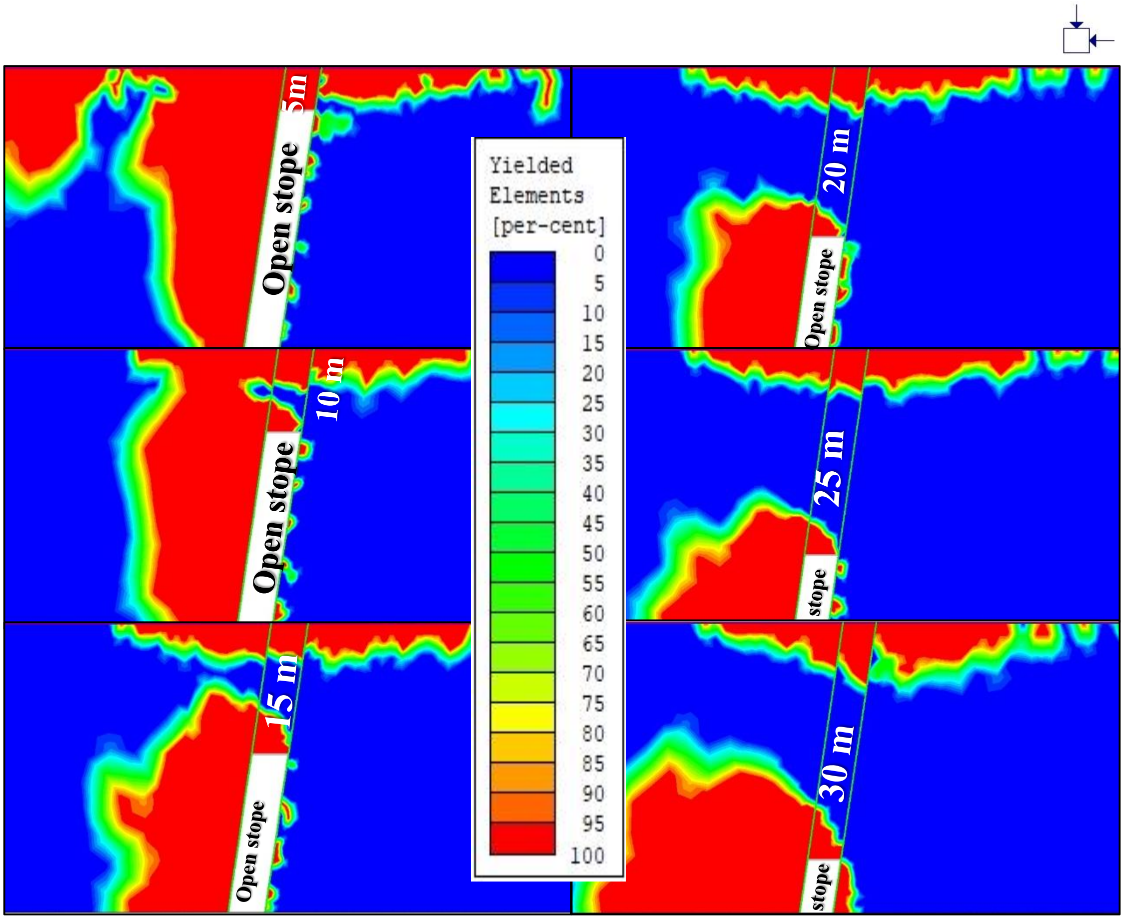


Figure 4.11. Simulation results of the crown pillars stability under various thicknesses with GSI 30.

On a different note, Figure 4.12 shows that induced stress and the yielded zone are high at 5 m both around the crown pillar and the stope area. However, beyond 5 m, induced stress and the yielded zone are less concentrated around the crown pillar and get redistributed around the stope area. Meanwhile, the simulation results for GSI 55 presented in Figure 4.13 shows a similar phenomenon with simulation results for GSI 46 at 5 m crown pillar thickness but beyond 5 m it is dissimilar. At crown pillar thickness of 10 m and above the induced stress and the yielded zone are concentrate in the hanging wall side of the crown pillar where the rock mass is weak and the areas around the excavated stope are more stable. According to the simulation results, effective stability of the crown pillar can be well controlled by using artificial support.

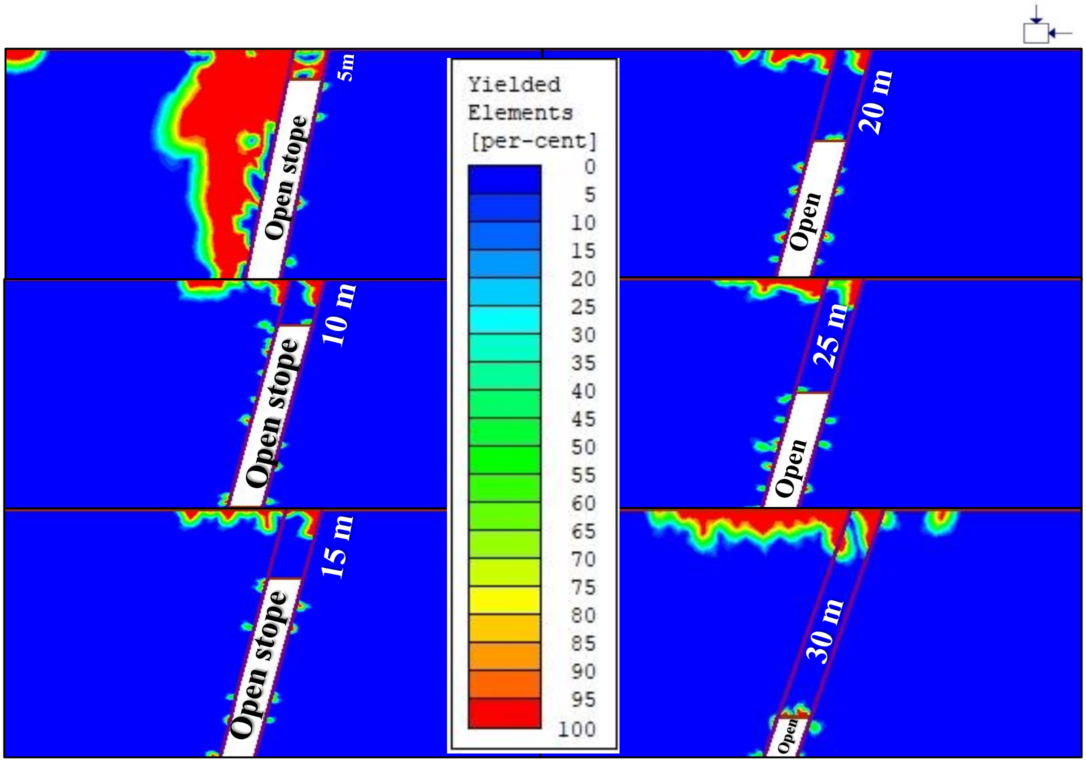


Figure 4.12. Simulation results of the crown pillars stability under various thicknesses with GSI 46.

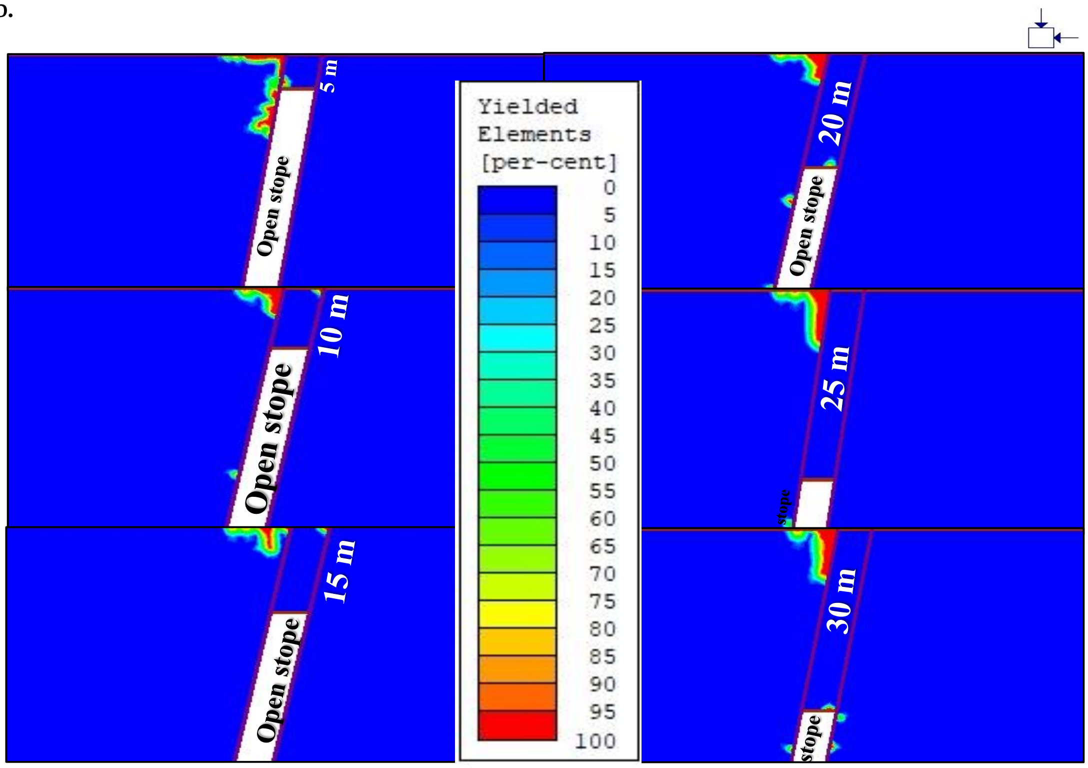


Figure 4.13. Simulation results of the crown pillars stability under various thicknesses with GSI 55.

4.4 Optimization of the Instability of Hermyingyi (Sn-W) Deposits Mine

From the preceding results, it can be observed that stability of crown pillar varies with the geological conditions. At the GSI value of 30, a crown pillar thickness of ≥ 20 m is recommended, at the GSI value of 46 a crown pillar thickness of ≥ 15 m is implementable and at GSI value of 55 a crown pillar thickness of ≥ 10 m is implementable. Although instabilities around the crown pillar still occur even at the largest crown thickness, they are manageable at such steep dipping inclined vein. However, the risk of higher instabilities is associated with gentle dipping inclined vein. Therefore, countermeasures need to be adopted to increase the stability of the crown pillar and the stope opening. There are a number of support systems that can be adopted. Hoek & Wood (1987) recommend the application of rock support using a combination of active and passive types, whereas Sasaoka et al. (2015) argue that the filling method is the effective support system of controlling the instability of the open stoping method. Therefore, in this research both scenarios were evaluated to determine the most effective intervention. The model for simulation is created with 5 x 5 m stope, 30 m in thickness of the crown pillar, raging GSI 30 to GSI 55 with 70° inclined vein.

4.4.1 Support using filling material

The application of filling material is one of the utilized countermeasures for optimizing the instability of stope and crown pillar in an underground mine. Therefore, the various types of filling materials are installed to control the failure of stope at the crown pillar in the shallow underground. The properties of the filling material are shown in Table 3.2. There are four types of filling materials that can be used, which include cement paste fill (CPF), cement rock fill (CRF), hydraulic fill, and waste rock fill.

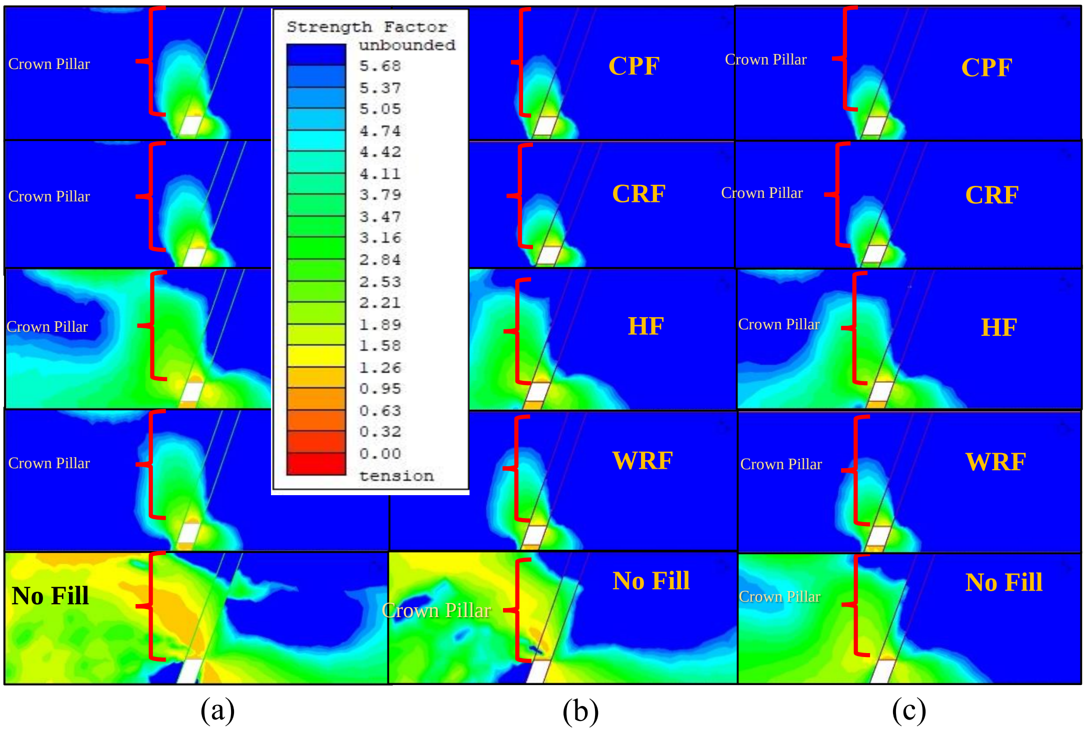


Figure 4.14. Simulation results of strength factor of stope at crown pillar with a different type of fill material as (a) GSI 30, (b) GSI 46, and (c) GSI 55.

Figures 4.14 (a), (b), and (c) show the stability of stope and crown pillar with various types of filling material for stabilization with GSI 30, 46, and 55. According to the results, the instability condition is critical prior to installation of support system. The strength factor is extremely reduced indicating high instability around the crown pillar especially in the hanging wall side. This implies a potential high risk of collapse for both stope area and crown pillar, which is the actual mine condition of open stoping at Hermyingyi (Sn-W) Deposit Mine. After the introduction of different filling materials, the strength factor increases indicating low instability both in stope opening and crown pillar. In terms of efficacy, the CPF, CRF and WRF types of filling material are more effective in reducing instability than the HF filling material. Comparing CPF, CRF and WRF, it can be observed that CPF and CRF have excellent stabilizing performance than the WRF. However, the WRF is a more recommended stabilizing method in the economic point of view because it is cheaper than the CPF and the CRF since the materials can be obtained from the dumping site near the mined-out area. As anticipated, comparison of the GSI values, the induced stress is dramatically decreased at GSI 55 at the crown pillar as shown in Figure 4.14 (c).

4.4.2 Support using support systems

Even though using the filling material method is effective especially CPF, CRF and WRF types of filling material, we noted that failure still occurs around the crown pillar of the stope. Thus the support system is installed in the open stoping method. To understand the effectiveness of active type of rock support as a countermeasure for both stope and crown pillar instability. The support system is installed in the model to stabilize the excavated stope and the crown pillar under the influence of environment stress distribution around the tin-tungsten deposit mine. Two types of support systems were installed to support the stope and crown pillar. The first is the active support using cable bolts and the second is passive support using shotcrete and the properties of support are given in Table 4.1. The active type of support has a different working principle from the active one. Its system is external to the rock and responds to inward movement of the rock around the excavation (Hoek & Wood, 1987). So, the active support provides a reactive force to the excavation boundary due to inward movement (Thompson & Windsor, 1993). This type of rock support stabilizes the stope by limiting the displacement that may further deteriorate the rock mass properties. However, the active type of rock support has very limited application such as in very loose ground and rock. In this condition, the application of an active support type support system might not be effective since it will not anchor properly. Under the high-stress condition, the displacement can be observed at the boundary of opening and the rock will continue movement even after being support by the active type support. Therefore, the application of passive-type support will limit the movement in such conditions, thus improving the opening stability.

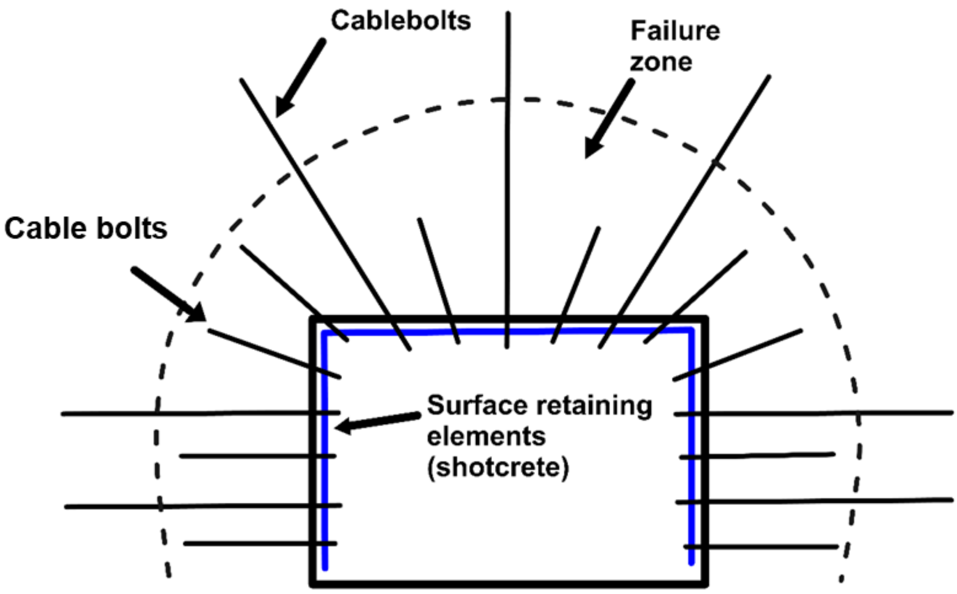
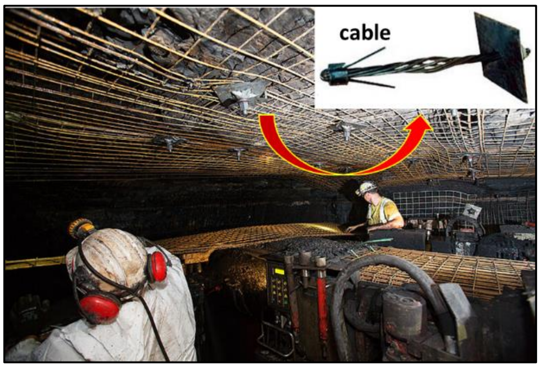


Figure 4.15. Methodology of installation for supporting system in the stope and the crown pillar.

There are 3 kinds of methodology for installation of the support system that can be used as shown in Figure 4.15. The procedures of installing are as follows: (1). Install shotcrete only, (2) Install cable bolt only, and (3) Install combined shotcrete and cable bolt. The shotcrete is installed with 0.1 m thickness to the wall of the stope, and the cable bolt is installed with 5 m length, and1 m spacing for each stope. Figure 3.16 shows the configuration of support in an underground mine.



(a)

(b)

Figure 4.16. Configuration of supporting in underground mine: (a) cable bolt support, and (b) shotcrete support.

Table 4.1. Properties of cable bolt and shotcrete supporting.

Cable Bolt Properties	
Type	Fully Bonded
Diameter (mm)	19
Cable Modulus (MPa)	200,000
Cable Tensile Capacity (MN)	0.1
Cable Residual Tensile Capacity (MN)	0.01

Shotcrete Properties	
Young's Modulus (MPa)	21,000
Poisson's ratio	0.15
Uniaxial Compressive Strength (MPa)	35
Tensile Yield (kN)	20
Residual Yield (kN)	0.01

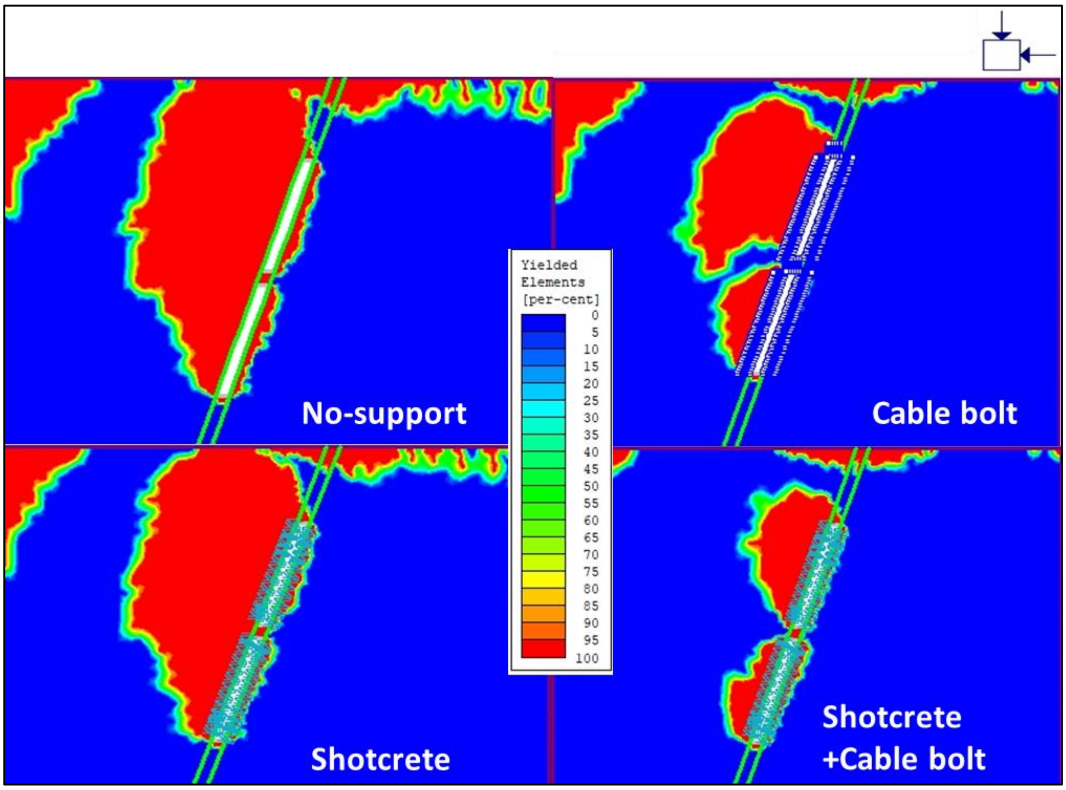


Figure 4.17. Simulation result of the instability of the stope area and the crown pillar with the open stoping method by active and passive supporting system with GSI 30.

The simulation results comparing the no-support and support conditions are given in Figure 4.17. In case of no-support, the induced stress becomes higher around the stope and the crown pillar because of the concentration of stress from the open excavated stope. When a combination of shotcrete and cable bolt is installed as shown in Figure 4.17, the yielded elements are greatly reduced compared to installation of shotcrete only or cable bolt only. The reduction in the stress and yielded zone happens both in the excavated area and around the crown pillar. The cable bolt only support type is also observed to be relatively more effective than the shotcrete only support type but instability is still high in the crown pillar area.

Figures 4.18, and 4.19 present the instability results of the open stoping method using the supporting system with GSI 46, and GSI 55. The simulation results of the instability of the open stoping method with GSI 46 and GSI 55 show that cable bolt only type is as effective as the combination of shotcrete and cable bolt around the excavated area although small yielding zone is observed around the crown pillar. This is because the rock mass is strong enough to respond to the binding effect of the cable bolts. Thus, in jointed hard rock masses, the application of cable bolt only support type is recommended. However, in weak and broken rock mass the application of cable bolt only type of support may not be effective.

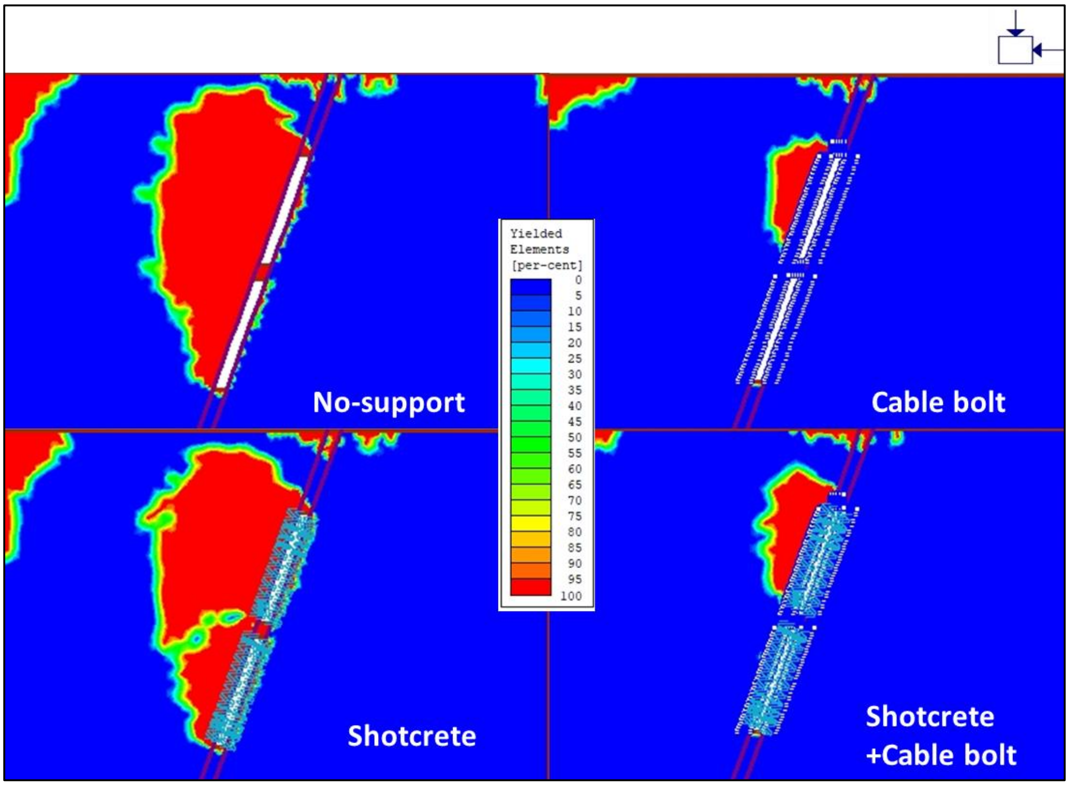


Figure 4.18. Simulation result of the instability of the stope area and the crown pillar with the open stoping method by active and passive supporting system with GSI 46.

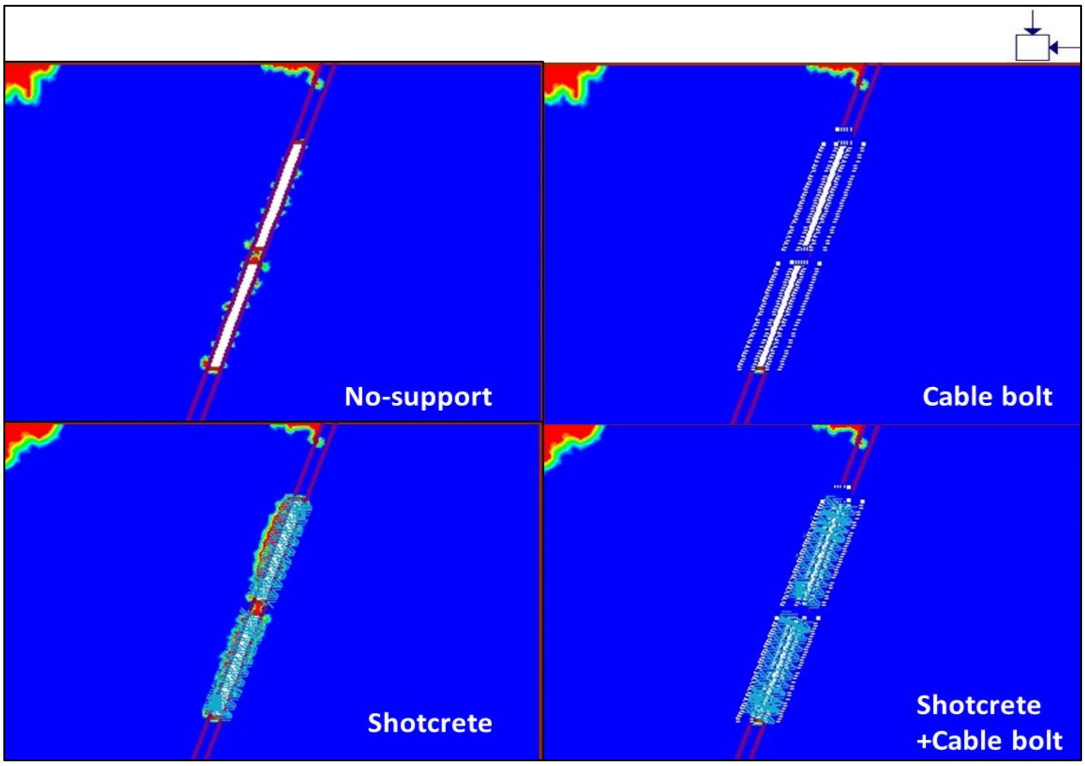


Figure 4.19. Simulation result of the instability of the stope area and the crown pillar with the open stoping method by active and passive supporting system with GSI 55.

Figure 4.20 shows the results of the crown pillar supported by the active and passive support systems for a model with 30 m thickness of the crown pillar by comparing GSI values of 30, 46, and 55. As expected, the results show that the yielded zone is extremely large around the crown pillar in no-support scenario. However, the collapsing potential can be greatly reduced by using the combined support system (cable bolt and shotcrete) especially at low GSI values i.e. GSI 30. At high GSI values, that is GSI 46 and GSI 55 the use of cable bolt only support type would be more effective and economical. With respect to geological conditions, good rock masses require less supporting needs than the poor rock masses. Thus, the guideline in the section of the support system to be installed, has to be guided by the economic aspect, the value of the ore to being extracted, the rock mass condition and most importantly the efficacy of the method in restoring stability of the crown pillar.

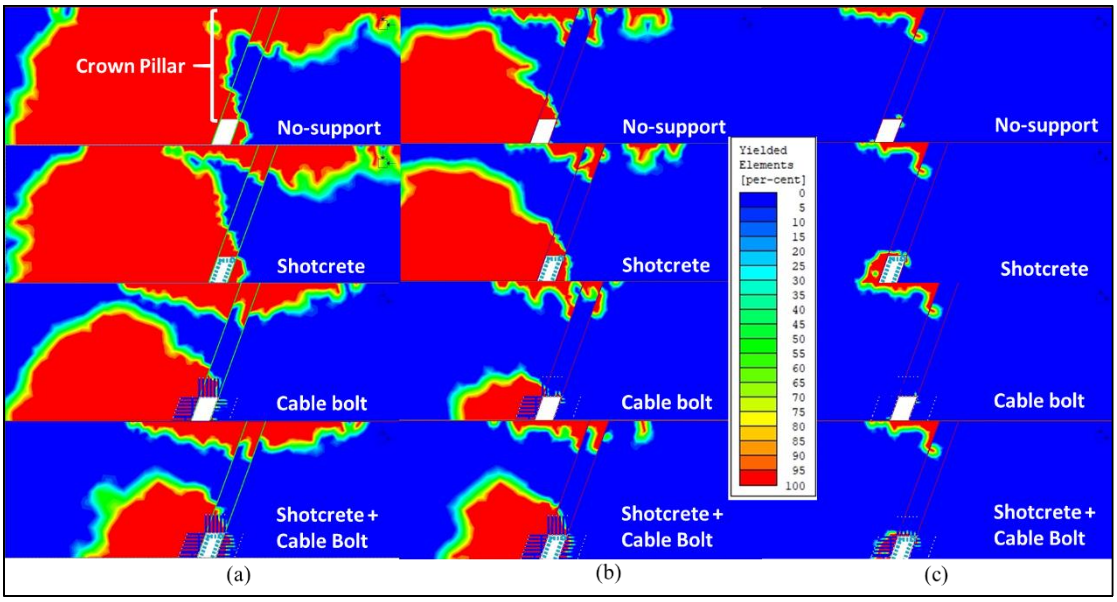


Figure 4.20. Simulation result of the instability of the crown pillar by supporting system with (a) GSI 30, (b) GSI 46, and (c) GSI 55.

4.5 Conclusion

In conclusion, this chapter evaluated the stability conditions of the crown pillar under different geology and structural conditions of the mine area with the aim of providing the guidelines for the selection of the thickness of the crown pillar. From the results, the geological conditions of the rock mass, and the inclination of the ore vein have a strong influence in the selection of the remaining thickness of crown pillar. Generally, the more inclined the dip of the vein is, the more dangerous it is to the stability of the crown pillar. Similarly, the weaker the geological condition of the rock mass, the high the risk of collapse of crown pillar. Thus, when designing the thickness of the crown pillar, the aforementioned aspects need to be carefully considered. From this study case, the following crown pillar designing guidelines are recommended: in weak rock masses a crown pillar thickness of ≥ 20 m in steep dipping ore vein e.g. ≥ 80° can be left with minimal instability risk which can be secured by supporting systems. On the other hand, in the similar weak rock masses with gentle dipping vein angle ≤ 70° the crown pillar thickness of ≥ 30 m should be left to prevent the collapsing of the mine. In hard rock masses, the crown pillar thickness of 15 m to 20m is recommended.

The small thickness is suited for a near vertical dipping ore vein while the large thickness is suited for gentle dipping vein.

In the case where the crown pillar hosts more valuable ore which may be required to be extracted, support systems have been proposed as countermeasures to crown pillar instability and/or collapse. In very weak geological conditions where artificial active support is not effective, the use of waste rock filling, cement paste filling or cement rock filling can be adopted in an upward sequence stopping until 15 m thickness of crown pillar. Meanwhile, in blocky rock masses where active support is feasible, a combined support system of cable bolt and shotcrete is recommended until 10 m thickness of the crown pillar. While hard rock masses affected by discontinuities or fracturing, the cable bolt only support system is proposed for implementation to stabilize the crown pillar. Thus, the decision in the implementation of the support systems need to properly consider the economic aspect, the value of the ore to being extracted, the rock mass condition and most importantly the efficacy of the method in restoring stability and maintaining of the crown pillar.

4.6 References

Betournay, M. C. (1989). What We Really Know About Surface Crown Pillars. *Procd. Surface Crown Pillar Evaluation of Active and Abandoned Metal Mines*, (pp. 17-34). Timmins.

Brown, E. T., & Hoek, E. (1978). Trends in relationships between measured rock in situ stresses and depth. *Int. J. Rock Mech. Min. Sci. & Geomech.*, 211-215.

Carter, T. G. (1989). Design Lessons from Evaluation of Old Crown Pillar Failures. *Procd. Surface Crown Pilalr Evaluaiton of Active and Abandoned Metal Mines*, (pp. 177-187). Timmins.

Carter, T. G. (2014). *Guidelines for use of the Scaled Span Method for Surface Crown Pillar Stability.* Toronto: Golder Associates.

Hoek, E., & Brown, E. T. (1978). Trends in Relationship Between In Situ Rock Stresses and Depth. *Int. J. Rock Mech. Sci & Geomech*, 211-215.

Hoek, E., & Brown, E. T. (1988). The Hoek-Brown Failure Criterion - A 1988 Update. *Proceedings of the 15th Canadian Rock Mechanics Symposium* (pp. 31-38). Toronto: Civil Engineering Department, University of Toronto.

Hoek, E., & Wood, D. F. (1987). *Support in Underground Hard Rock Mines.* Montreal: In Udd, J., Ed., Undergound Support Systems, Canadian Institute of Mining and Metallurgy.

Kwon, S., Cho , W. J., & Lee, J. O. (2013). An analysis of the thermal and mechanical behavior of engineering barriers in a high-level radioactive waste repository. *Nuclear Engineering and Technology, 45(1)*, 41-52.

Sasaoka, T., Takamoto, H., Shimada, H., Oya, J., Hamanaka, A., & Matsui, K. (2015). Surface sub-sidence due to underground mining operation under weak geological in Indonesia. *International Journal of Rock Mechanics and Geotechnical Engineering*, 337-344.

Tavakoi, M. (1994). *Underground Metal Mine Crwon Pillar Stabilty Analysis.* Wollongong: Research Online.

Thompson, A., & Windsor, C. R. (1993). Theory and Strategy for Monitoring the Performance of Rock Reinforcement. *Proceedings Geotechnical Instrumentation and Monitoring in Open Pit and Underground Mining* (pp. 473-482). Rotterdam: Balkema.

CHAPTER FIVE

5 Stability of Multiple Veins Excavation in Tin-Tungsten Deposits Mine

5.1 Introduction

In the previous chapter the instability of single vein excavation and its related results have been discussed as well as the instability for both stope and crown pillar. This chapter is intended to evaluate the effective approach to excavate multiple veins in shallow overburden and high-stress distribution around the tin-tungsten deposit mine. The purpose is to develop guidelines of the design based on the effect of instability under multiple excavations at the Sn-W Deposit Mine. Moreover, it provides the strategy to promote the production of tin-tungsten hard metals since these metals have been expansive such that the increasing price and demand for tin-tungsten metal caused by global economic growth and exploitation of Sn-W Deposits. In several mines, ores located at the higher level or near-surface contains high grade than ore at the lower level. Moreover, Hermyingyi Sn-W Deposit Mine, not only as an example but also multiple veins exists. Hermyingyi Mine was probably the largest tin-tungsten mine in the world in 1947, production amounted to 1,051 tons among the district's production of 3,653.5 tons. Therefore, this chapter was determines the instability of stope under the excavation of multiple veins compared to the previous chapter. Concerning the study of this chapter, the trend of veins is developed as parallel incline across to the mined-out area.

The results focus on the factor of safety, failure criteria, and displacement of stope by FLAC^{3D}ver. 5.0 simulation. But the different inclined vein and supporting system methods have been analyzed by Phase^{2} simulation for the stability of stope. Both simulations are utilized to determine the instability of stope. The multiple veins excavations application to stabilize the open stoping method and the cut and fill method will be introduced in this chapter. It will control the accumulation of induced stress at stope. The result will be given in the following geometry condition. To summarize, the mine design is introduced for the stability of stope at the tin-tungsten (Sn-W) Deposit Mine.

5.2 Excavated Area and Planning of the Mined-Out Area of Multiple Veins

The study area (Latitude 15°14′N, longitude 98°21′E) is located approximately 40 km northeast of Dawei Township, Thanintharyi Division, Southern Part in Myanmar shown in Figure 5.1 (a). The veins lie parallel across to the mine area at the research area as shown in Figure 5.1 (b). The research area is developed with under multiple veins system. Recently, mineral exploration has been conducted at multiple veins in the Sn-W Deposit Mine.

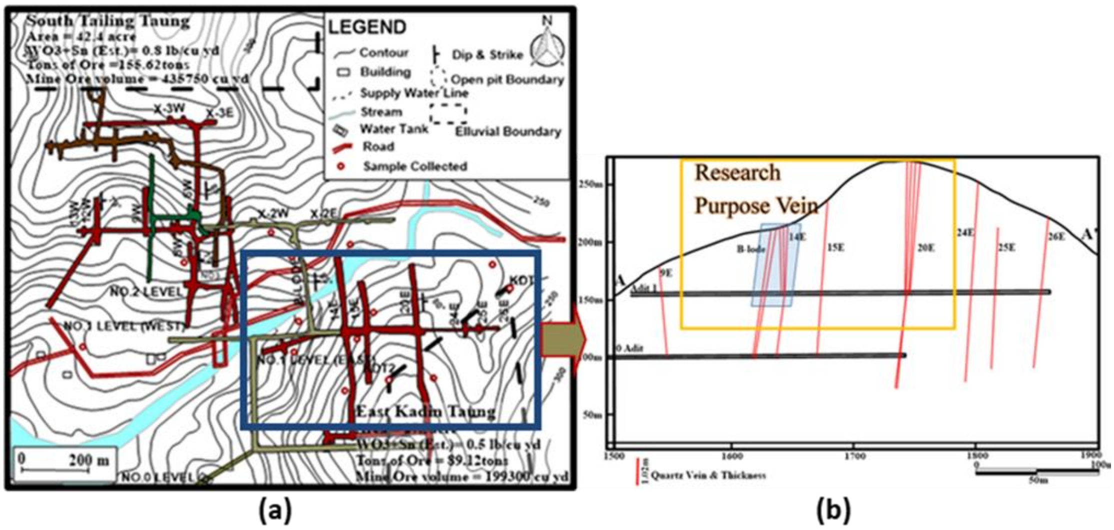


Figure 5.1. The planning out-cross map at Hermyingyi (Sn-W) Deposit Mine (modified after Aung Tun Oo, 2018, unpublished).

5.2.1 Numerical simulation

Figure 5.2 shows the model was applied with Mohr-Coulomb constitutive model in FLAC^{3D}5.0, the upper boundary of z-direction is free and the bottom of z-direction is fixed. For all vertical boundaries of x, y directions are fixed. The gravitational stress is applied. The mine excavated area is located 150 m in depth from the surface. The model was constructed 150 m in width, 150 m in length, and 240 m in height. The face of mine is inclined at 36° and the adit is adopted at 150 m in depth, in which, the excavation is 150 m in length, 2 m in height, and 2 m in width, respectively. The stope dimension is 30 m in length, 5 m in width, and 2 m in height. The excavation steps are 32 steps for all veins. The distance of interval in-situ rock (footwall) measuring 14 m is left between vein 1 and vein 2, and 20 m between vein 2 and vein 3. Some analysis, their distance of in-situ rock has different width due to the consideration of safety for stope stability and development of the vein system in the mine.

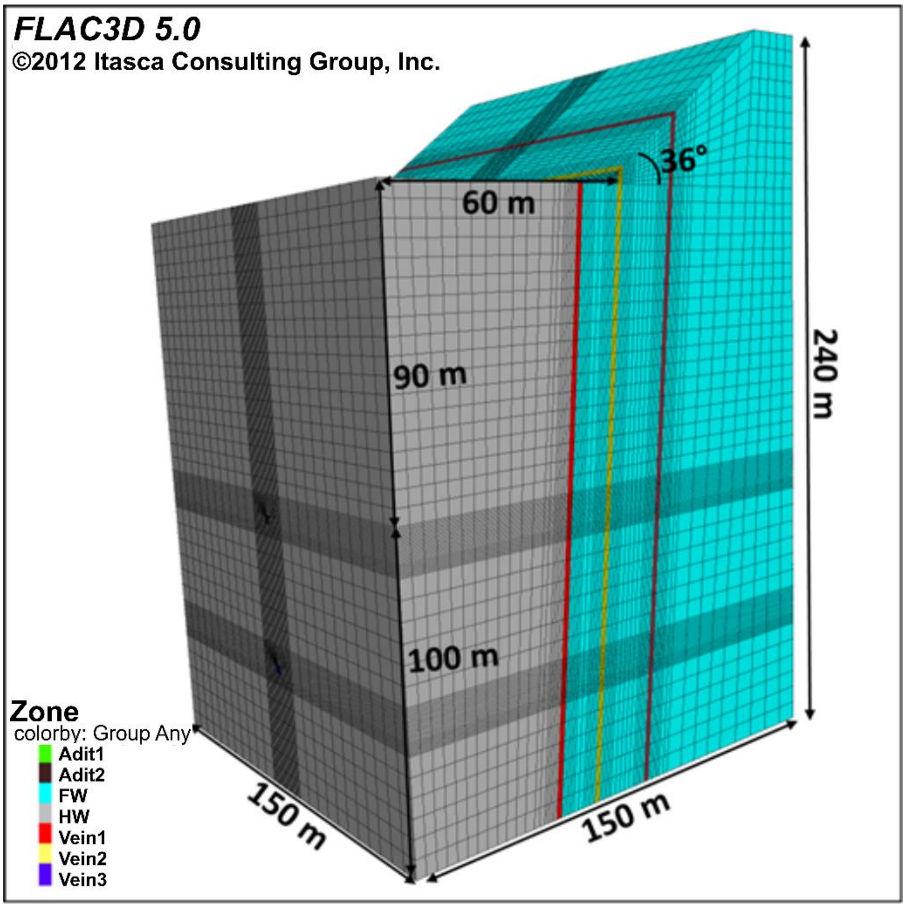


Figure 5.2. Constructed model with a detailed explanation of the whole block.

5.3 Evaluation of Stope Stability Under Filling and Without Filling Material

The stability of underground excavations has become an important issue in the underground mining operation due to mine enlargement and excavation of deeper mineral resources (Sasaoka, et al., 2015). The open stoping method and the cut and fill method are commonly used as underground mining methods in Myanmar because the ore body, which is from the primary source, is located at a considerable depth, especially at the Hermyingyi (W-Sn) Deposit Mine. The application of the open stoping method and the cut and fill method for underground mining encounter failures due to the geological structure and high regional stress conditions. Although the stoping method is cardinally influenced by environmental effects related to the induced stress of natural conditions and the structure of geometry, the quartz-vein type is a typical (Sn-W) Deposit in this study area. Therefore, controlling stope stability is crucial, not only for the safety of the mine-workers and transportation of equipment but also for the capacity of mine production capacity and profitability by minimizing unexpected ore body suspension.

Various research works on underground mining have focused on the interaction between underground openings. For instance, Purwanto et al. (2013) studied the influence of the stope design on hanging wall decline stability under multiple excavations. The study revealed that weakening rock mass conditions can lead to the initiation of instability. In relation to stress, the effects of large-scale stress adjustment and redistribution, hysteretic backfill, low strength of filling body, unfilled void space, as well as repeated mining activities were studied and showed that the conditions currently caused ground deformation (Zhao , Ma , Zhang , & Guo, 2013; Sepehri, Apel, & Liu, 2017). However, very few researches and publications are found on the influence of in-situ stress conditions on the stability of open excavation in a comparative approach.

Since the stability of mine openings depends on the mining method, induced stress, geology, and trend of the vein in the ground, numerical assessment has shown to be helpful to understand the situation of stability in the stoping area. In this study, the unstable displacement and the factor of safety of the stope are calculated with different stress ratios; 0.5, 1.0, and 1.5 because the regional stresses are high around this area, which could potentially control stability conditions at Hermyingyi (Sn-W) Deposit Mine. Thus this chapter discusses the stability of underground openings, especially, the stope beside of excavation area in different mining methods and mining conditions.

5.3.1 Methodology of stope excavated

Two types of underground methods are evaluated such as 1. The open stoping method and 2. The cut and fill method. Figure 5.3 (a) shows the mine excavated area is located 150 m in depth from the surface as conducted in FLAC^{3D}5.0. Figure 5.3 (b) presents the open stoping technique. The stope dimension is 50 m in length, 5 m in width, and 2 m in height. The excavation steps are 32 steps for all veins. The pillar measuring 14 m is left between vein 1 and vein 2, and 20 m between vein 2 and vein 3. Their distance of in-situ rock is different widths due to the consideration of safety for stope stability and development of the vein system in the mine. In the post-excavation the stopes are free space not filled with any material.

Figure 5.3 (c) shows the numerical model of the cut and fill method. The first step is excavated from the lower stope to the final stope, it is 8 steps and then continuously excavate to the second vein with a similar excavation pattern as the previous one totaling 11 steps. In the third vein, 13 steps are used in the same way to excavate in order to obtain the objective. This method is used to fill material after excavating stope before progressing to the next step. The monitoring points are pinned above each stope to evaluate prevailing conditions. The key aspect is to determine the failure initiation and progression when excavating at the three parallel inclined veins under different stress ratios.

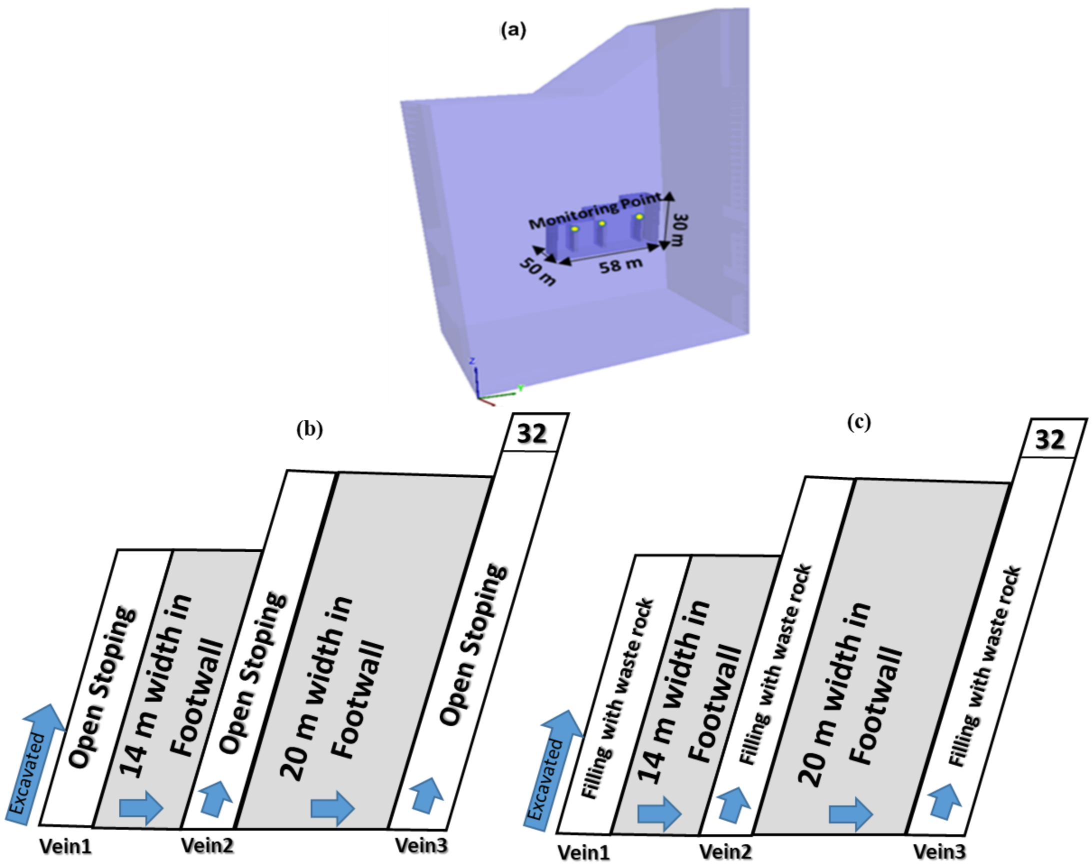


Figure 5.3. Systematic diagram for excavation; (a) Created the mined-out area, (b) The open stoping method, and (c) The cut and fill method at the Hermyingyi (Sn-W) Deposit Mine.

5.3.2 Results and discussions

5.3.2.1 Factor of safety (FoS)

While the essence of evaluation in this study is to determine the stability with the strength-to-stress interaction, it is generally acknowledged that higher stress develops in the roof of the intersection of the mined-out area. Therefore, the different stress ratios, K = 0.5, 1.0,1.5 are used to analyze the stability of the open stoping method and the cut and fill method. The factor of safety determination is vital at the ongoing mined-out area. Figure 5.4, presents two cases that are compared in which the open stoping method and the cut and fill method are under different stress ratios, K = 0.5, 1.0, 1.5. According to the results, it can be said that the stability around the stope is low at K = 0.5. The trend of FoS dramatically increased under stress ratio, 0.5 in both cases. In comparison with the open stoping method and the cut and fill method under stress ratio, K = 0.5: The open stoping method is shown in Figure 5.4 (a). The FoS value, in this case, is 0.9 where the instability is expansive around the excavated area, whereas the first stope of hanging wall and the third stope of footwall failure is observed at 9-11 m in width. Moreover, the failures appear to have occurred around the pillar in all stopes. Calculating the step by step of the stope, the failure is found at the crown pillar for each stope. On the other hand, in the cut and fill method, FoS value is 1.35. Based on the results, the failure zone is developed around the first and the second stopes, and the stability of the interval of footwall decreases due to the connecting failure zone. The third stope is not much affected but the failure of the crown pillar. Hence, it can be said that the application of the cut and fill method can improve the stability of the stope obviously, but 20 m or more thicker intervals of footwall should be left for safe operation under the stress ratio of K = 0.5.

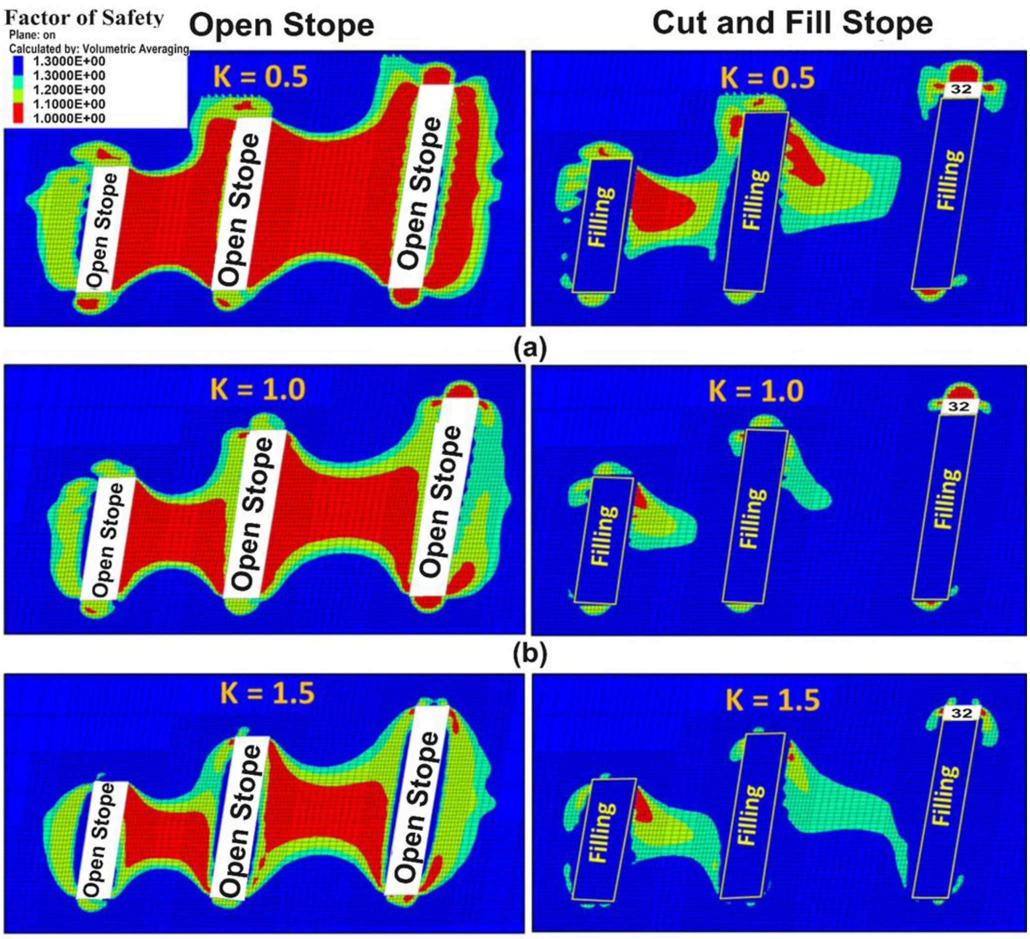


Figure 5.4. Demonstrated the result of FoS in the study area (a) Stress ratio, K = 0.5, (b) Stress ratio, K = 1.0, and (c) Stress ratio, K = 1.5.

Figure 5.4 (b) shows the stress ratio, K = 1.0. Comparing the two types of mining methods: the results show that the FoS value is 1.1 for the open stoping method and 1.45 for the cut and fill method. The failure appears to have developed at 5-6 m thick in the hanging wall of the first stope and the footwall of the third stope under the open stoping method versus 3-4 m thickness in the hanging wall of the first stope and the footwall of the third stope under the cut and fill method. The failure of the interval of footwall appears connected with all stopes in the open stoping method but the cut and fill method are not connected. Hence, it can be said that the application of the cut and fill method can maintain the stability of stopes, and 14 m or less interval of footwall is enough for safe operation under the stress ratio of K = 1.0. Figure 5.4 (c) presents the results of K = 1.5. The FoS in the case of the open stoping method is 1.3 and that of the cut and fill method is 1.5. The stability of the stope increase with increasing the stress ratio.

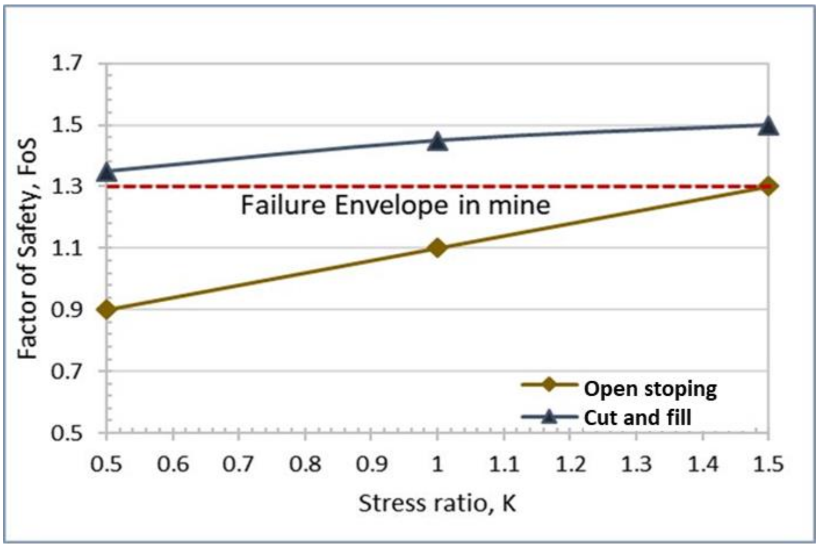


Figure 5.5. Comparison of the FoS with the open stoping method and the cut and fill method.

The comparative trend of stability for the two methods, based on FoS, is represented in Figure 5.5. From the outcomes, it can be deduced about the Sn-W deposit that the open stoping method has an increased risk of instability as reflected in the FoS, which is below the benchmarked value of 1.3. Thus, it can be concluded that the application of the cut and fill method and/or width of interval footwall between the vein should be considered carefully based on the in-situ stress conditions.

5.3.2.2 Horizontal displacement

Mining activities induce volumetric as well as stress-strain changes in the rock mass. Once deformation surpasses the limits controlled by the rock strength, instability is generated on the mining excavation. To manage unprecedented failures, it is crucial to record the stability of rocks surrounding the stopes. Unlike when mining operations are in progress where monitoring is carried out periodically and sometimes automatically using reference points, at the simulation level, this was achieved by pinning monitoring points on strategic positions.

Measuring displacement at the underground ongoing mine revealed risky conditions against failure. The contours of horizontal (y-direction) displacement are shown in Figure 5.6. The contour line shows that the displacement value is increased if the contour scale is small. From Figure 5.6 (a), the minimum displacement of the open stoping method is 40 mm and the cut and fill method is 31 mm at the stress ratio, K = 0.5 as monitored around the footwall. Besides, the maximum horizontal displacement of the open stoping method is 46 mm, and the cut and fill method is 37 mm, these results are recorded in the hanging wall. All the results are based on the final excavation step of 32.

Figure 5.6 (b) shows the horizontal (y-direction) displacement results at stress ratio, K = 1.0. The results demonstrate that the maximum displacement is 31 mm based on the open stoping method, 25 mm based on the cut and fill method, moreover, the minimum displacement is 27 mm for Open stoping and 21 mm for the cut and fill method. Figure 5.6 (c) presents the displacement conditions at stress ratio, K = 1.5. The outcome indicates that the maximum horizontal displacement is 18 mm for Open stoping and 14 mm in the cut and fill method and the minimum horizontal displacement is 16 mm for the open stoping method and 12 mm for the cut and fill method. After comparing the outcome, the displacement is exactly increasing around the stope in the open stoping method relative to the cut and fill method. These displacement findings are in agreement with a study by Sepehri, Apel, & Liu, 2017 who showed that the horizontal stress to vertical stress ratio has a significant impact on the development and propagation of the relaxation and yielding zones around underground openings such that under smaller gravitational loading the relaxation increases leading to reduced impact of yielding.

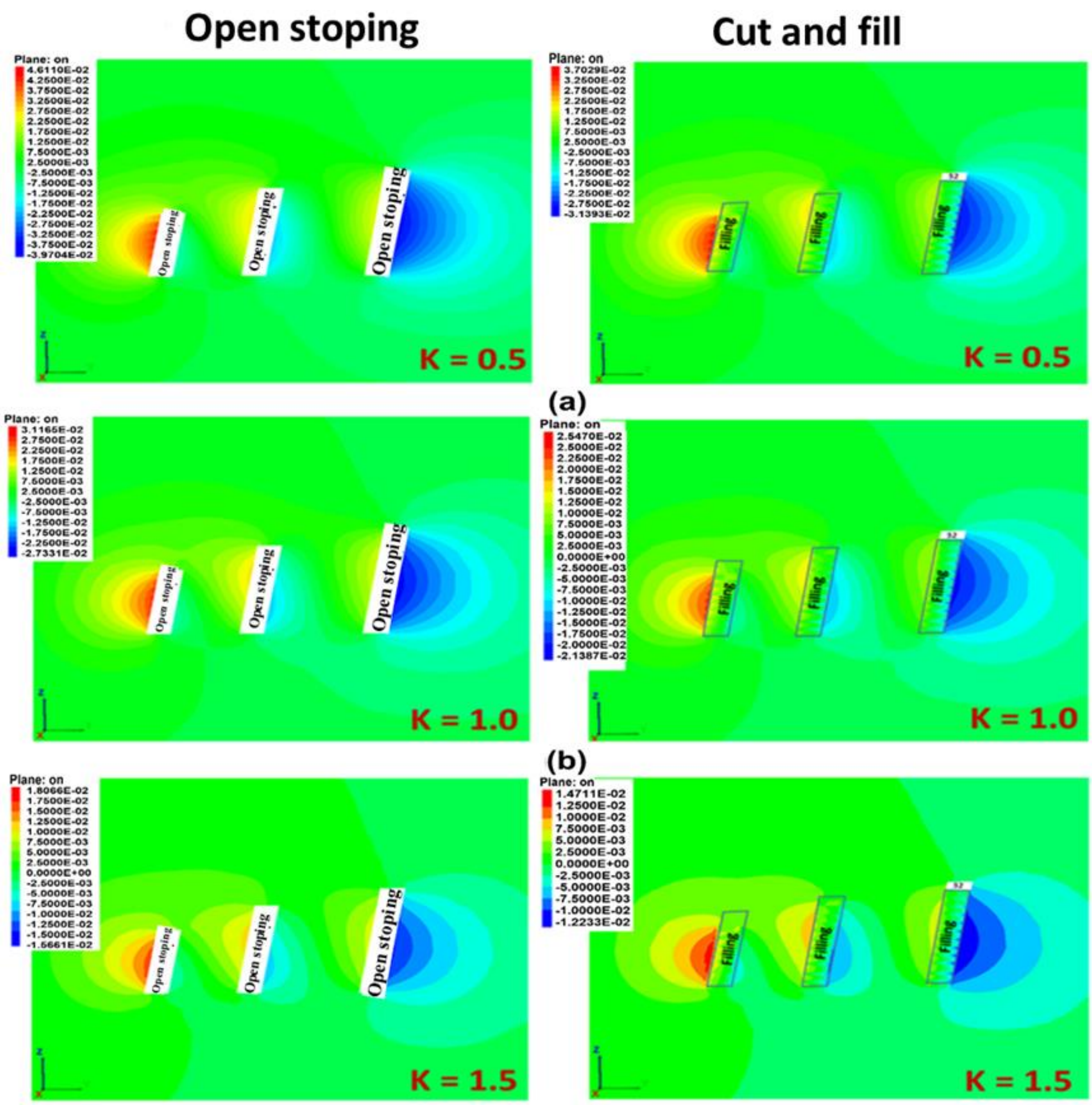


Figure 5.6. Horizontal displacement results in the study area (a) Stress ratio, K = 0.5, (b) Stress ratio, K = 1.0, and (c) Stress ratio, K = 1.5.

Therefore, the result should be considered for the excavation in the adjacent of mined-out area. The results of displacement correlate well with the stress direction as presented in Figure 5.7. The head of the stress direction is dipping into the excavated area and maximum stress came from the hanging wall of the first vein and the footwall of the third vein, but the second vein has minimal stress concentration relative to the two veins. Furthermore, a comparison of the two methods, the rocks surrounding the stopes show a significant risk of collapse into the stope under the open stoping method, but the cut and fill method can be controlled for the collapse of the surrounding rock mass of stope.

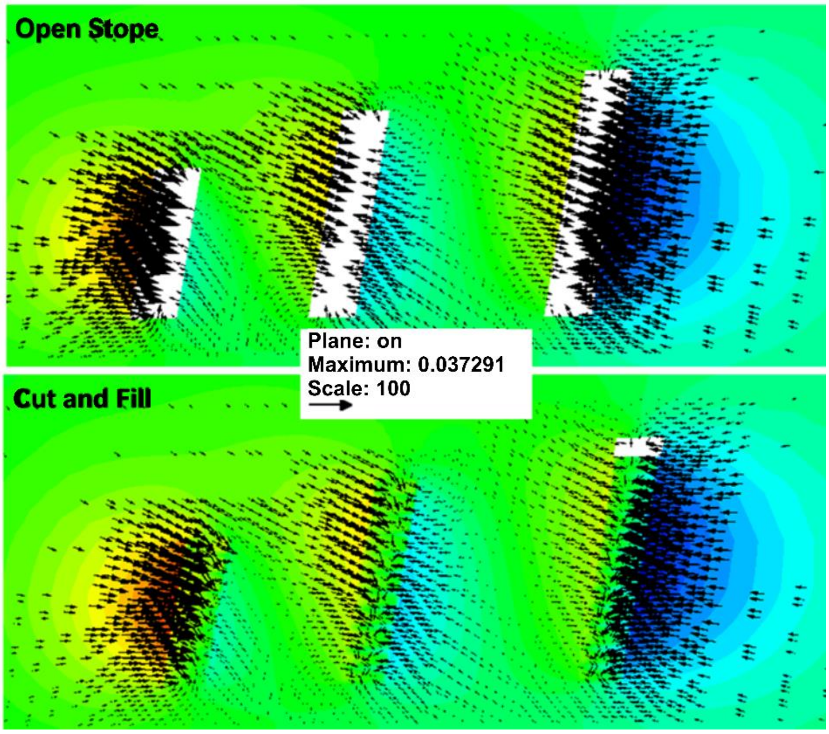


Figure 5.7. Stress distribution of the stope surrounding the stope in the underground.

5.4 The Effect of Multiples Ore Body Excavated with Cut and Fill Method Under Different Condition

The geometry of mine conditions has been analyzed such as stress ratio, GSI values, the width of the ore body, order of excavated steps, the width of adjacent rock between the vein, and the location of the mined-out area. Those conditions should be understood before the excavate the stope. Therefore, the researchers have been evaluated under this kind of various conditions. To estimate the instability of mine, the factor of safety, the failure criteria, and the displacement are applied in the model.

5.4.1 Process of excavation with numerical simulation

Concerning of excavation method, the sequences of stope excavate are 32 steps for all veins in Figure 5.8. The procedure of excavation is applied for an order from vein 1 to vein 3, it is 8 steps and then continuously excavate to the second vein with a similar excavation pattern as the previous one totaling 11 steps. In the third vein, 13 steps are used in the same way to excavate in order to meet the objective. The stope dimension is used at various heights, 50 m in width is used for all the results of research in this chapter. This method is used to fill material after excavating stope before progressing to the next step. The monitoring points are pinned above each stope to evaluate prevailing conditions. The Geological Strength Index, GSI values is adopted as 30 in the mine site, on the other hand, the vein inclined is used 80 ° for evaluating the instability of stope. The located of the mine site is 150 m from the surface. To achieve the objective in this chapter, the FoS, the failure zone, and the displacement are determined of stope stability.

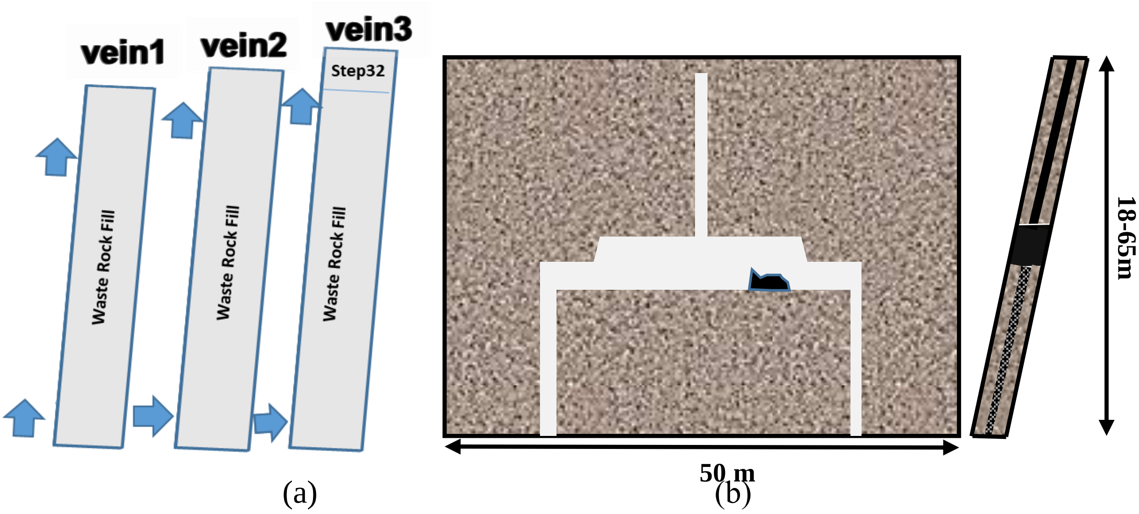


Figure 5.8. Methodology of excavation sequences for the cut and fill method: (a) The vertical plan view of excavation stope and (b) The cross-section view of excavation stope and it is related design.

5.4.2 Geological strength index

The first proposed instability of stope is assessed by using various GSI parameter. GSI values were simulated at GSI 25, 30, 38, 50, and 55, respectively for instability of stope in Sn-W Deposits Mine. Figure 5.9, shows the result of the instability of stope under various GSI values by the cut and fill method. Based on the results, the shear-p and shear-n tension-p is dramatically increased at GSI values from 25 to 38. This means that there is potential high risk of rock collapsed due to unstable condition of structural control and stress distribution of the environment. The GSI values, 46, 50, and 55 have high tension-p stage failure, implying that the regional force is counteracting the collapse of rock-fall from the side of the stope. Therefore, the failure of stope identified is principally controlled by the geological conditions, which demonstrates that rock mass condition is key to stop stability.

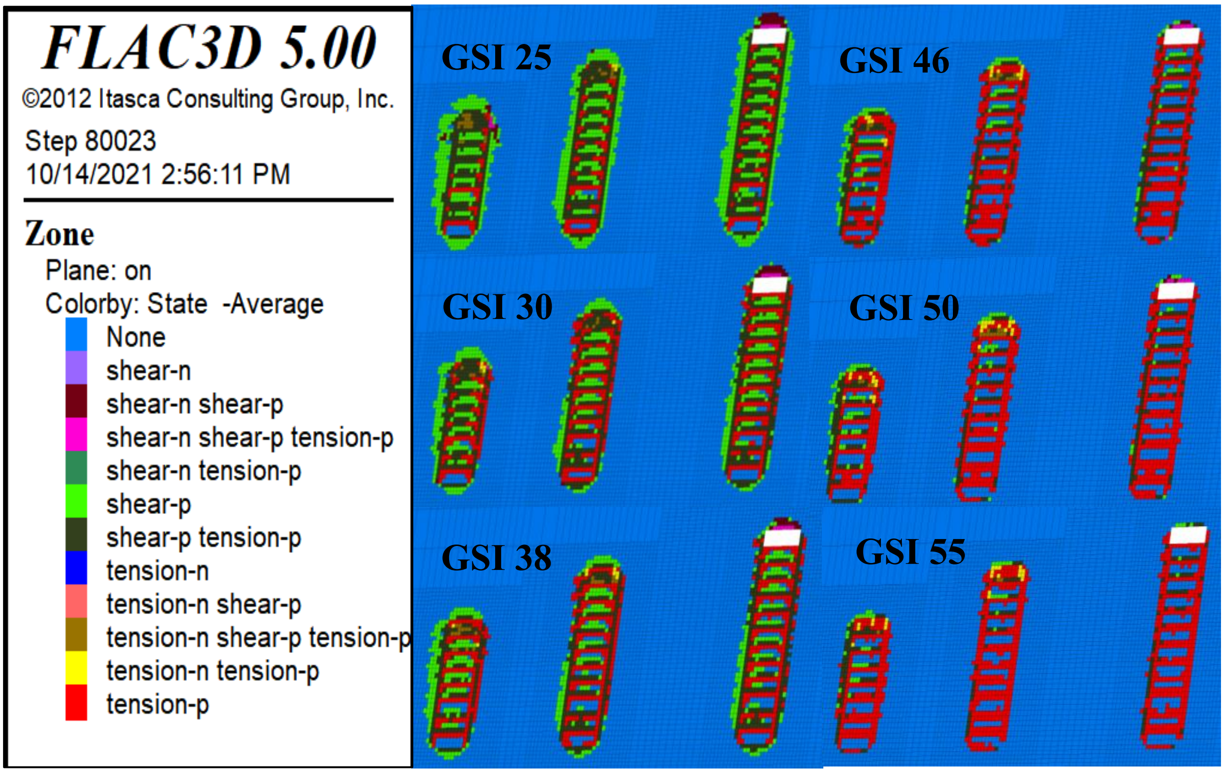


Figure 5.9. Simulation results of instability of stope under different GSI values.

Figure 5.10, shows the deformation condition of stope under different GSI values. The results can be said that the failure potential is inversely proportional to GSI, that is the failure potential decreases at GSI values increase and the reverse equally applies. Therefore, the research area is unstable for the multiples stope due to the geological condition because the actual GSI values for the Sn-W Deposit Mine at site is 30.

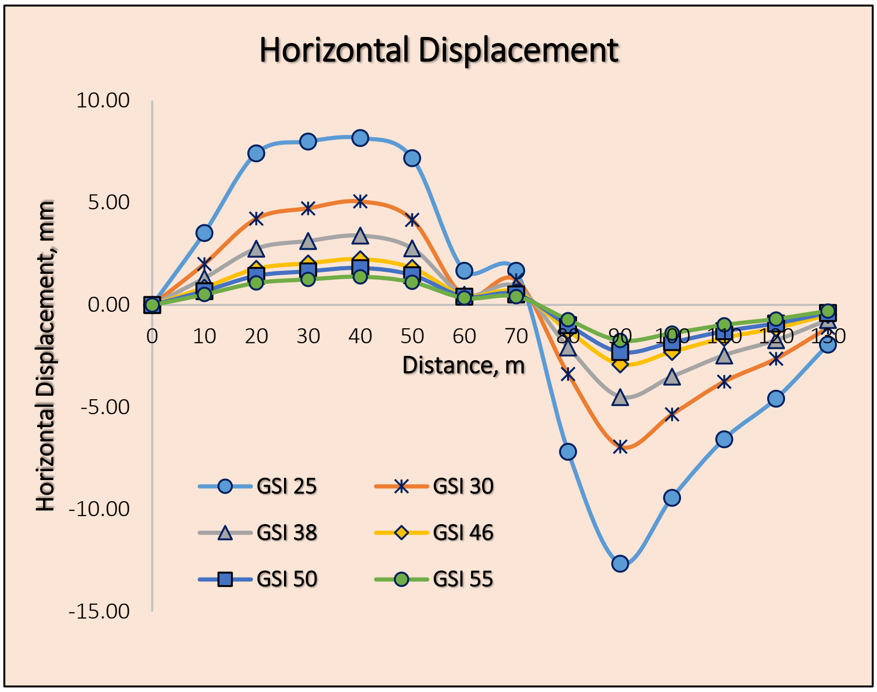


Figure 5.10. Compared to the displacement due to different GSI values.

5.4.3 Stress ratio

In the research area, the stress is widely distributed due to tectonic fault and Saging fault near the mined-out area. Moreover, the research area is located near the coastal region, where the regional stresses are adopted with the high-stress condition of mine. Therefore, the modified parameter is used stress ratio. The K ratio of horizontal stress to vertical stress ranged from 0.5 to 1.5 with GSI 30 at 80° parallel incline vein. The stress ratio below one indicates a high vertical stress condition around the stope due to a shallow underground mine. Figure 5.11 shows the failure mechanism and FoS of the cut and fill method under various stress ratios, K. The mode of shear-p failure is increased at K ratios 0.5 as shown in Figure 5.11 (a). There is high failure occurrence in the first vein with 10 m beside of the stope. For the second and third vein, the failure was observed around 4 m beside of the stope. When K ratio is 1.0 and 1.5, the results show that the failure zone is decreased than the K ratio of 0.5. This means that the vertical stress has high influence surrounding the stope of mine due to the shallow underground mine. Figure 5.11 (b) shows the simulation results of FoS of the cut and fill method under various stress ratios. The FoS has increased both surrounding the stope and among the vein at multiple veins excavated as K value 0.5, whereas the induced stress is more severe at the first excavated vein compared than the other vein excavated. By comparing the stress ratio, the FoS is reduced at 1.0 and 1.5 of K ratio. As a comparison of stress ratio result, the large failure zone and factor of safety can leave to severe between Vein 1 and Vein 2 excavated area, it was the attention of the excavate especially on K = 0.5.

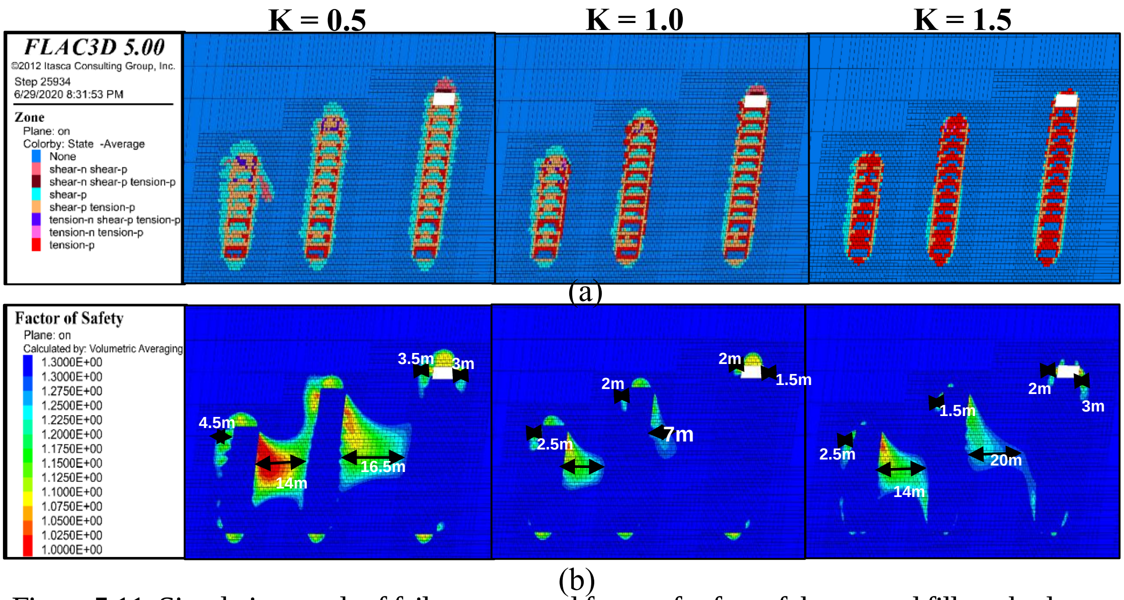


Figure 5.11. Simulation result of failure zone and factor of safety of the cut and fill method.

5.4.4 Different widths of ore body excavated

According to the simulation of previous results, the induced stresses still occur surrounding the stope area even when K value decreases. Therefore, the application of different widths of the ore body excavated is driven with K ratio 1.0 in Figure 5.12 and Figure 5.13. The numerical model is installed with different vein widths such as results of 2 m in width, 3 m in width, 5 m in width, and 2,3,5 m in width with GSI 30 at 80° parallel inclined vein, respectively. Figure 5.12 shows the determination of unstable stope based on various widths of the vein, based on the result, the failure zone becomes highest at 5 m in width of vein excavated due to the accumulation of induced stress from the stoping area. After comparing the results of different width, the failure is decreased at 2 m in width of vein.

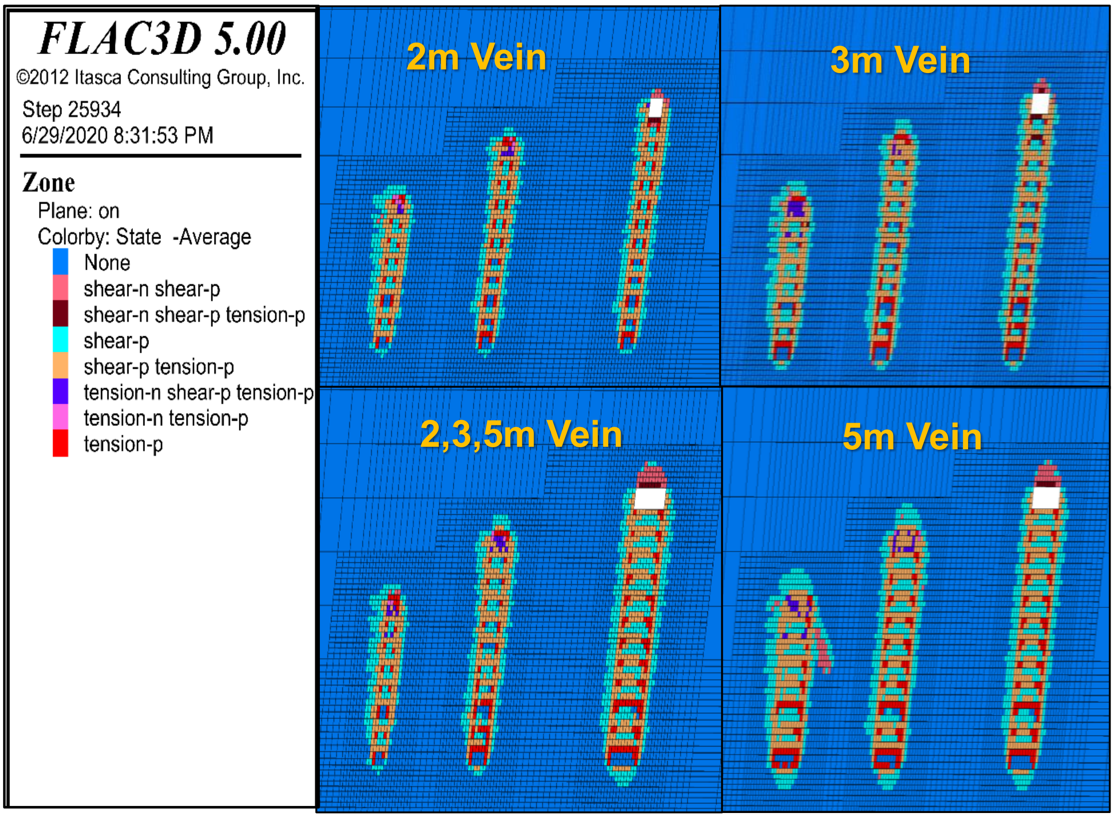


Figure 5.12. Simulation result of failure zone of the cut and fill method under various widths of the vein.

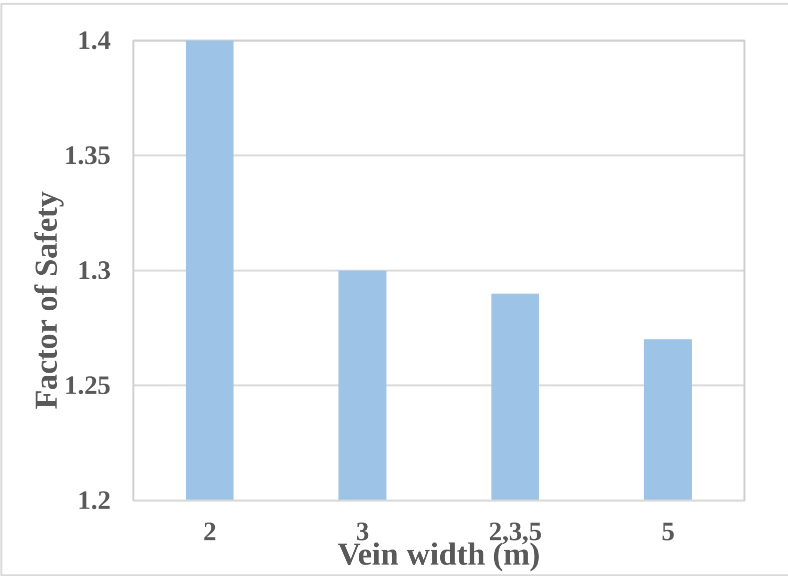


Figure 5.13. Simulation result of the factor of safety of the cut and fill method under various widths of the vein.

On the other hand, Figure 5.13 shows the simulation result of the factor of safety at surrounding the stope. Based on these results, the FoS of 1.3 is used for stability benchmarking. FoS value of 1.4, was obtained at all 2 m vein width, FoS 1.3 is observed at all 3 m vein width, FoS 1.29 is observed at 2,3,5 m vein width, and FoS 1.27 is observed at all 5 m vein width, respectively. As shown in the result vein width affect the stability of stopes because the factor of safety is the lowest at all 5m vein width falling below the benchmark than any other width combinations. Thus, it is important to control the excavation width of the stopes to ensure a high stability of the excavated stopes in underground tin-tungsten mine.

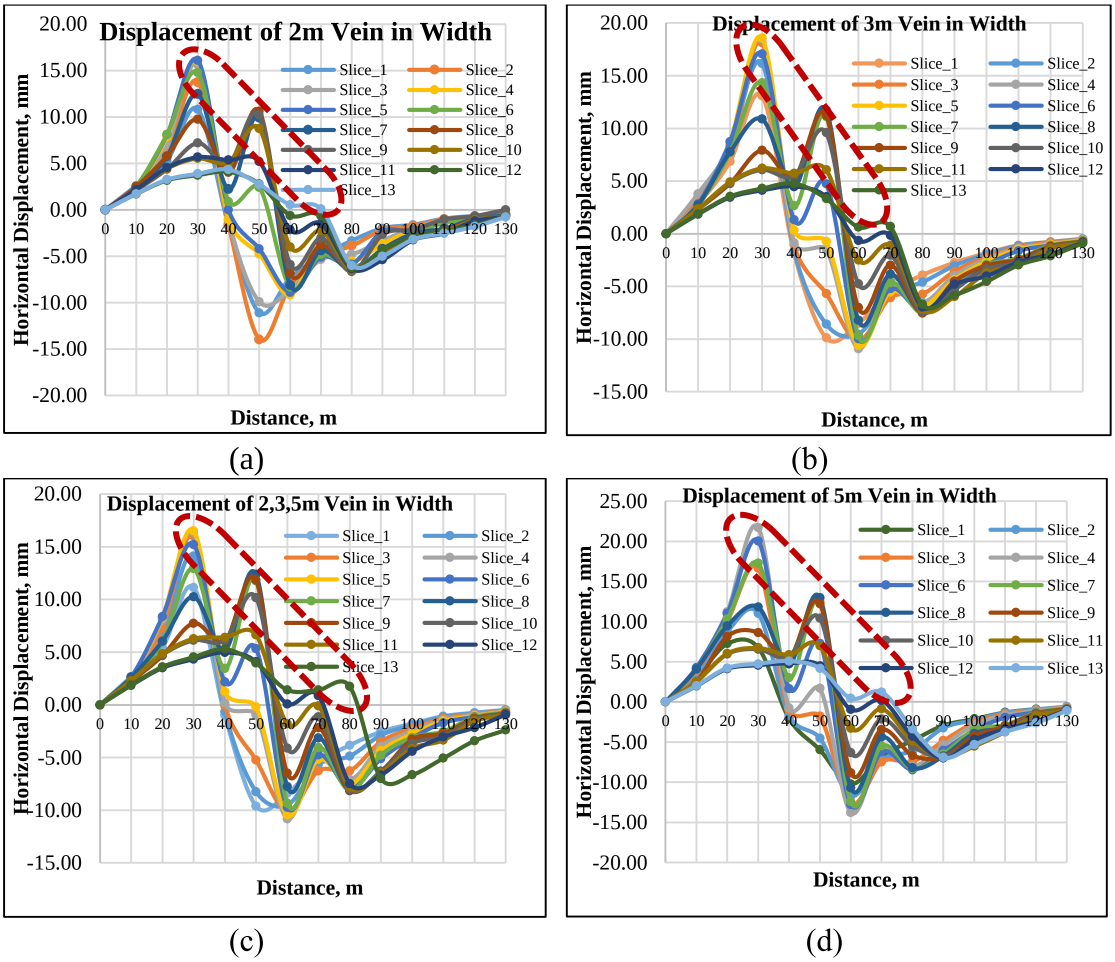
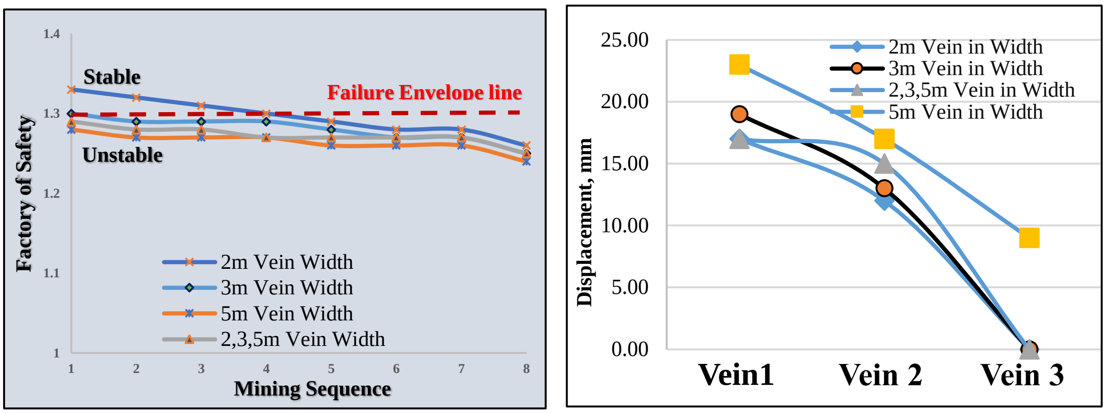


Figure 5.14. Comparison of horizontal displacement for the cut and fill method under the various vein width excavated.

Figure 5.14 shows the comparative horizontal displacement results under different widths of the vein excavated. The displacement appears to have increased specially at Vein 1 where vein 1 is first excavated for multiple excavations. As observed from Figure 5.14(a), the maximum displacements recorded are (+17 mm) and (– 14 mm) at vein 1, (+12 mm) and (– 9 mm) at vein 2, and (+7 mm) and (– 7 mm) at vein 3.

In Figure 5.14(b), the result of 3 m vein width, the displacements ranges observed are (+19 mm) and (– 12 mm) at vein 1, (+13 mm) and (– 11 mm) at vein 2, and (+8 mm) and (– 8 mm) at vein 3. Figure 5.14(c) results for 2,3,5 vein width were recorded as follows ; (+17 mm) and (– 12 mm) at vein1, (+15 mm) and (– 11 mm) at vein 2, and (+9 mm) and (– 9 mm) at vein 3. And Figure 5.14(d) shows the result of 5 m vein width with ranges (+23 mm) and (– 15 mm) at vein1, (+14 mm) and (– 14 mm) at vein 2, and (+8 mm) and (– 8 mm) at vein3 respectively. According to the result, it can be concluded that maximum displacements occurred at vein 1 with 5 m vein width excavated in the Sn-W Deposit Mine.



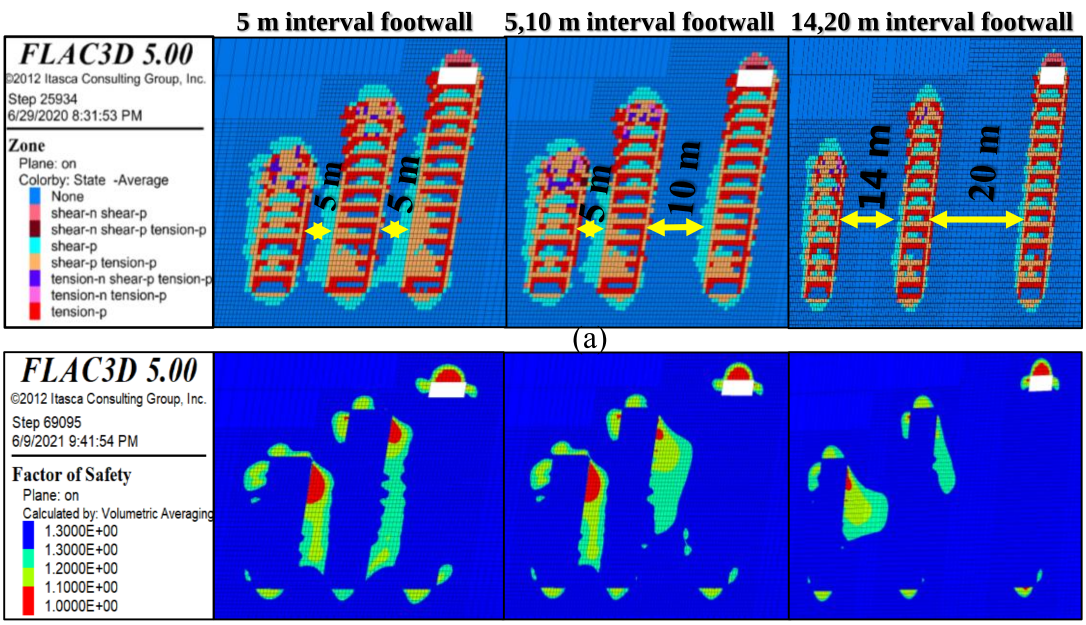
(a) (b)

Figure 5.15. Comparison of the factor of safety and the horizontal displacement under different vein widths.

Figure 5.15 shows the comparison of FoS of the cut and fill method and horizontal displacement of stope compared with vein 1, vein 2, and vein 3 under different vein widths. Figure 5.15 (a) shows the FoS result with respect to mining sequence. As expected, in the first excavation, the FoS is high and gradually decreases with advancing sequences in all simulated scenarios. Thus, when advancing the excavation sequence, it is important to carefully consider application of artificial support. On the other hand, Figure 5.15 (b) shows the horizontal displacement of stope under different vein width. The result clearly indicate that the maximum displacement is observed at vein 1, which is the initial vein excavation of stope in an underground mine. In the subsequent vein excavation much of the built up stress is redistributed resulting into relatively low displacement values.

5.4.5 Influence of different interval footwall between the excavated vein

The interval of space between multiple veins excavated in the underground is important aspect that can control the instability of stope. Therefore, as a parametric assessment, the simulation was conducted for the assessment of the instability of parallel stope excavated under different interval footwall (host rock) between the vein viz. all 5 m interval footwall, 5, 10 m interval footwall, and 14, 20 m interval footwall.



(b)

Figure 5.16. Simulation result of the instability of the cut and fill method due to the various interval of host rock between the vein excavated.

Figure 5.6 shows the simulation result of the instability of the cut and fill method due to various interval footwall (host rock) between the vein excavated. In Figure 5.16(a), the 5 m interval footwall registered high instability around the stopes at shallow depth as well as collapse for all excavated areas. However, the induced stress is greatly reduced at the 14 m, 20 m interval footwall between vein excavated when compared to the all 5 m interval footwall. Thus, to control the risk of collapsing, the rock mass in the footwall (host rock) needs to be wide in order to lower the induced stress effect. Moreover, Figure 5.16 (b) shows that the potential large shearing occurs at all 5 m intervals of host rock between the vein excavated implying high instability. After comparing with the other host rock intervals, the lowest FoS represented by the extent of shear failure is observed at 5 m intervals of host rock than the other interval combinations of host rock between veins excavated. Therefore, the interval of host rock between the vein excavated should be considerably wide enough to ensure stable stopes in the underground Sn-W Deposit Mine.

5.4.6 Instability of stope under the order of excavated vein

5.4.6.1 Effected of excavation steps

Beyond the study of mining stope under various mine excavated steps, the instability of stope is still increased at multiple veins of the research area. Therefore, the order of veins excavated has been evaluated for the understanding of the stability of the mine such as parallel excavating all veins or step-by-step excavating of the multiple veins. The procedure of stoping was carried out in a total of 8 steps per vein as shown in Figure 5.17.

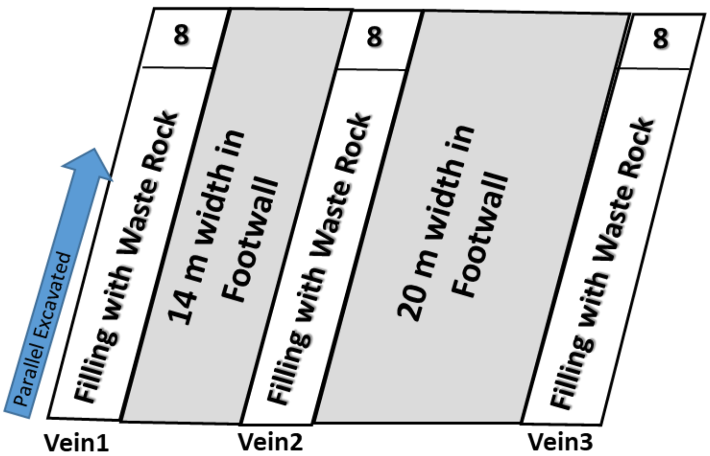


Figure 5.17. Constructed diagram for the procedures of stope excavated for each vein.

The simulation results of FoS of stope compared with parallel and step-by-step excavated in the mine are shown in Figure 5.18. Based on the result, the step-by-step excavated application appear very effective for the stability of stopes. Therefore, this order of excavation sequence technique is possible to be applied for the safety of mine.

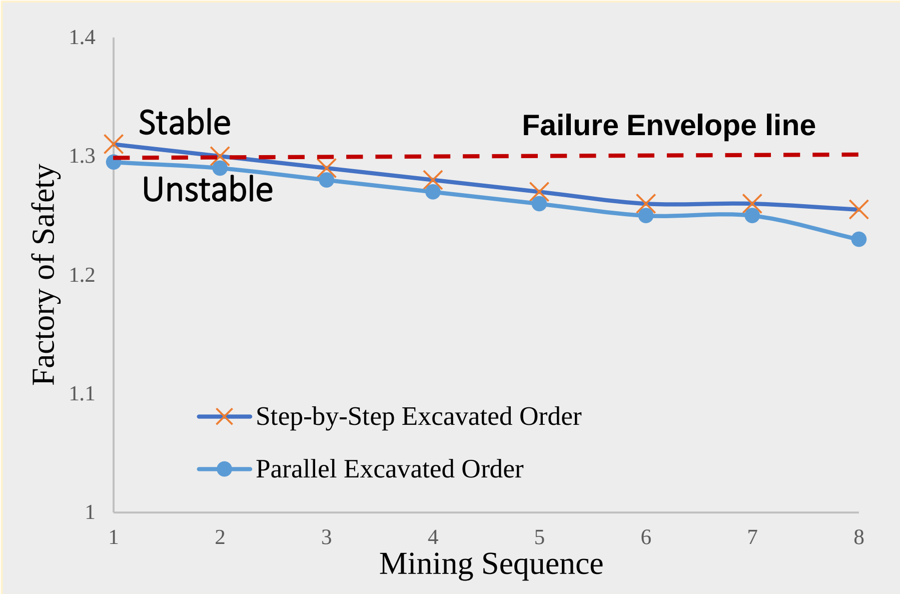


Figure 5.18. The results of the factor of safety compared with parallel and step-by-step excavated at multiple veins.

5.4.6.2 Order of ore body excavated

After estimating the previous result, continuously, the simulation of the order of ore body excavated is applied for understanding the stability of stope. Because of the environmental point of view, the research area is located in the weak rock of the hanging wall. Therefore, this impact could potentially affect the vein inclined. The induced stress is increased at the initial excavation of the vein.

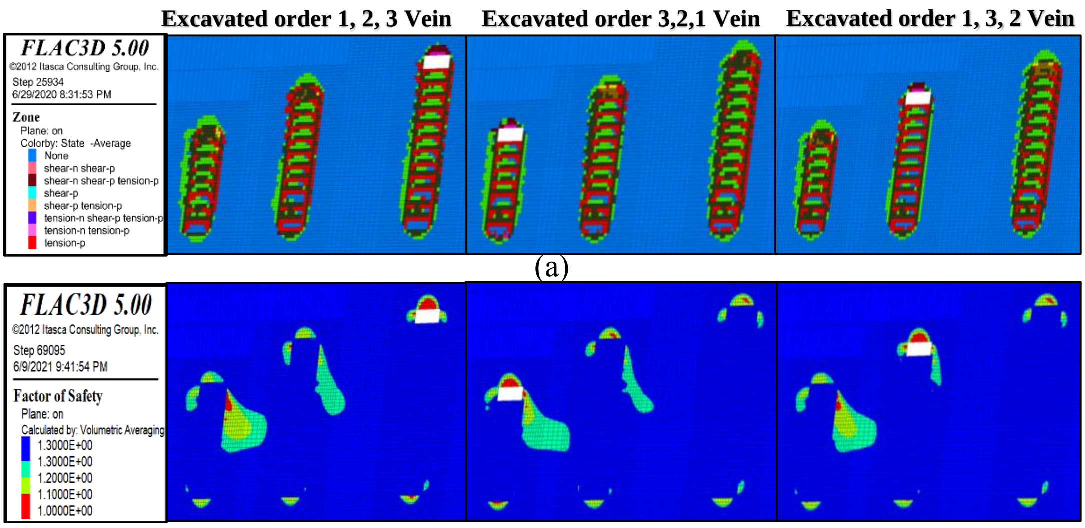


Figure 5.19. Simulation results of the instability of stope by comparing parallel excavated and step-by-step excavated for each vein at multiple veins.

Figure 5.19 (a) shows the result that shear-p is increased at vein 1 of excavated order 1,2,3 vein, where the first excavated area is the vein 1 at multiple veins. And then, the excavated order 3,2,1 vein, in which the failure is more increased at the vein, the first excavated is vein 3. The last result of excavation order 1,3,2 vein shows that the failure is increased at vein 1. On the other hand, the FoS results have been evacuated under a different order of veins excavated as shown in Figure 5.19 (b). The result indicate that the instability has occurred in the same condition as the previous explanation result of the failure mechanism. The FoS value around the stope is below the 1.3 benchmark, which demonstrates potential instability for the temporary mine. Despite the aforementioned fact on stability performance, the reverse order as 3,2,1 excavated vein is recommended due to it is safer than the other.

5.4.6.3 Evaluation of stability of mine under various depth of mined-out area

While examples of ground control problems associated with mining depth are abound, high horizontal stress at low overburdens also exit in some locations and can significantly impact local mine ground control. Factors responsible for these excessive levels of stress need to be understood and engineering controls must be developed and implemented in order to reduce the risk of collapse of ground to miners (Anthony, Dennis, & Thomas, 2005). To evaluate these factors, a prominent Sn-W Deposit Mining was created with various depths excavated underground at parallel veins. The range of depth is 100 m, 150 m, and 200 m at the depth of certain excavated vein as shown in Figure 5.20.

Figure 5.21 shows that the instability is increased at deeper mine depth than others. Because the horizontal stress tend to increase at deep excavated mine level. By comparing the various depth result, the induced stress is increased at 200 m in the depth of mined-out area and the FoS value is also low with a large area of potential shear failure compared to the 150 m in the depth of mined-out area and 100 m depth of mined-out area from the result of Figures 5.21 (a) and (b).

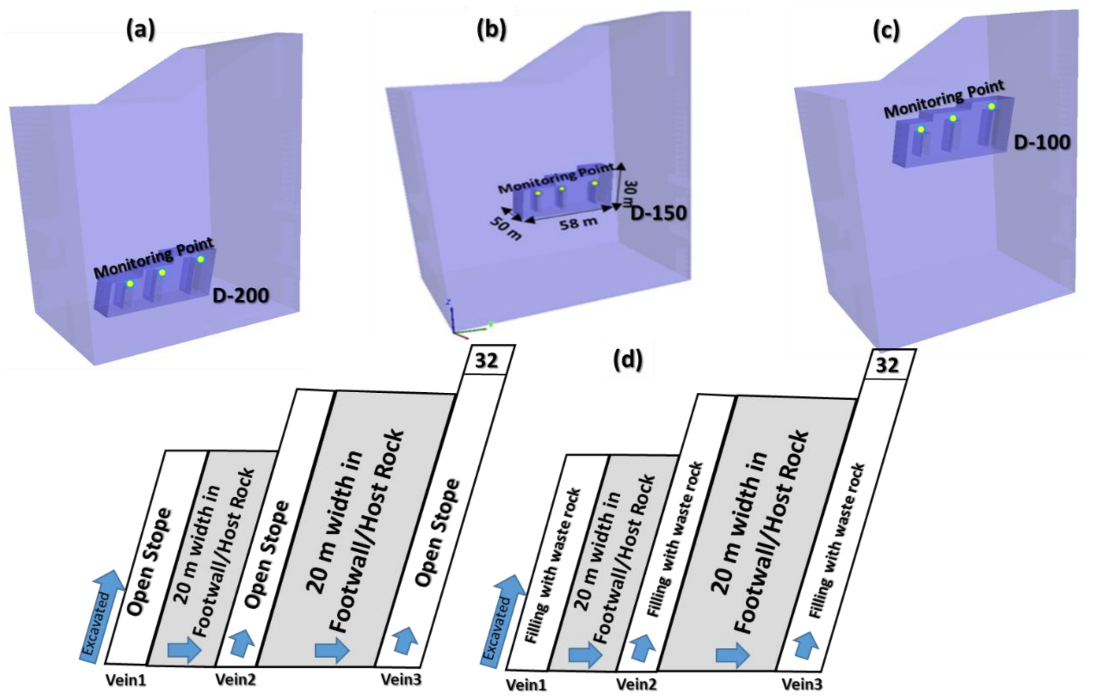


Figure 5.20. A simulation model with different depth and plan view of the mined-out area, (a) 200 m depth of located excavated mined-out area, (b) 150 m depth of located excavated mined-out area, (a) 100 m depth of located excavated mined-out area, and (d) The plan view of stope sequences in underground (Sn-W) Deposit Mine.

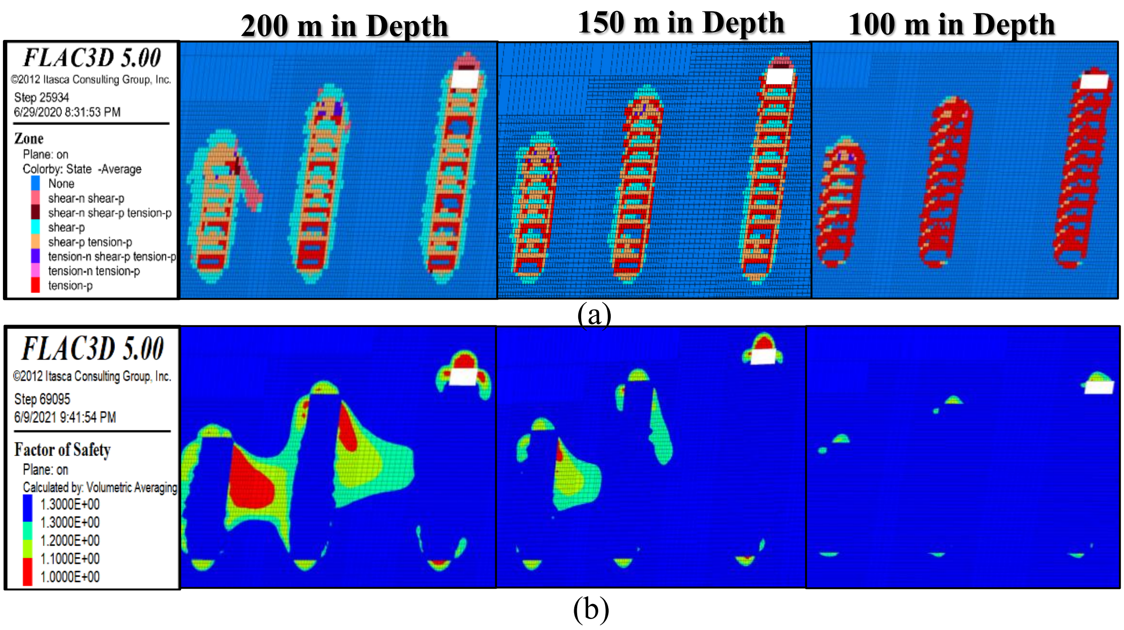


Figure 5.21. Simulation results of the instability of stope; (a) Failure zone and (b) Factor of safety under various depths of located the mined-out area.

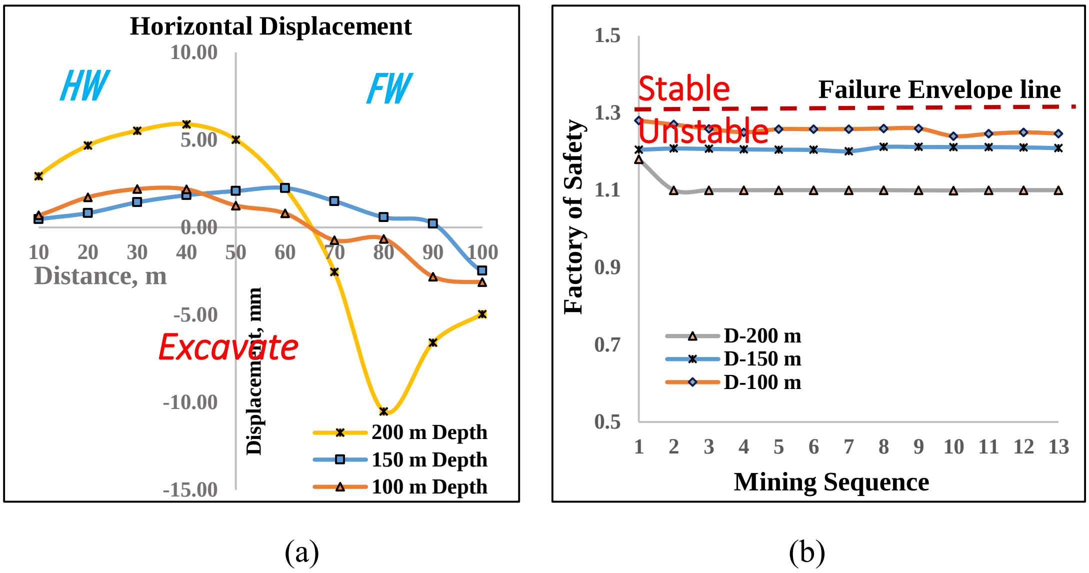


Figure 5.22. The results of instability under the various depths of located mined-out area, (a) Comparison on horizontal displacement and distance of cross-section of the mine area, and (b) Comparison on FoS and mining sequences in Hermyingyi Sn-W Deposits Mine.

Figure 5.22 depict the outcome of FoS and displacement compared at different depths of the mined-out area. According to the result, the highest value of instability of stope is observed at deep mine. In which case, the FoS value is much less than 1.3 compared to the shallow depths. The meaning of these results is that in deeper mine overburdened load and stress are high and redistributed around the stopes. Thus, if the excavation depth is deep in Sn-W deposit mine, a deliberate effort should be undertaken for stabilizing the stopes.

5.5 Assessment of Stope Instability Due to Various Angle of Inclined Vein

From the analysis in previous different parametric analysis observation, the key of the stope failure depends on not only initially excavated vein but also inclined vein. Therefore, the simulations are carried out to obtain the stability of the cut and fill method with a variety of unsystematic inclined veins for multiple excavated veins by GSI 30. The result will be compared later with the different incline vein angles with systematic/ unsystematic and interval of host rock with vein angle 40° is applied. The host rock analysis is done with vein angle 80° in the previous section. To achieve the objective, the model is created with 240 m(height) x 360 m(length), the stope is 5 m x

5 m by Phase^2 simulation for this section as shown in Figure 5.23.

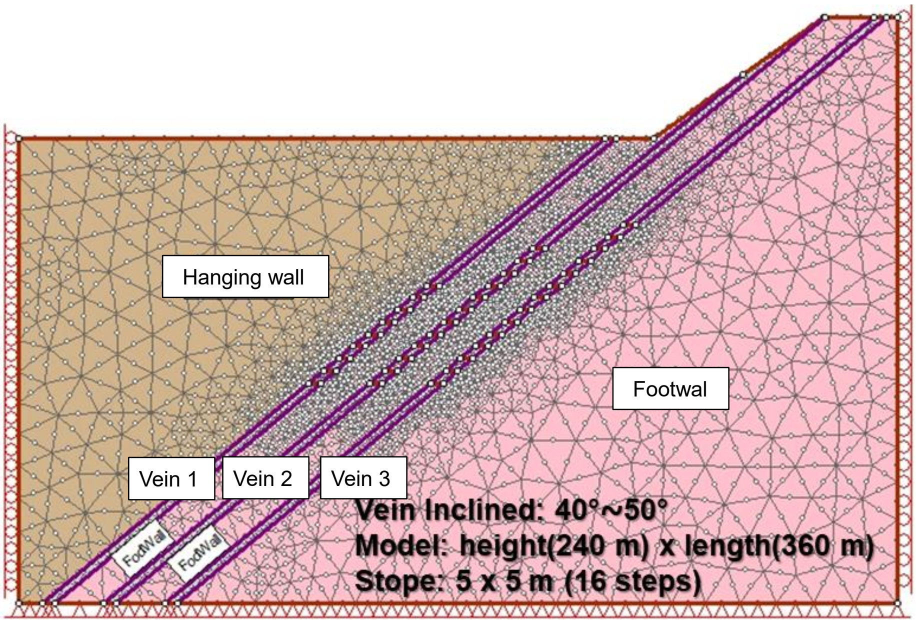


Figure 5.23. Numerical model for simulation of the cut and fill stope by Phase^2.

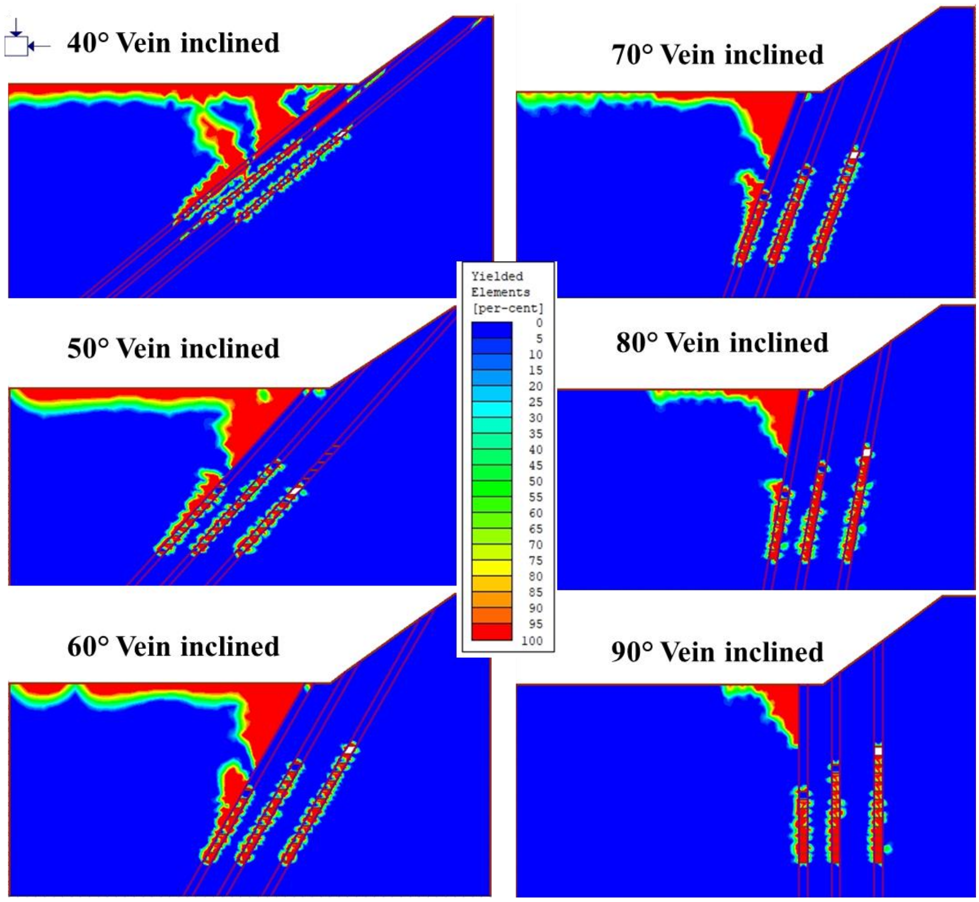


Figure 5.24. Simulation result of yielded zone of stope under various parallel inclined veins with GSI 30.

Figure 5.24 shows the simulation results of instability of stope under various parallel inclined vein angles. Based on the simulation results, the yielded zone at the hanging wall size and stope area have been observed to be high at gentle dipping angle of the veins. For the vein angle of 40°, the induced stress is more severe at the hanging wall due to the weak rock existence in the hanging wall. Nevertheless, the result of the yielded zone at the hanging wall side is reduced significantly at vein dipping angle of 90°. Therefore, it can be said that at almost vertical inclined angle of the vein a higher stability can be achieved.

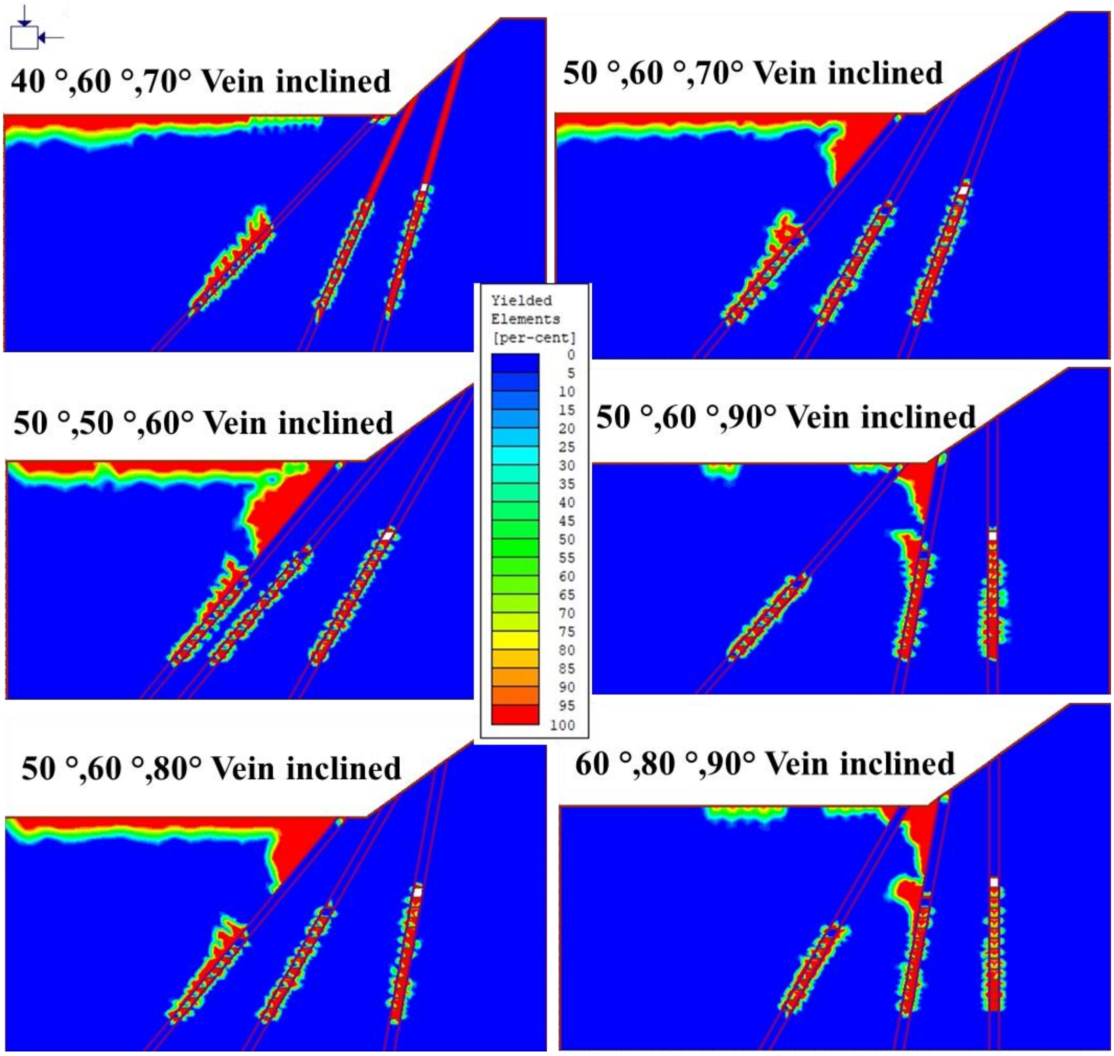


Figure 5.25. Simulation result of yielded elements at a variety of inclined vein angles at multiple veins with GSI 30.

Based on the previous parallel vein inclined results, the unparallel incline vein angel were evaluated to understand effect on the stability of stope in undergoing mine. Figure 5.25, depicts the results of the yielded zone that have occurred at an unsystematic vein angle combinations namely; (40°, 60°, 70°), (50°, 60°, 70°), (50°, 50°, 60°), (50°, 60°, 90°), (50°, 60°, 80°), and (60°, 80°, 90°) respectively. The maximum of the yielded zone is observed at (50°, 50°, 60°) inclined vein combination. This is because the first vein and second vein are inclined angle at an equal dipping angle, in which case the width of the host rock is narrow between them. Therefore, when vein angle is not parallel with the same dipping angle, the stability is increased permitting the extraction of the deposit in a more stable stope in underground Sn-W Deposit Mine.

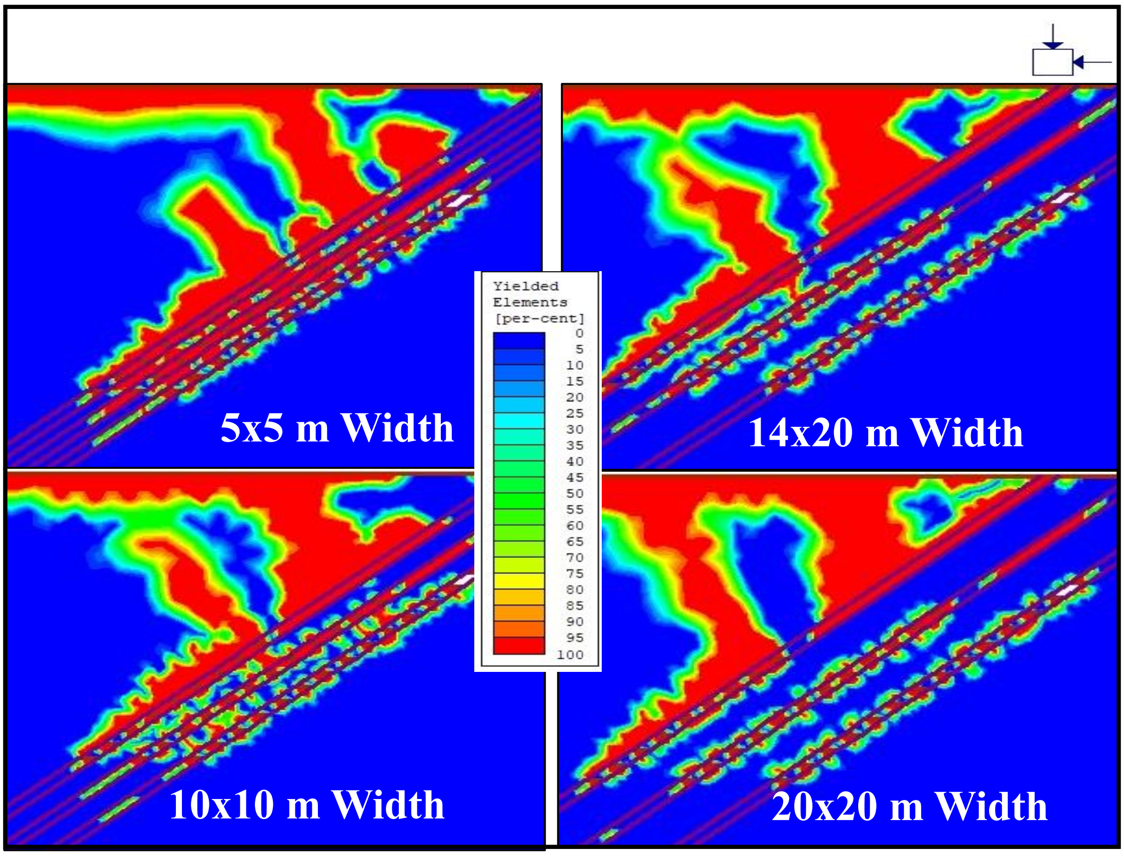


Figure 5.26. Simulation result of yielded zone of stope under various interval footwall (host rock)with 40° vein angle with GSI 30.

After scenario analysis of the parallel and unparallel inclined vein, the high yielded zone is observed both at gentle dipping vein angles of 40° and narrow interval footwall (host rock) within excavated vein. From this situation, the application of vein angels 40° with different interval footwall (host rock) between the excavated vein could help in stabilizing the stopes.

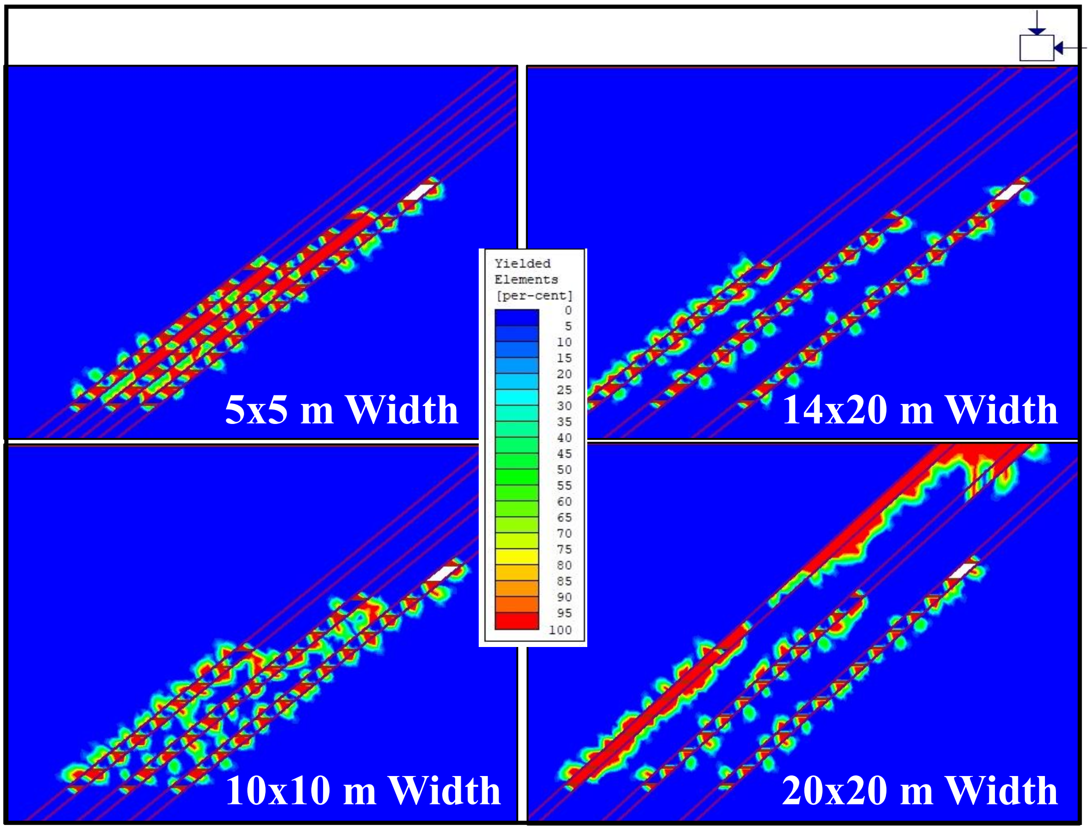


Figure 5.27. Simulation result of yielded zone of stope under various interval footwall (host rock) with 40° vein angle with GSI 46.

Figure 5.26 shows the result of the instability of stope under various widths of host rock with 40° inclined vein with GSI 30. Based on the results, the yielded zone is dramatically increased around the stope and hanging wall side at all 5 m width of host rock between the excavated vein. When applied the other width of the host rock, the induced stress becomes high in the hanging wall side and first vein and not the other veins relatively deep. Therefore, the failure can be reduced by applying the wide range of host rock in the underground. At all 5 m width of the host rock it can be recommended that the excavation processes needs to be simultaneous than step by step vein excavation and the waste rock could also be mined out as presented in Figure 5.29. The understating of stope stability is dependent on the inclined vein, as the 20 x 20 m interval footwall (host rock) result shows that the induced stress was increased at the hanging wall side but between the excavated vein, the failure is reduced than the hanging wall. Therefore, the application countermeasure should be used at the hanging wall for controlling the stability of the stopes in the first vein.

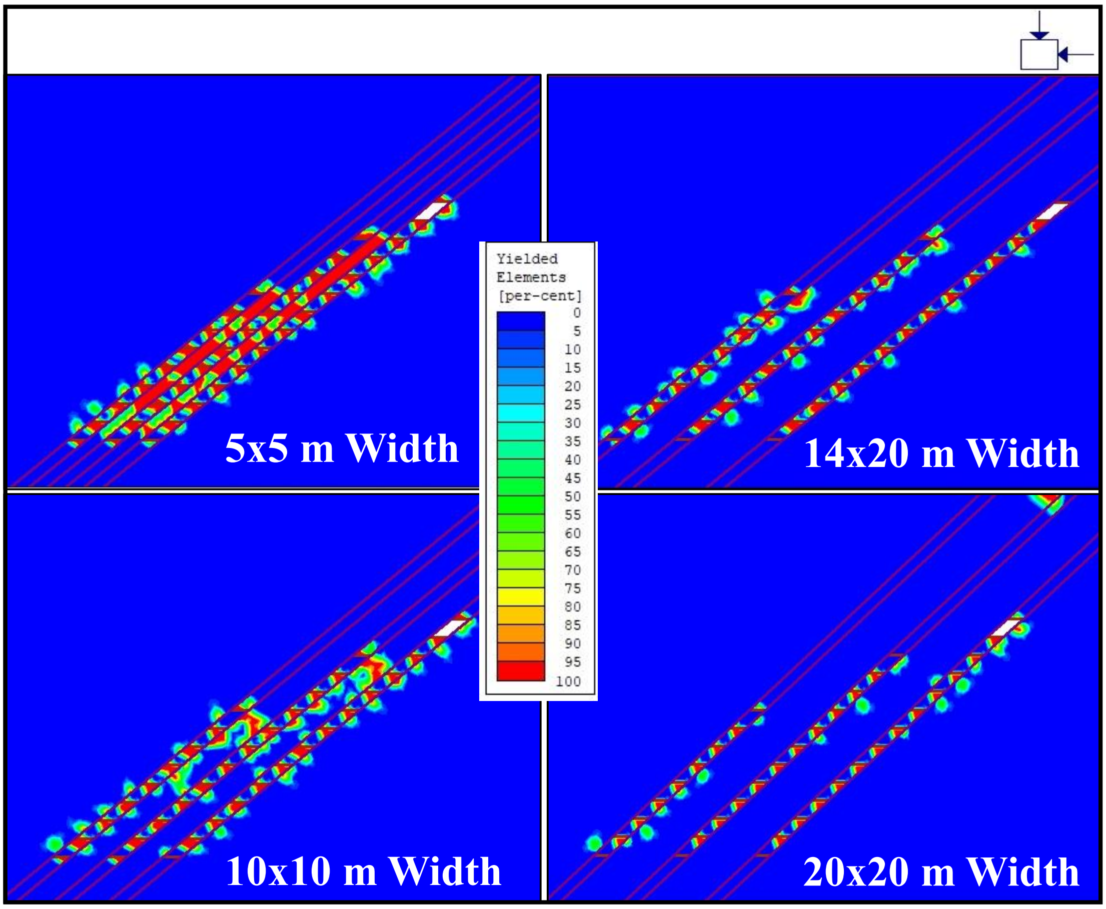
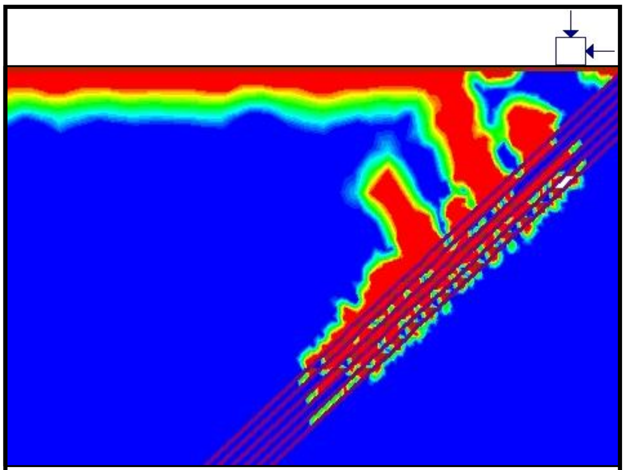


Figure 5.28. Simulation result of yielded zone of stope under various interval footwall (host rock) with 40° vein angle with GSI 55.

Figure 5.27 and Figure 5.28 show the simulation result of yielded zone of stope under various intervals of footwall (host rock) with 40° vein angle with GSI 46 and 55. As shown in the result, the induced stress is decreased at 5 m interval footwall (host rock) for all GSI values. Although, the failure is dramatically decreased at GSI 55. According to the result, the geological condition is an important parameter worthy considering

Figure 5.29 (b) shows the result of the all vein simultaneous excavation under the same conditions. From this figure, the induced stress is very height than the previous result of step-by-step excavation in Figure 5.29 (a) at multiple veins. Therefore, even though the excavation cost can reduce, the failure is very high for the stope area in the underground Sn-W Deposit Mine. Thus, this proposal is not recommended for adoption since safety is a priority in mining operations.



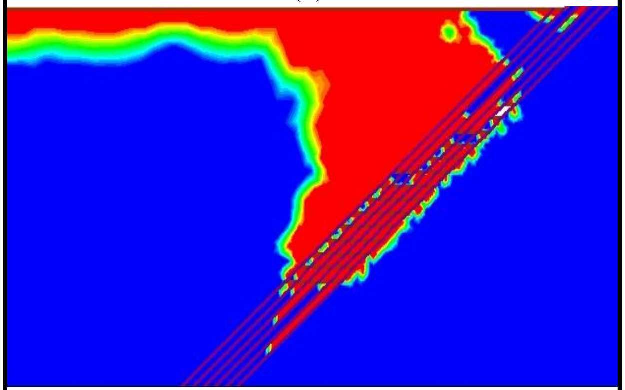


Figure 5.29. Simulation result of yielded zone of stope under the same excavated at all veins with GSI 30.

5.6 Optimization of the Instability of Multiple Veins Excavated

In this mine area, the induced stresses are observed around the stope of opening that actual mine condition. According to this problem, the countermeasure needs to be used to control the unstable stope at (Sn-W) Deposits Mine. Therefore, two kinds of countermeasure methods are proposed in this section, such as using different filling materials and supporting systems for multiple veins excavated.

5.6.1 Different types of filling material

To optimize the instability of stope in the open stoping method have the filling material can be used. This objective is to suggest the suitable filling material that can be used to reduce the instability of stope under current conditions. Figure 5.30 shows the simulation result of stope by different filling material for a model in (Sn-W) Deposit Mine with GSI 30, the failure zone is widely developed in the hanging wall with hydraulic fill that other filling materials.

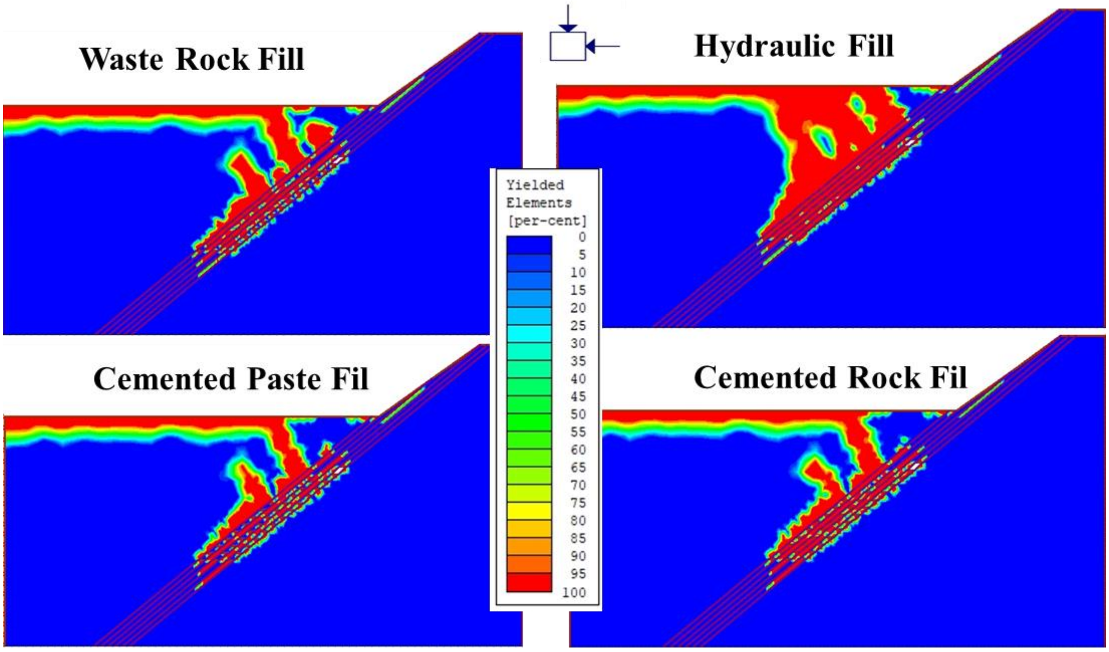


Figure 5.30. Simulation result of stope by different filling material for a model in Sn-W Deposit Mine with GSI 30 at 40° vein dip.

On the other hand, Figure 5.31 and Figure 5.32 shows the result of yielded element of parallel excavated stope under 5 m interval host rock with GSI 46 and GSI 55. The result can be said that the induced stress has been reduced surrounding the stope with GSI 55 after using CRF and CPF filling material. Based on the simulation result by comparing GSI values, the induced stress is increased at parallel excavated stope in GSI 30 even using the filling material.

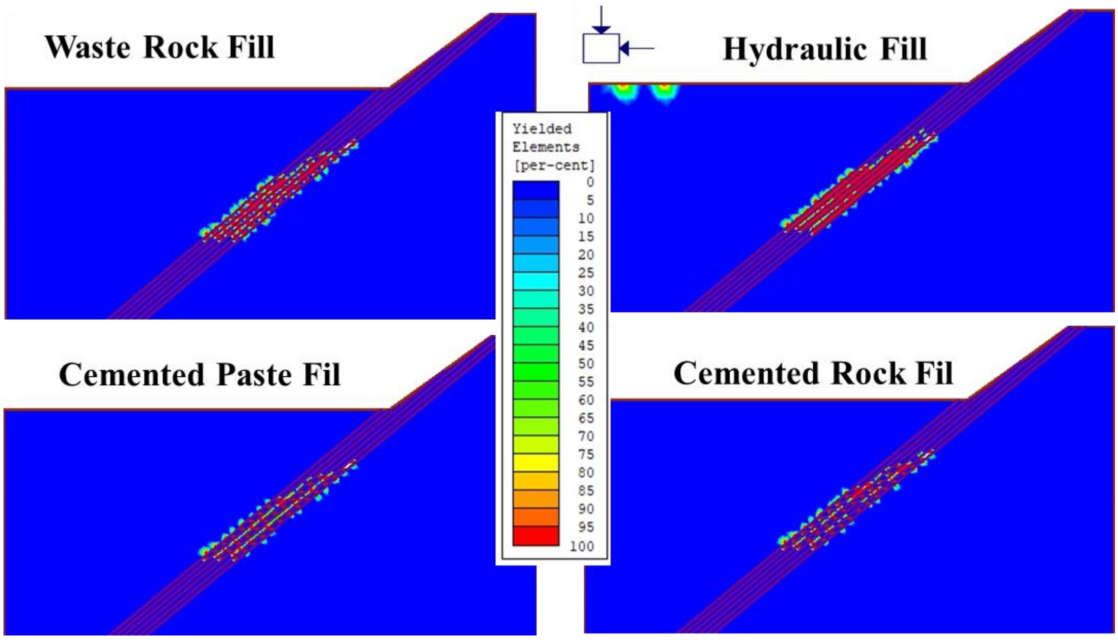


Figure 5.31. Simulation result of stope by different filling material for a model in Sn-W Deposit Mine with GSI 46 at 40° vein dip.

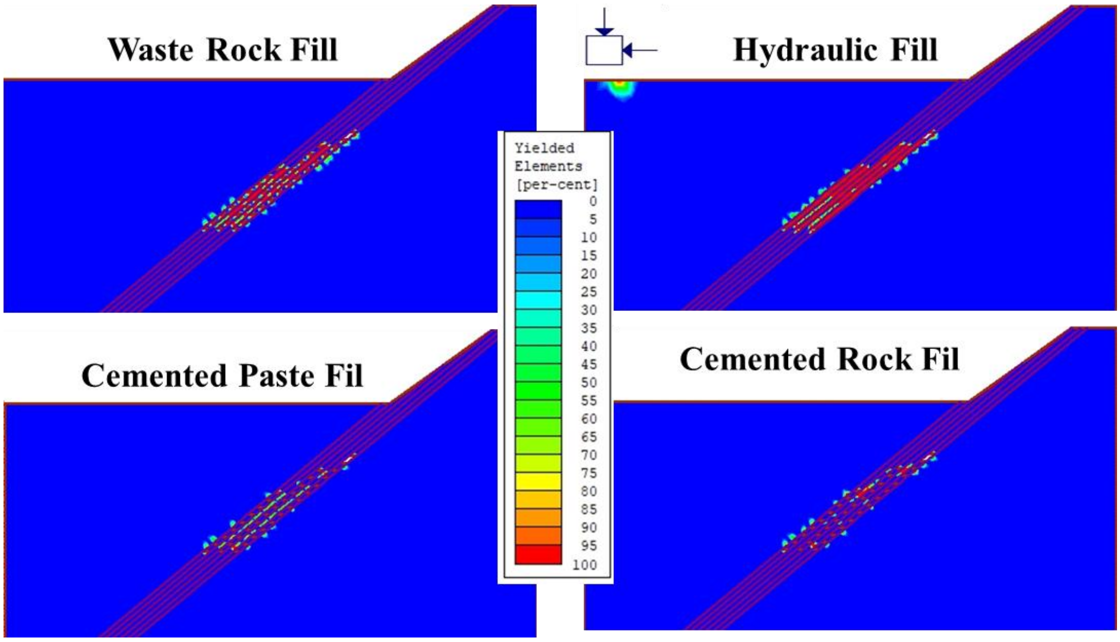


Figure 5.32. Simulation result of stope by different filling material for a model in Sn-W Deposit Mine with GSI 55 at 40° vein dip.

The filling material can reduce the failure of stopes such as waste rock, CPF, and CRF. As a result of the application of filling material, the induced stress is drastically decreased after using CPR and CRF. Because the strength of filling material properties is higher than the rock strength, therefore those materials are suitable for the optimization of stope under steeply inclined vein. Additionally, from an economic point of view, the waste rock filling material is better for the instability of stope.

5.6.2 Supporting system

To understand the effectiveness of plasticity of rock type, support as a countermeasure for the stope instability under excavated multiple vein two different types of support are installed for the stope such as active support and passive support in the numerical model (Hoek & Wood, 1987). The support types were installed in the simulation model to evaluate their efficacy. The passive type of support is shotcrete and active is cable bolt. When the rock support system are installed, the induced stress is particularly decreased around the stope. The numerical simulations are carried out with different GSI values under by using supporting system as shown in Figures 5.33, 5.34, and 5.35. Figure 5.33 shows the yield zone by application of different support system for the inclined vein excavated as multiple vein with GSI 30 at 70° dip of vein. After being supported by the shotcrete, a large yielded zone surrounding the stope still occur as shown in Figure 5.33. Displacement around 8 cm is noted at the center of roof perimeter at the stope. On the other hand, the application of cable bolts support shows that the yield zone is decreased than the shotcrete support. Despite the application of combination of active and passive rock support, there so no change in the yielded zone surrounding the stope. As a result, it can be said that the excellent support system is a combination of active and passive type rock support. However, considering economic perspective, the application of cable bolt only is sufficient to stabilize the stopes.

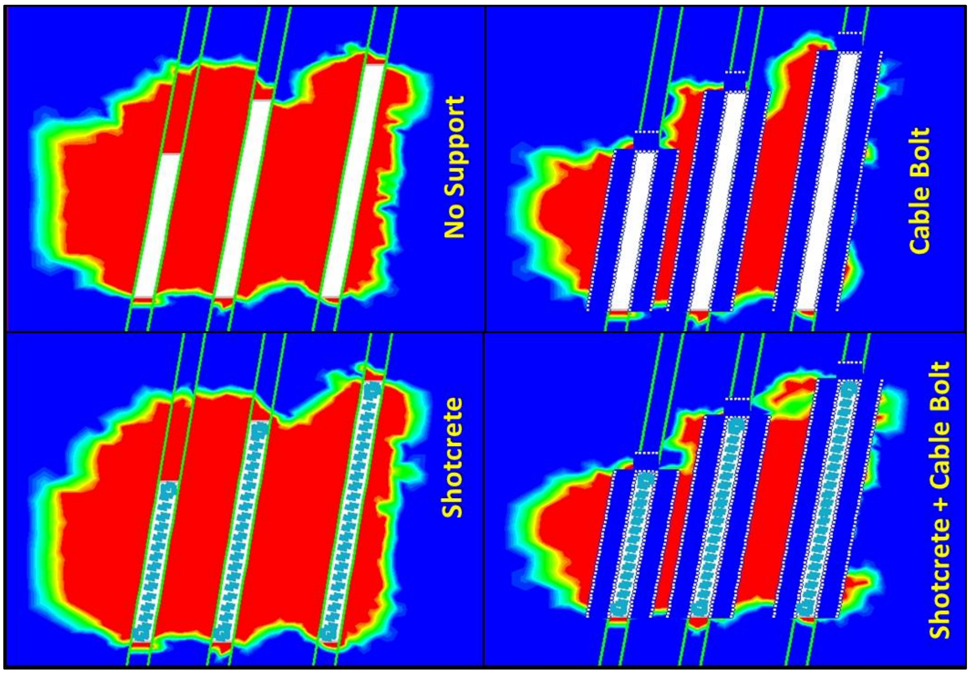


Figure 5.33. Simulation result of stope by different types of support system for a model in Sn-W Deposit Mine with GSI 30 at 70° vein dip.

After applying the supporting system at the geological rock condition of GSI 30, the other GSI values are evaluated to understand the failure condition of stope in the research area as shown in Figures 5.34 and Figure 5.35. By comparing the GSI value, the result can be said that the large induced stress has occurred at GSI 30. Although the stope failure is still observed at GSI 46 and 55 even using the supporting systems. Nevertheless, the combination of 0.1 m thickness shotcrete and 1 m x 1m cable bolt need to be applied to support the multiple vein in the open stoping method. The results indicate that the failure of stope has been maintained for the multiple stope excavated in Sn-W Deposit Mine with GSI 46 shown in Figures 5.34. Figure 5.35 shows the result of the open stoping method by using supporting under GSI 55, according to the result the induced stress is further decreased surrounding the stope by compared with GSI 30 and GSI 46. Therefore, the failure of stope at multiple vein excavations has been observed with the geological condition of this research area. Therefore, to optimize the multiple stope recovery, a combination of shotcrete and cable bolt can give high improvement to stope stability condition in the open stope area in all situations of rock mass condition but using cable bolt only is sufficient in the balancing of the geotechnical safety and economics.

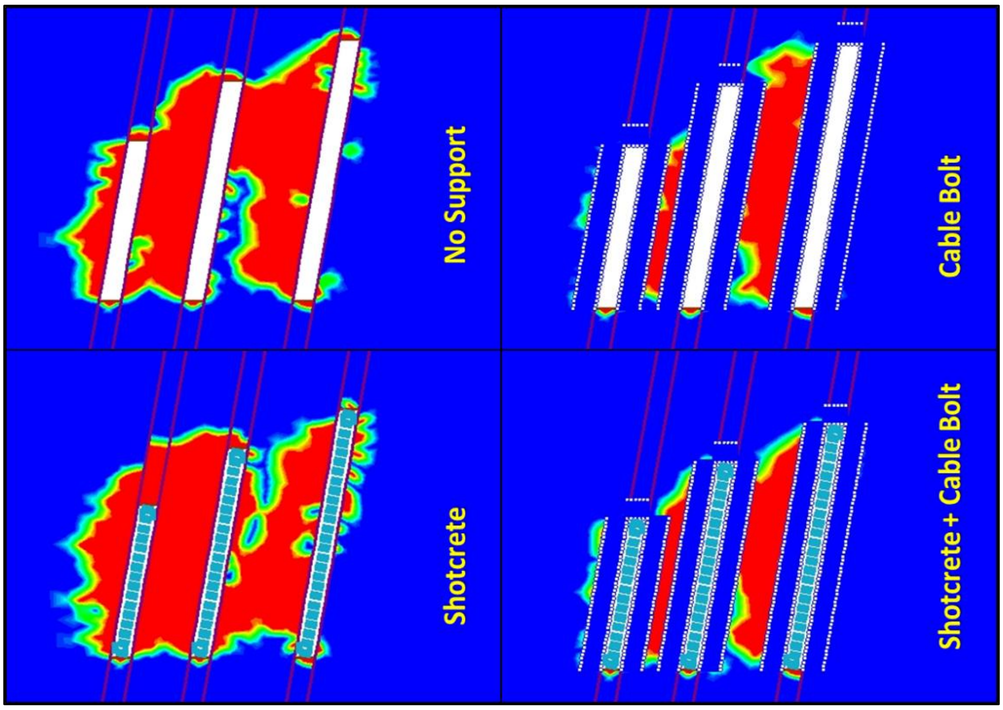


Figure 5.34. Simulation result of stope by different types of support system for a model in Sn-W Deposit Mine with GSI 46 at 70° vein dip.

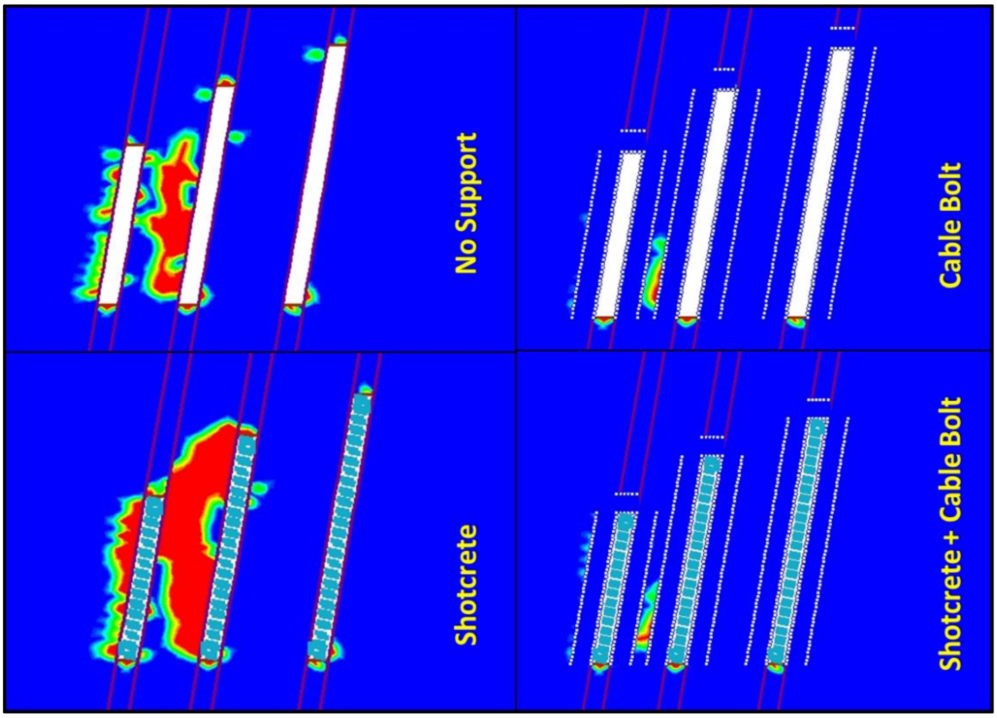


Figure 5.35. Simulation result of stope by different types of support system for a model in Sn-W Deposit Mine with GSI 55 at 70° vein dip.

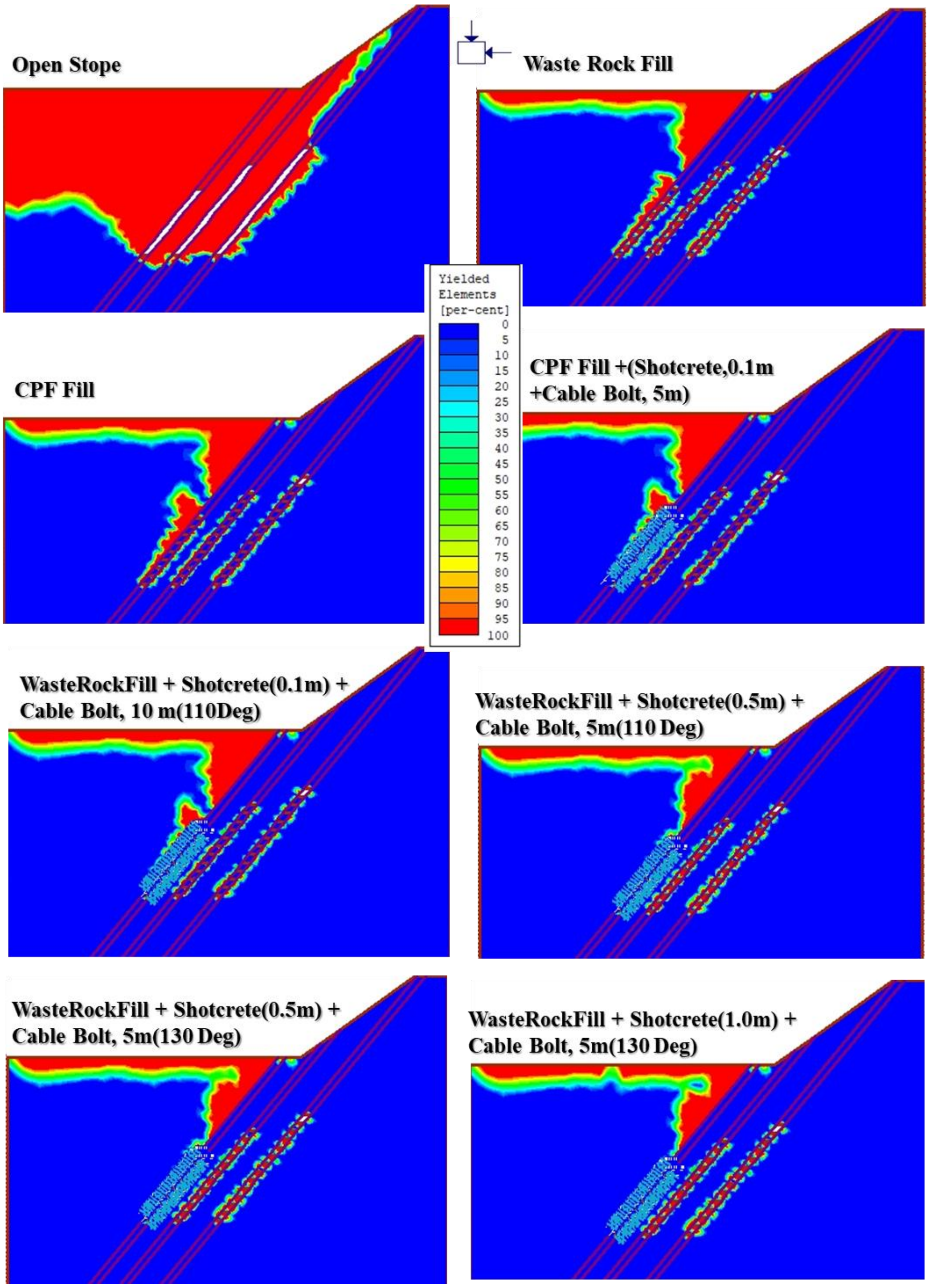


Figure 5.36. Simulation result of scenario evaluation the stability of stope under various countermeasure method with GSI 30 at dip 50° vein dip.

On the other hand, the most suitable supporting system should be used for deeply inclined vein at in Sn-W Deposit Mine; therefore, the scenario evaluation of stope stabilities is analyzed by various type of countermeasure to control the instability of multiple excavated vein. The numerical simulations are undertaken with various type of countermeasures with GSI 30 at 50° vein dip as shown in Figure 5.36. Simulation results of multiple veins of stope have been supported with the active rock type support for the model. The result strengthens the previous statement of 50° dip vein with GSI 30 that the application for the different filling material and active and passive type of support system in this condition is needed to ensure the stability of stope. In Figure 5.36, the critical point of the instability of stope has occurred at open stope. Therefore, as it is installed with filling material as waste rock and cement paste fill, the induced stress is reduced compared with without filling. After using supporting, the best countermeasure is using the combination of (1 m thickness shotcrete + 5 m length cable bolt + waste rock filling) to reduce the failure of stope at multiple veins excavated in this mine.

Nevertheless, instability still occurs when support is installed in the stope at multiple veins excavate with several supporting types, especially the highest induced stresses occur at hanging wall side of the stope. Thus, it recommended that when mine is located at an area with steeply inclined vein as GSI 30, passive type support needs to be installed. Because higher supporting capacity is needed as the weak zone and complex structure become higher. Therefore, installing passive type rock support such as shotcrete needs also to be considered when the stope is opened in a mine with the weak geological condition.

5.7 Proposed Design for Tin-Tungsten Deposit Underground Mine

Based on the whole result of this dissertation, the design is proposed for improving not only the mine safety but also the economic aspect of the Sn-W Deposit Mine. Figure 5.37 shows the proposed design based on the previous accumulation results. In the research area, the three main fundamental conditions that have been found to cause instability of stope, crown pillar, and sill pillar of mine are; geotechnical parameters, regional stress distribution, and ore deposit type from the field study and simulation result. The geological condition are related to the geotechnical properties as GSI value, and it is correlated with RQD, Q-system, and RMR results from the field study. Application of GSI value, the result carried out showed that the high induced stress occurred at GSI value under 30. If the GSI value is more than 30, the induced stress is particularly reduced for all the analyses of stope not only single vein excavation but also multiple vein exaction in Sn-W Deposit Mine. Regarding the environmental stress condition, the mine is located at a shallow underground mine. Moreover, the research area has been influenced by the geological structure, which is the reason for the main left-lateral strike-slip Sagaing fault. Therefore, the application of the different stress ratios is applied. Where the high induced stresses have been observed at the K = 0.5 for both single vein and multiple excavated, which means the vertical stress is widely distributed surrounding the shallow excavated mien-out area in the area. Another reason to apply the type of ore deposit is high-stress conditions occurred around the opening due to the ore being hard rock than the surrounding mother rock in this research area.

Based on the above condition, the supporting system and filling material is applied in the simulation model to optimize the instability of stope under the various condition of the environment and mining method. As a simulation result the improvement on the crown pillar, sill pillar, and stope were made at the hanging wall zone but the stope and crown pillar may still experience failure. Optimization of crown pillar thickness was carried out by installing the passive rock type support. Moreover, for instability of stope, countermeasure such as filling material and active and passive type support system need to be considered to optimize the instability of stope. Therefore, Figure 5.37 explains in detail the procedure for excavation of stope with respect to important parameters considered in the simulation.

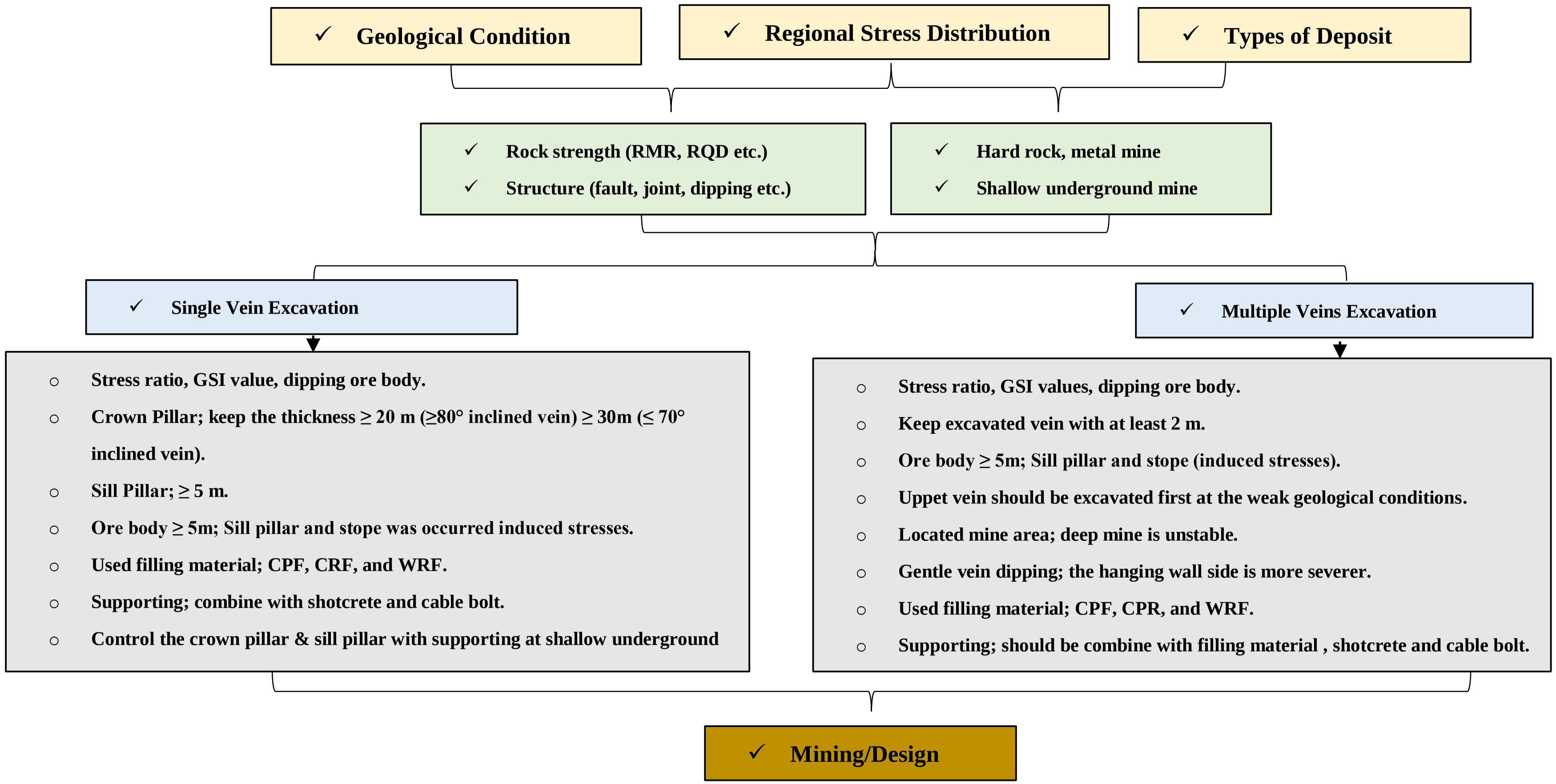


Figure 5.37. Proposed of Constructive Mine Design for Tin-Tungsten Ore Deposit.

5.8 Conclusion

In this chapter, the stability of multiple veins excavated in the Sn-W deposit mine by comparing the open stoping and the cut and fill methods was conducted. Basically, the increasing demand for tin-tungsten industry and its attractive high market price has created a motivation for mining companies to extract deep-seated tin-tungsten deposits mine in Myanmar at a fast pace to meet the world demands. Thus, $FLAC^{3D}$ver. 5.0 and $Phase^2$ver. 7.0 simulation were used to assess the stope stability of multiple vein excavation under various GSI values, different stress ratios, different width of stope, order of vein excavation for both open stoping and the cut and fill methods.

The outcome of the analyses revealed that excavation of stopes with ≥ 5 m width in all the parallel multiple vein simultaneously is not recommendable as the stopes experience high potential stope failure, hence, the author proposes the development of stopes with width dimensions of ≤ 3 m in all the parallel multiple vein simultaneously. In terms of space intervals between parallel inclined veins, it is recommendable to undertake multiple vein excavation for gentle inclined veins in weak rock condition like Hermyingyi Mine only if the interval of the veins are emplaced at ≥ 14 m and ≥ 10 m for steep dipping inclined veins.

Regarding environmental stress conditions, the high vertical stress at K = 0.5 in shallow underground mines such as the Hermyingyi Mine greatly affect the stability of excavated stopes and the impact is more pronounced in open stoping method than in cut and fill method. On another note, the excavated stopes in multiple veins in weak rock masses conditions with GSI value of ≤ 38 is extremely a risky adventure because the excavated stopes would be greatly unstable. Hence, this option can only be undertaken in good rock masses with GSI value of ≥ 46 in which the stopes in the multiple veins excavated concurrently could reliably be stable.

For artificial support mechanisms for excavated stopes in gentle inclined dip vein excavated simultaneously under weak rock mass condition, like GSI 30 for Hermyingyi Mine, combined support systems need to be adopted in which case the passive support type precedes the active support type so the that passive support type provides the initial strength to the loose broken rock blocks for effective stabilization upon the installation of the active support system.

5.9 References

Anthony, T. I., Dennis, R. D., & Thomas, P. M. (2005). *High Stress Mining Under Shallow Overburden in Underground U.S. Stone Mines.* Pittsburgh: National Institute for Occupational Safety and Health, Pittsburgh Research Laboratory.

Hoek, E., & Wood, D. F. (1987). *Support in Underground Hard Rock Mines.* Montreal: In Udd, J., Ed., Undergound Support Systems, Canadian Institute of Mining and Metallurgy.

Purwanto, Shimada, H., Takashi, S., & Wattimena, R. K. (2013). Influence of Stope Design on Stability of Hanging Wall Declined in Cibaliung Underground Gold Mine. *International Journal of Geosciences, 4*, 1-8.

Sasaoka, T., Takamoto, H., Shimada, H., Oya, J., Hamanaka, A., & Matsui, K. (2015). Surface sub-sidence due to underground mining operation under weak geological in Indonesia. *International Journal of Rock Mechanics and Geotechnical Engineering*, 337-344.

Sepehri, M., Apel, D., & Liu, W. V. (2017). Stope Stability Assessment and Effect of Horizontal to Vertical Stress Ratio on the Yielding and Relaxation Zones Around Underground Open Stopes Using Empirical and Finite Element Methods. *Archives of Mining Sciences, 62*(3), 653-669.

Zhao , H., Ma , F., Zhang , Y., & Guo, J. (2013). Monitoring and mechanisms of ground deformation and ground fissures induced by cut-and-fill mining in the Jinchuan Mine 2, China. *Environ Earth Sci*(68), 1903–1911. doi:DOI 10.1007/s12665-012-1877-7

CHAPTER SIX

6 Conclusions

Myanmar is endowed with a wide range of gemstones and ore deposits such as, tin, tungsten, copper, gold, zinc, lead, and nickel. The tin-tungsten deposit, which was studied in this research, was discovered at the Dawei District and other locations in Myanmar in the 1900s. The underground mining methods are widely used across Myanmar for all the mineralizations. Generally, mining method selected depends on the characteristics of the ore body, particularly thickness and dip, and the competency of the surrounding rock. In Myanmar, unsupported methods of mining are used to extract mineral deposits that are generally roughly tabular, plus flat, and steep dipping and are generally associated with strong ore and surrounding rock. Therefore, these methods are termed unsupported because they do not apply any artificial support systems to assist in the support of the openings, even though, a generous amount of roof bolting and localized support measures should be used often. In the research area, the instability problem occurred surrounding the excavated area of the open stoping. The causes of the collapse of the stope at shallow depth during excavation was due to the fact that the tin-tungsten metal is hard than the host rock and also due to the environmental stress conditions. To control the instability, the use of the underground mining method like cut and fill is highly recommended. However, several subsidences caused by the application of this method in some sections at the mine site have been recorded. In which case, the supporting system was introduced late to control the instability of stope in the underground mine. Thus, the purpose of this dissertation is to analyze and propose, suitable mine design and interventions aimed at effectively reducing the instability of the stopes, sill pillar, and crown pillar of the open stoping and the cut and fill methods under a single vein and/or multiple veins excavation at Sn-W Deposits Mine. This is because, the mine design is an essential issue for mine development. To achieve the objective, the research evaluated the stability conditions of Hermyingyi Mine under different conditions using FLAC^{3D}ver. 5.0 and Phase^{2}ver. 7.0 software for simulation.

Detailed explanation of the results is presented in six chapters are as follows;

Chapter 1: This chapter provided the general information of tin-tungsten deposits in Myanmar, which is perhaps one of the largest tin-tungsten producing country in the world. On the other hand, Myanmar is endowed with gemstones and ore deposits such as, copper, gold, zinc, lead, and nickel. The tin-tungsten deposit was discovered at the Dawei District and other sites in Myanmar in the 1900s. The open stoping and the cut and fill methods are applied in the underground Sn-W Deposit mine even though the open stoping is widely adopted in the research area. The chapter also introduces the price and demand of tin and tungsten in the world market, and the outlines and literature review of this dissertation.

Chapter 2: This chapter introduced the overview of the research region. The mining area is located in the southern part of Myanmar which is characterized by complex tectonic structure exacerbated by the Sagaing fault which crosses the mine site. According to the geological setting, the host rocks are Mergui (meta-sedimentary) group, alluvial, and Irrawaddy formation. The major granite and associated Sn-W Deposits is in the Southeast Asia Tin Belt. The Sn-W ores occur in granite, aplite, pegmatites, greisens, and quartz veins. Field work and laboratory tests were conducted to understand the conditions of the mine site and the geology. The laboratory tests carried out provided the geotechnical properties of the rock mass such as unconfined compressive strength (UCS), Brazilian tensile strength (BTS) etc. Based on the laboratory tests, the hanging wall side of the mine area is weaker than the footwall but the ore body is stronger than the hanging wall. The problem statement of the research area, which associated with the current mining method being used, and the objectives of the study are also presented in this chapter.

Chapter 3: This chapter gave the instability conditions of the open stoping and the cut and fill methods in underground Sn-W Deposits Mine at the single vein. Several parameters including stress ratios, geological conditions described by GSI values, various veins inclination angle, different stope width, sill pillar thickness, and excavation sequences were evaluated and compared among the open stoping and the cut and fill methods. In this case study, the results obtained were based on the mine site evaluated GSI value of 30 and 80° inclined veins for all the evaluation of various kinds of simulation. The investigations revealed that weak rock masses with GSI value of ≤ 38 are susceptible to instability and require artificial supporting mechanisms. Regarding environmental stress conditions, it was observed that high vertical stress at K = 0.5 in shallow underground mines like the Hermyingyi Mine greatly affect the stability of excavated stopes, however at a depth of ≥ 150 m the impactful stress is the horizontal stress. On another note, the inclination of the veins demonstrated that inclined veins dipping at ≤ 70° in weak rock mass condition, like at the Hermyingyi Mine, expose the excavated stopes to higher risk of instability and/or collapse particularly in the hanging wall section. Concerning the design of sill pillar thickness, the underground mines intending to develop large excavated stopes of ≥ 5 m width should keep a sill pillar thickness of ≥ 10 m but in excavated stopes of ≤ 3 m, a sill pillar thickness of 5 m is sufficient. Another interesting finding is that the mining sequencing determines the stability of the underground mine and that over-cut mining sequence approach is by far the best mining approach for underground mining. Additionally, among the economical supporting solutions that can be applied to control excessive instability, the cement paste fill (CPF) and the cement rock fill (CRF) types of filling material demonstrated excellent stabilizing performance in excavated stopes with CRF being the most excellent one. However, the waste rock fill (WRF), which has equally an excellent stabilizing performance close to the CPF and CRF, is a better stabilizing option in the economic perspective because the materials can be obtained easily from the dumping site near the mined-out area.

Chapter 4: This chapter presented the stability conditions of the crown pillar of the open stoping at the shallow underground Sn-W Deposit Mine. To achieve the purpose in this chapter, the instability of the crown pillar was evaluated under different stress ratios, GSI values, and various veins inclination angle. After the analyses, the author recommends that the design of the crown pillar in weak rock masses should be such that the crown pillar thickness is ≥ 20 m in steep dipping ore vein e.g. ≥ 80° can be left with minimal instability risk which can be secured by supporting systems. On the other hand, in the similar weak rock masses with gentle dipping vein angle ≤ 70° the crown pillar thickness of ≥ 30 m should be left to prevent the collapsing of the mine. In hard rock masses, the crown pillar thickness of 15 m to 20 m is recommended where the small thickness is suited for a near vertical dipping ore vein while the large thickness is suited for gentle dipping vein. In the case where the crown pillar hosts more valuable ore which may be required to be extracted, support systems evaluated as countermeasures to crown pillar instability and/or collapse demonstrated that in very weak geological conditions where artificial active support is not effective, the use of economical supporting methods like WRF, CPF or CRF is effective allowing upward sequence stopping until 15 m thickness of crown pillar. Meanwhile, in blocky rock masses where active support is feasible, a combined support system of cable bolt and shotcrete is recommended until 10 m thickness of the crown pillar. While hard rock masses affected by discontinuities or fracturing, the cable bolt only support system is proposed for implementation to stabilize the crown pillar. Thus, the decision in the implementation of the support systems need to properly consider the economic aspect, the value of the ore to be extracted, the rock mass condition and most importantly the efficacy of the method in restoring stability and maintaining of the crown pillar.

Chapter 5: This chapter discussed the stability of multiple veins excavated in the Sn-W Deposit Mine by comparing the open stoping and the cut and fill methods. The increasing demand for tin-tungsten industry and its attractive high market price has created a motivation for mining companies to extract deep-seated tin-tungsten deposits mine in Myanmar at a fast pace to meet the world demands. Thus, FLAC^{3D}ver. 5.0 simulation and Phase^{2}ver. 7.0 simulation were used to assess the stope stability of multiple vein excavation under various GSI values, different stress ratios, different width of stope, order of vein excavation for both open stoping and the cut and fill methods.

The outcome of the analyses revealed that excavation of stopes with ≥ 5 m width in all the parallel multiple vein simultaneously is not recommendable as the stopes experience high potential stope failure, hence, the author proposes the development of stopes with width dimensions of ≤ 3 m in all the parallel multiple vein simultaneously. In terms of space intervals between parallel inclined veins, it is recommendable to undertake multiple vein excavation for gentle inclined veins in weak rock condition like Hermyingyi Mine only if the interval of the veins are emplaced at ≥ 14 m and ≥ 10 m for steep dipping inclined veins. Regarding environmental stress conditions, the high vertical stress at K = 0.5 in shallow underground mines such as the Hermyingyi Mine greatly affect the stability of excavated stopes and the impact is more pronounced in open stoping method than in cut and fill method. On another note, the excavated stopes in multiple veins in weak rock masses conditions with GSI value of ≤ 38 is extremely a risky adventure because the excavated stopes would be greatly unstable. Hence, this option can only be undertaken in good rock masses with GSI value of ≥ 46 in which the stopes in the multiple veins excavated concurrently could reliably be stable. For artificial support mechanisms for excavated stopes in gentle inclined dip vein excavated simultaneously under weak rock mass condition, like GSI 30 for Hermyingyi Mine, combined support systems need to be adopted in which case the passive support type precedes the active support type so the that passive support type provides the initial strength to the loose broken rock blocks for effective stabilization upon the installation of the active support system.