STUDY ON ASSESSMENTS AND COUNTER MEASURES FOR THE STABILITY OF STOPE DUE TO THE PREVIOUS MINED-OUT ACTIVITIES IN CUT-AND-FILL UNDERGROUND GOLD MINE IN MYANMAR

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STUDY ON ASSESSMENTS AND COUNTER MEASURES FOR THE STABILITY OF STOPE DUE TO THE PREVIOUS MINED-OUT ACTIVITIES IN CUT-AND-FILL UNDERGROUND GOLD MINE IN MYANMAR

A DOCTORAL DISSERTATION

Submitted to the Department of Earth Resources Engineering
Graduate School of Engineering
Kyushu University

As a partial fulfillment of the requirements for the degree of
Doctor of Engineering

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ABSTRACT

In Myanmar, as the development of high-grade ore existing in the shallow area becomes to be important in recent years, the cut-and-fill method is suitable among the various underground mining methods for maintaining the stability of working stope and minimizing the impact of mining activities on the surface. Since the stope is filled with backfilling material such as waste rock, cut-and-fill method is a method that can control the stability of rock mass around the mined-out area and prevent surface subsidence. Hence, the environmental impacts due to the mining activities becomes to be small. Besides, the sill pillar is the ore that is left below the mined-out stope to prevent the collapse of working stope. The ore between surface and stope is also left as the crown pillar to maintain the stability of working stope and prevent the occurrence of subsidence. Modi Taung gold mine which is targeted in this research is one of the largest underground gold mines in Myanmar and applies as an overhand cut-and-fill method. Since the rock mass condition in shallow area is poor and mechanical properties of rocks is weak, not only much supports have to be installed in the working stope but also a plenty of ore has to be left as sill pillar and/or crown pillar in order to maintain the stability of working stope and control surface subsidence. Moreover, the conditions of previous mined-out area also have an obvious impact on the stability of working stope and surrounding rock mass. From these backgrounds, the purpose of this research is to develop appropriate design guidelines and effective stabilization measures for sill pillar and crown pillar considering with the influence of previous mining activity. An attempt has been made to investigate the optimum design for sill and crown pillars and the effectiveness of stabilization measures by means of FLAC3D. 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 in Myanmar, the background of this research, the overview of cut-and-fill mining method, the factors influenced on the stability of stope and subsidence and then the overview of problem statements in this research area. The objectives and the outline of the dissertation is also described in this chapter.

Chapter 2: This chapter describes the mining conditions of Modi Taung gold mine. The exploration works for this mine area have been conducted since 1996. The gold deposit is hosted in the sedimentary units of the Mergui Group, which is mainly composed of

mudstone, sandstone, limestone and igneous intrusions. The cut-and-fill mining method is applied in this mine. Based on the results of laboratory experiments, the intact rocks in this underground mine are strong. However, according to the field observation and bore hole core logging, the rock mass in this mine has many discontinuities. From the results of laboratory tests and field investigations, it can be found that the rock mass condition in this mine site is very poor condition within 30 m depth from the surface, and poor to fair condition deeper than 80 m depth from the surface. Additionally, as heavy rain is a common in this mining region, the conditions in the underground openings at Modi Taung gold mine are very humid with meteoric water seeping through the geological structures. As a result, weathering of the rock mass and backfilling material in the previous mined-out area was found and these conditions should be paid attention to the safety in order to prevent accidents due to the instability of rock mass around and inside of the stope.

Chapter 3: This chapter discusses the effect of previous mined-out area on working stope. The mining operation in Modi Taung gold mine has been developed in shallow regions so far due to their easy access. Hence, the mining activities are going to extend the deeper levels below the previous mined-out area. Accordingly, a new stope developed below the previous mined-out area is influenced by not only its own induced stress but also the stress redistributions from the previous mined-out area. In order to evaluate the stability of working stope below the previous mined-out area, a series of numerical investigations are carried out in different geological and mining conditions in order to fully understand the stability of the stope and sill pillar due to the influence of previous mined-out activities. From the simulation results, it can be found that the stability of rock mass around the stope obviously decreases with progression of the stope operation towards the upper slices, and then the failure zone propagates to the upper previous mined-out area. Therefore, the sill pillar should be left at least 3 m in thickness in order to stabilize the working stope for safe operation. The stability around the stope also decreases when the distance between the working stope and previous mined-out area is larger than 5 m, the monitoring of the rock mass around the stope should be conducted. As the feasible instability of rock mass is likely to occur more in lower vein dip, more severe geological conditions, wider stope width and higher horizontal-vertical stress ratio, the wider the sill pillars are required. Moreover, when the mining activities are carried out below the previous mined-out area, the effect of the conditions of backfilling material in previous mined-out area such as the deterioration of backfilling material and surrounding rock mass, filling rate with backfilling material, have to be taken into account. For the considerations of the deterioration of backfilling material in the previous mined-out area, in the case that the mechanical properties of backfilling material decrease to 25 %, the thickness of sill pillar should be more than 3.5 m as the area of unstable rock mass around the stope and pillar increases. In addition, in the case of no backfill condition during mining activities in previous mined-out area, the sill pillar should be left more than 4 m in thickness in order to ensure the stability of the sill pillar and working stope under the previous mined-out area. From the above results, it can be concluded that not only geological and mining conditions but also the condition of mind-out area have an obvious impact on the stability of the working stope. Therefore, the condition of mined-out area adjacent of the working stope has to be investigated before designing the sill pillar and the support of the working stope.

Chapter 4: This chapter discusses the effect of the mining activity on the stability of the slope surface in different geological and mining conditions because most of the primary deposit of metal mines in Myanmar are located in mountainous regions and the stopes have been developed close to the slope surface. From the results of a series of numerical simulations, it can be found that rock mass under the slope surface is affected by unequal differential stress due to the weight of overlaying rocks, the instability of rock mass around stope arises and the failure zone can develop around the mining activities increase with decreasing the distance between stope and slope surface, especially in case that the distance is less than 25 m. Subsequently, mining activities under slope topography are affected more by the variation of stress and failure zones than other places of rock mass due to the influence of the slope condition. Moreover, as the failure zones around the stope are propagating to the slope surface when the distance between the stope and surface is less than 15 m. the subsidence of the slope surface may occur, and subsequently it may induce a slope slide. Therefore, the crown pillar should be left more than 20 m in thickness. Additionally, the monitoring should be conducted when the distance between the stope and stope surface is less than 25 m in order to detect ground movement and prevent subsidence of slope and slope slide.

Chapter 5: This chapter discusses the countermeasures for maintaining the stability of the stope and surrounding rock mass affected by the previous mined-out activities and the slope surface. Two types of countermeasures, the installation of a cable bolt and shotcrete, are selected and used to improve the stability of stope and pillars. From an economical point of view, the installation of a cable bolt is preferred to that of shotcrete due to its lower cost and faster installation, however the countermeasure with higher supporting capacity should be considered where the potential of rock failure is large. From the results of a series of numerical analyses, in the case where a new stope is developed below the previous mined-out area, it can be made clear that even though the stability of the rock mass around stope can be improved by the installation of cable bolts, the stability of sill pillar is not improved obviously and more than 3 m thickness of the sill pillar still needs to be left. On the other hand, the installation of shotcrete can improve the stability of the rock mass around the stope effectively and the thickness of sill pillar can be decreased from 3.0 m to 2.5 m. Moreover, it is also effective in wider stope and higher stress conditions. When the stope is developed near the slope surface, it can be said that the installation of cable bolts has no obvious impact on the stabilities of stope and slope surface. On the other hand, the installation of shotcrete can effectively improve the stability of the crown pillar, the thickness of crown pillar can be reduced from 20 m to 10 m. Therefore, it can be concluded that an optimum mining operation can be done according to the grade of ore by applying shotcrete.

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

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Naung Naung
Fukuoka, Japan
March 2020

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CHAPTER 1

INTRODUCTION

1.1. Mining industry of Myanmar

The mining industry is considered as one of the fastest growing economies all over the world but in particular, the countries who export mineral products can get more benefits for their economic development. Myanmar is a geologically-diverse country containing a wide array of mineral resources, such as gold, silver, copper, tin, tungsten, zinc, jade and gemstones as shown in Figure 1.1 (Zaw 2017) (Soe Win and Malar Myo Myint 1998) (Gardiner, Robb, and Searle 2014) (Connette et al. 2016). The country has one of the most diverse and richly endowed collections of natural resources in Southeast Asia and still remain unexplored of its various mineral commodities.

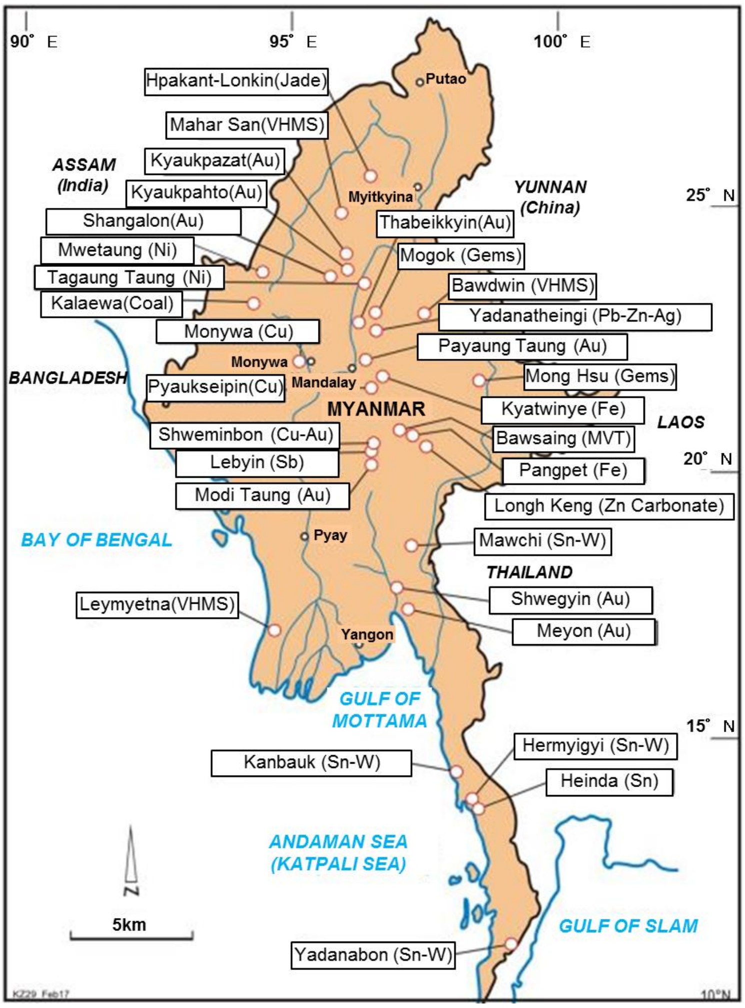

Figure 1.1 Major ore deposits in Myanmar (modified from Khin Zaw, 2017).

The extraction of mineral resources has been one of the major sources of income for national economy for many years, and mineral exports are likely to improve under the new Myanmar mines law and rules. So far, the contribution of export incomes from mineral products to the country's economy is steadily increasing year by year as shown in Figure 1.2 (Ministry of Commerce 2019). After promulgating the amendments of Myanmar mines law in December 2015 and Myanmar mines rules in February 2018, Myanmar's mining industry is expected to expand in the near future on higher foreign and domestic investments. Currently, the Myanmar government is encouraging proposals for new mining developments following a series of new reforms aimed at incentivizing investments. Because of the great potential of investments in various mineral commodities, mining sectors both surface and underground mining have to be developed in the near future of Myanmar.

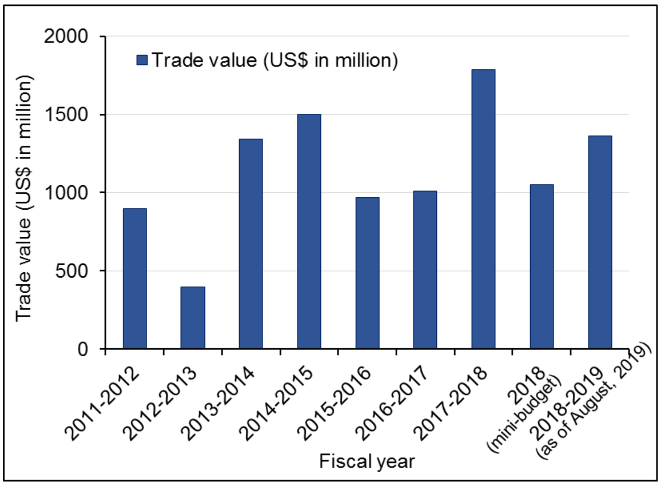

Figure 1.2 Export incomes from mineral commodities of Myanmar (source: Ministry of Commerce, Myanmar).

1.2. Literature review

Mineral consumption is gradually increasing as the global standard of living increases and mineral demand will be largely concentrated in developing countries experiencing economic development progressively. This implies mineral extraction from greater depths both surface mining and underground mining. However, underground mining will become more important in the future as environmental and social concerns make surface mining less attractive (Karian 2016). Once the method is usually employed when the depth of the deposit and stripping ratio (waste to ore ratio) are too large to start a surface mining operation. It has been observed that there are many kinds of underground mining methods, and ore extraction by an underground mining method involves many considerable range of functions. The design selection of a mining method requires a systematic approach, with the dip, size, and shape of an ore body, strength of the ore and host rock mass, as well as economics being some of the fundamental parameters influencing the planning and design process (Brady and Brown 2004) (Villaescusa 2014). Reflecting the importance of ground support, underground mining methods are categorized in three classes on the basis of the extend of support required: supported method, unsupported method and caving method (Hartman 1987).

Caving method is typically applied to large, fairly flat-dipping ore bodies with rock mass characteristics that are amenable to sustainable massive caving. Three major caving methods are recognized i.e. longwall mining, sublevel caving, and block caving (Hartman 1987). Longwall mining is used in horizontal, tabular deposits and mainly employed to coal mining applications, while the other two methods have applied in inclined or vertical, and massive deposits. An understanding of cavability, fragmentation, and stress are primary elements in designing and operating a caving mine (Brannon, Carlson, and Casten 1992). Figure 1.3 shows the layouts of typical cave mining methods (Copco 2007) (Department of Environment Australia 2014).

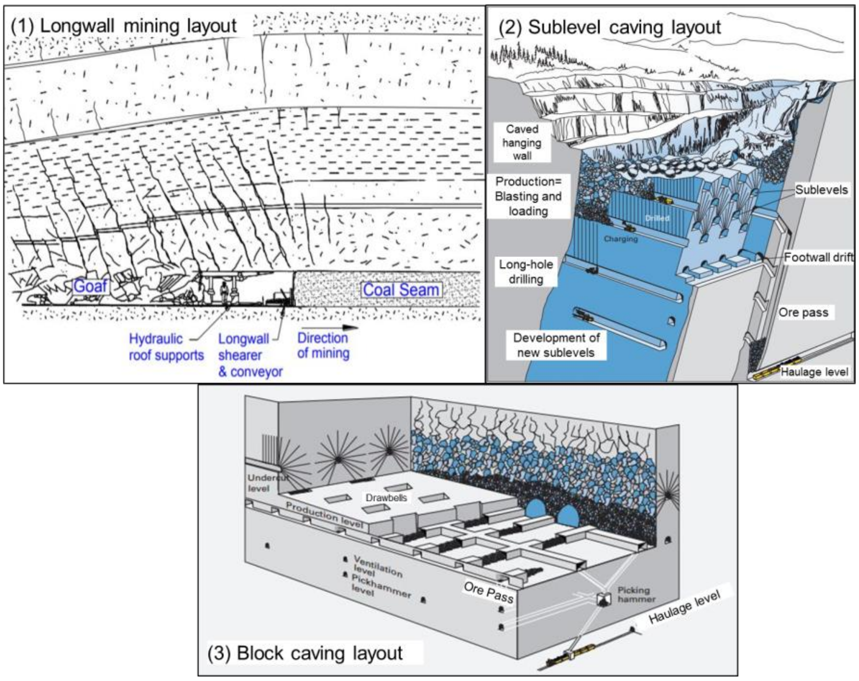

Figure 1.3 Typical cave mining methods (1) longwall mining (2) sublevel caving (3) block caving, source: (Atlas Copco) (Dept: of Environment, Australia 2014).

Unsupported mining method, in which the host rock is essentially self-supporting and no artificial support is necessary to carry the load of overlaying rock. This method can be applied in room and pillar method, stope and pillar method, shrinkage stoping, and sublevel stoping (Okubo and Yamatomi 2009). The ore deposit type with flat-dipping and tabular shape can be employed with room and pillar whereas shrinkage and sublevel stoping are applied to steeply inclined ore bodies. However, the major concerns when applying these methods are the long term stability of the opening after mine closure. Abandoned mine working possess subsidence threat in the future as reported by several researchers (Statham and Treharne 1991) (Longoni et al. 2016). Moreover, those methods offer 60 % to 80 % recovery due to the need to left pillar (Hartman 1987) (Brannon, Carlson, and Casten 1992). Therefore, some considerations upon ore recovery and safety issues are needed when unsupported mining method is employed to underground mining. Figure 1.4 describes the layouts of unsupported mining methods (Copco 2007).

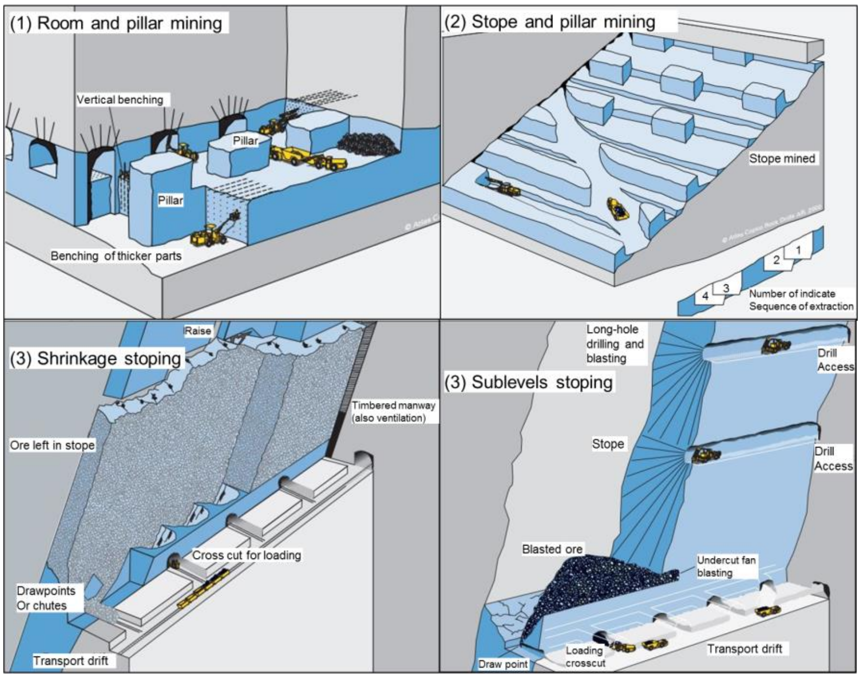

Figure 1.4 Unsupported mining methods (1) room and pillar mining (2) stope and pillar mining (3) shrinkage stoping (4) sublevel stoping (source: Atlas Copco).

Supported mining methods require some types of backfill to provide substantial amounts of artificial supports to maintain stability in the exploitation openings of mine, as well as systematic ground control throughout the mine (Hartman 1987). Supported methods are used when mine openings are not sufficient stable to remain excavations during mine operation. In other words, the supported mining method is intended for application under the surrounding ground conditions ranging in competency from moderate to incompetent. No field or laboratory tests have been devised to determine competency in large rock masses. In order to determine the rock mass is competence or not, the best empirical approaches is the rock quality designation (RQD) based on drill core evaluation. There are three specific methods in the supported mining method which are cut-and-fill stoping, stull stoping and square-set stoping (Hartman 1987). Cut-and-fill and stull stoping methods are intended for moderately competent rock, while square set stoping is suitable for the least competent rock. Cut-and-fill stoping is the only method of supported class in common use whereas 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 (Okubo and Yamatomi 2009). Figure 1.5 describes a schematic diagram of supported mining method (Karian 2016) (Harraz 2016).

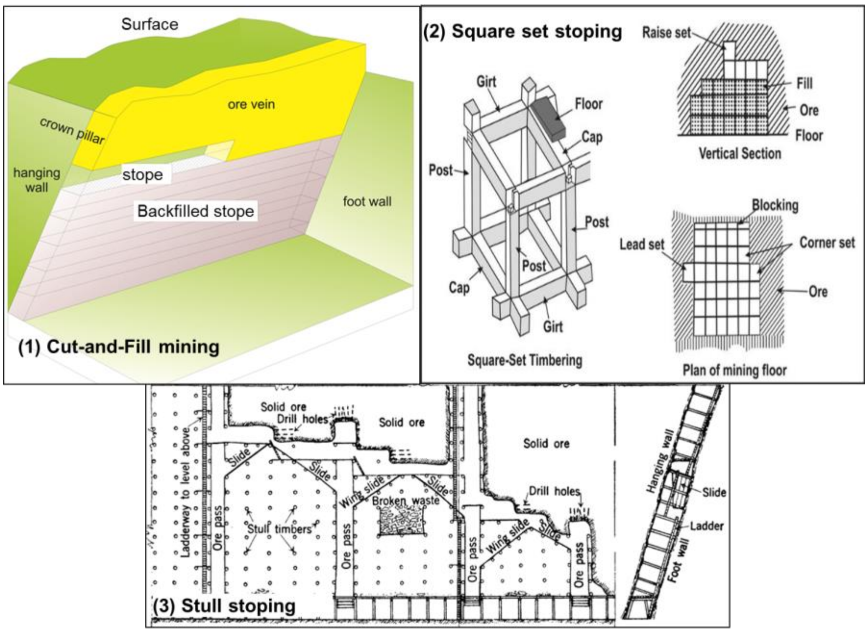

Figure 1.5 Supported mining method; (1) cut-and-fill mining (2) square-set stoping (3) stull stoping, source: (Karian.T, 2016) (Harraz 2016).

A description of cut and fill stoping is necessary to avoid confusion with similar practices that are part of other mining methods. The method is primarily utilized for steeply dipping vein deposits and large, irregularly-shaped deposits and ore value is relatively high

(Hamrin 2001). Ore deposit is mined in a horizontal slice while the mined out slice then backfilled by using backfill material to provide additional support for the country rock surrounding the stope. In fully-mechanized operation roadway is driven from the surface and connected to the stope by using cross cuts. High ore recovery rate (90% - 100%), adaptable to mechanization, possibility to use surface waste as filling material are among advantages of this method (Hartman 1987). Discontinuous operation due to filling operation is one of disadvantages of applying this method. Another major disadvantage is its cost which fairly expensive compared to the others due to labor extensive and, mainly, the application of backfilling. Backfilling application may take up to 20% of the total operating cost of the mine (Grice 1998). In addition, conventional ground support in the form of rock bolts, cable bolt, wire mesh, and shotcrete still will be required to temporarily support the stope openings during production mining.

There are numerous types of cut-and-fill mining methods, however the widely practiced method is overhand cut-and-fill mining method (Purwanto 2015). This type of mining typically done upwards from lower levels, so the backfill is used to provide a new working platform for further mining. Overhand cut-and-fill mining method can be applied in the good quality rock mass around the ore deposit. Application of overhand cut-and-fill method in weak rock mass is possible but may lead to excessive need of rock support (William A. and Richard L. 2001). In this method, Sill pillars and crown pillars are used to separate stope vertically to make a stable working environment. The dimensions of these pillars are determined by the geotechnical environment and the stresses induced by mining (Stephan 2011). As the stability of these pillars is one of the important issues in overhand cut-and-fill stoping, the optimum pillars should be determined to avoid the failure. Currently, most of the underground metal mines in Myanmar are applying the overhand cut-and-fill mining method in their operation.

1.3. Mine instability in cut-and-fill underground mine operation

Mine stability is the most important issues in underground mining. Mine development and/or production instability can cause production delay, loss of reserves, as well as injury to miners. As described above, most of the underground gold mines in Myanmar are operating with cut-and-fill mining method. However, the assessments on the stability of stope still remain quite limited. Currently, many underground gold mines are being mined-out or still mining at the easily accessible shallow places. After a period of mining, easily accessible shallow mineral resources are being mined out and the deposits of rest mine are left in deeper regions. Therefore, underground mining activities are going to continue to progress into deeper levels to fulfill the demand of gold, for example, the mining plan of Modi Taung gold mine as shown in Figure 1.6. Accordingly, stress condition in deeper mine will be greatly changed and the mining process would be more complicated.

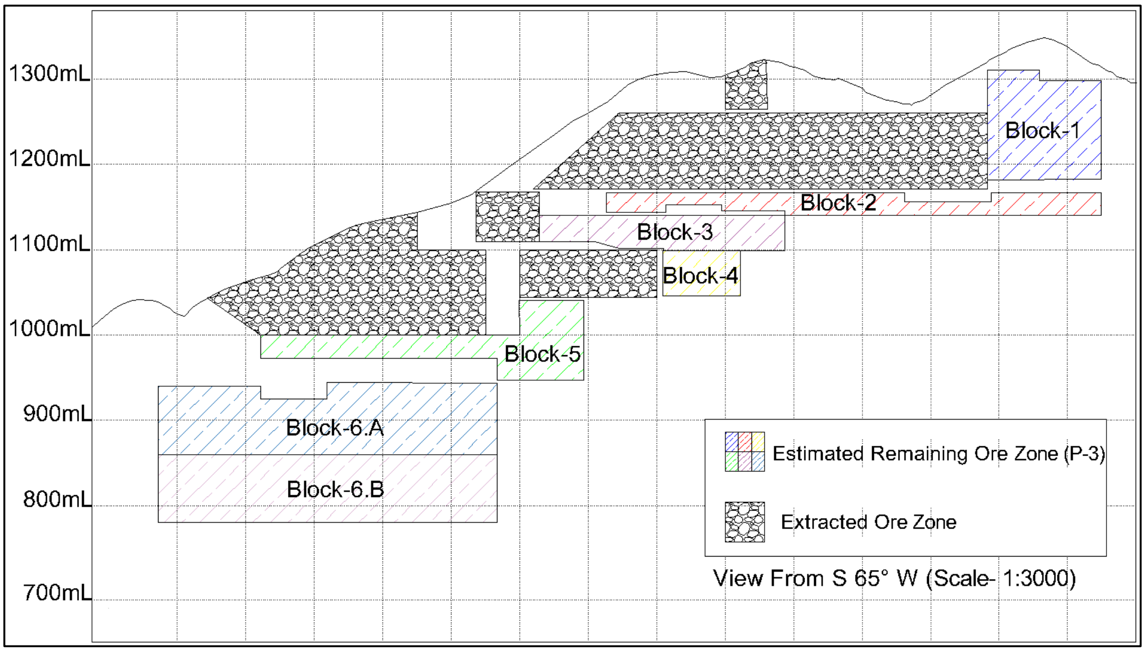

Figure 1.6 Simplified mine plan of Modi Taung underground gold mine (source: NPGPGL).

The stability of deep underground excavations depends upon the strength of the rock mass surrounding the excavations and upon the stresses induced in this rock (Evert Hoek 2000). Mining-induced stresses are the redistribution of field stresses as a result of the geometry and orientation of the excavations. Substantial stress redistribution directly influences the stability of the development openings. The effect of stress redistribution will vary depending on the geometry and orientation of the underground opening in relation to the in-situ stress field (Carlsson and Olsson 1993). Another factors affected the stability of underground excavation depend on the conditions of surrounding rock mass. The strength of rock mass will be reduced if there are many discontinuities and weathering of rock inside. At Modi Taung gold deposit, even though the host rock (intact condition) is strong, it can be found many discontinuities inside the rock mass especially in the oxidation zone of host rock as shown in Figure 1.6. Additionally, heavy rainfall is one of the causes of weathering of rock mass. The rate of water charge increases after periods of heavy rain which is common in this area. This meteoric water interacts with the surrounding rocks which result in weathering of host rocks, leading to deterioration of host rocks. All these conditions will give effects to the instability of underground excavations and should be paid attention to prevent the opening collapse. Thus, it is important to understand the stability of stope under near the previous mined-out area due to the redistribution and accumulation of stress under previous mining activities.

Figure 1.7 Rock mass condition showing discontinuities in Modi Taung gold mine.

Another factors for instability of stope is the in-situ stress distribution characteristics of the rock mass near the slope surface and the redistribution of these effects to the stope. Most of the primary gold deposits are generally hosted in mountainous regions in Myanmar. Considering the environmental and social impacts, therefore the underground mining method might be adopted in mountainous regions and the mining operations may be conducted under near the slope surface. Rock mass conditions under the slope surface may change under varying depths of overburden as well as the distribution characteristics of initial vertical and horizontal in-situ stresses (Li et al. 2017). If in-situ stresses are not equal from all directions, then the differential stress will appeared and affected to the surrounding rock mass. As well known, differential stress is the difference between the major and the minor principal stresses, and rock mass can deform (break/flow) due to differential stress. There are three kinds of differential stress: tensional stress which stretches rock, compressional stress which squeezes rock, and shear stress which result in slippage and translation (Nelson 2015) as shown in Figure 1.7.

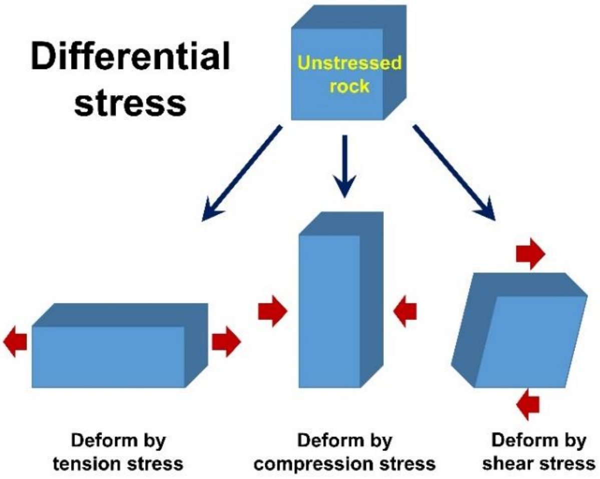

Figure 1.8 Differential stresses by tension, compression and shear force.

In the mountain slope, the in-situ stress values are much greater than the predicted values according to the vertical overburden depth, especially within 100 m from the slope surface (Li et al. 2017). In these areas, the horizontal tectonic stresses are always larger than the vertical in-situ stress. Therefore, the in-situ differential stress of that areas to the slope surface exhibits a higher value owing to the mountain slope effect (Li et al. 2017). Subsequently, the potential of rock instability become large at the mountain slope areas.

As described in above, the in-situ stress condition has an obvious influences on the stability of rock mass around the stope and slope surface. The rock medium is subjected to initial the stress prior to excavation. Finally, post-excavation state of stress in the structure is the resultant of the initial state of stress and stresses induced by excavation (Brady and Brown 2004). Therefore, the stability of stope under slope surface has to be paid attention not only by the stress induced by excavation but also the influence of topography.

In common practice, increasing crown pillar thickness under the slope surface could be a measure to prevent subsidence. The instability of stope will decrease with increasing the thickness of the pillars, however it will reduce mining recovery since higher volumes of ore body are left behind as a pillar (Stephan 2011). Therefore, determining the optimum thickness crown pillar and maintaining stope stability during ore extraction under the sloping area become the key to prevent surface subsidence and stope failure in cut-and-fill mining application.

1.4. Objectives of this research

As described above, most of the underground gold mines in Myanmar are being mined-out or still mining at the easily accessible shallow places. However, the assessments on the stability of stope still remain quite limited. Moreover, there are not so many recorded data regarding rock mass failures cases in underground mines of Myanmar. Hence, the study on the stability of stope under previous mined-out activities become one of the important issues to mitigate the unpredictable rock/slope failures. In addition, because the impact of in-situ stress conditiion at the mountain regions, the investigations on the potential instability of rock mass in slope surface and the stability of stope influenced by the topography/slope surface need to be discussed. Considering the importance of the stability of stope under different conditions mentioned above, the objectives of this research are listed below:

1. Understanding geological and geotechnical characteristics of rock mass in research mine site.
2. Investigating the stability of stope under previous mined-out activities in various mine conditions.
3. Evaluating the deformability of rock mass influenced by the mountain slope surface and the potential instability of underground mining affected by slope surface.
4. Introducing the effective rock supports as countermeasure method for stope stability by decreasing the risks of slope slide surface and the effect of previous mined-out activities.

1.5. Outlines of dissertation

The following chapters are included in this dissertation:

Chapter 1 introduces the background of this research, the overview of cut-and-fill mining method, factor affecting on the stability of underground mine opening, objectives of this research, and outlines of dissertation.

Chapter 2 describes the background of research mine site, Modi Taung gold mine, regional and detail geology, ore deposit and geotechnical informations of this mine site.

Chapter 3 discusses the potential instability of stope mining under previous mined-out activities. The instability of new stope opening under overlaying mined-out regions are investigated in various mining conditions by means of numerical simulations.

Chapter 4 discussed the importance on the evaluation of rock mass condition before mining and the potential failure of surrounding rock during mining under sloping surface. The occurrence of in-situ differential stress near the mountain slope and the influence of slope effects to the underground opening are investigated.

Chapter 5 introduces the application of countermeasures to maintain the stability of stope influenced by the effect of previous mined-out activities and the risk of slope failure. Two types of countermeasures, cable bolt and shotcrete support, are used to maintain the stability of stope and pillars.

Chapter 6 summarizes the conclusions of each chapter.

CHAPTER 2

MINE BACKGROUND, GEOLOGY, AND ROCK MASS EVALUATIONS

2.1. Background

Myanmar has more than 300 gold deposits across the country which are classified as either primary or alluvial types (Swe, Aye, and Zaw 2017). Most of the primary gold deposits are hosted at the mountainous regions as shown in Figure 2.1 and mostly are extracted by underground mining methods. Open stope mining is the most common mining method adopted in underground metal mines in Myanmar. However, the assessments on the stability of underground mining still remain quite limited in those mining industries. Moreover, there are not so many recorded data regarding rock mass failures cases in underground mining in cut-and-fill mining methods. Hence, the study on the stability of underground openings become one of the important issues to mitigate the unpredictable nature of rock failures. In order to understand the stability of underground excavation under previous mined-out activities and mountain regions, the purpose of this research, the investigations on deposit’s geology, host rock condition, and geotechnical characteristics are firstly carried out at the Modi Taung gold mine which is one of the largest underground gold mines in Myanmar.

2.2. History of Modi Taung gold mine

The Modi Taung gold mine is situated 1,200 m above sea level in approximately 150 km southeast of Mandalay and 385 km north of Yangon as shown in Figure 2.2. Alluvial gold was first discovered in local river of south of the Modi Taung lease area in 1996. In August 1996, an agreement was signed between the Ivanhoe Myanmar Holdings (Exploration) Ltd. (IMHL) and the government to explore in Block 10 area as shown in Figure 2.3, covering approximately 1,400 km^2 in central Myanmar. Starting from that time, the company was carried out the exploration works in Block 10, and ceased operations in August, 2005. The IMHL has identified a 100 km^2 gold district in their Block 10 area, the Modi Taung–Nankwe gold district, with features characteristic of slate-hosted mesothermal quartz–gold vein deposits (Mitchell et al. 2004). During IMHL period, the company had carried out 1,600 m of surface trenching, 7,000 m of underground drifting and raising, and more than 5,000 m of diamond drilling (IMHL 2003).

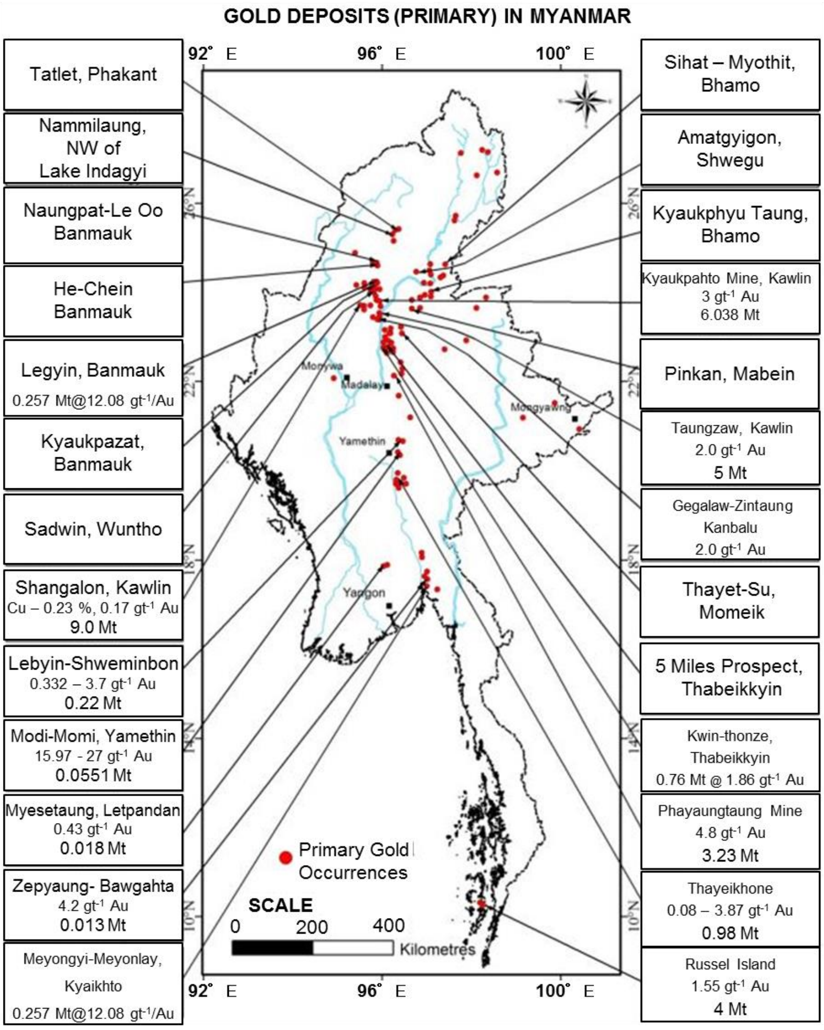

Figure 2.1 Primary gold deposits in Myanmar (Ye Myint Swe *et al*, 2017).

The mining lease was later taken over by the National Prosperity Group Production Group Limited (NPGPGL), a Myanmar based mining company, during October 2011 to February 2018. During their period, the company had explored for finding new gold discoveries, and many drilling projects are carried at Modi Taung gold mine. According to NPGPGL, the estimation of total gold reserve is 14,647 kg with the contribution of 2,193 kg from Htongyi vein, 167 kg from Seintaung (Htongyi vein), 2,421 kg from Sinthay (Htongyi vein), and 9,866 kg from Shwesin vein system. During NPGPGL mining periods, the refined gold production from Modi Taung gold mine had increased year by year as shown in Figure 2.4. Compared to the gold production of the whole Myanmar as shown in Figure 2.5, it can be seen that the contribution of the production of refined gold of Modi Taung gold mine is over half production of the whole Myanmar yearly. Moreover, according to ore reserves, the potential of gold production in this mine in the future might be increased.

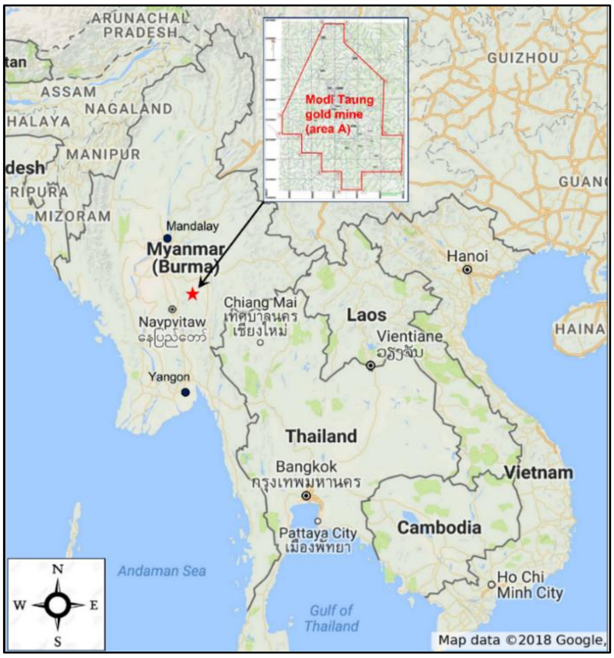

Figure 2.2 Location of Modi Taung gold mine.

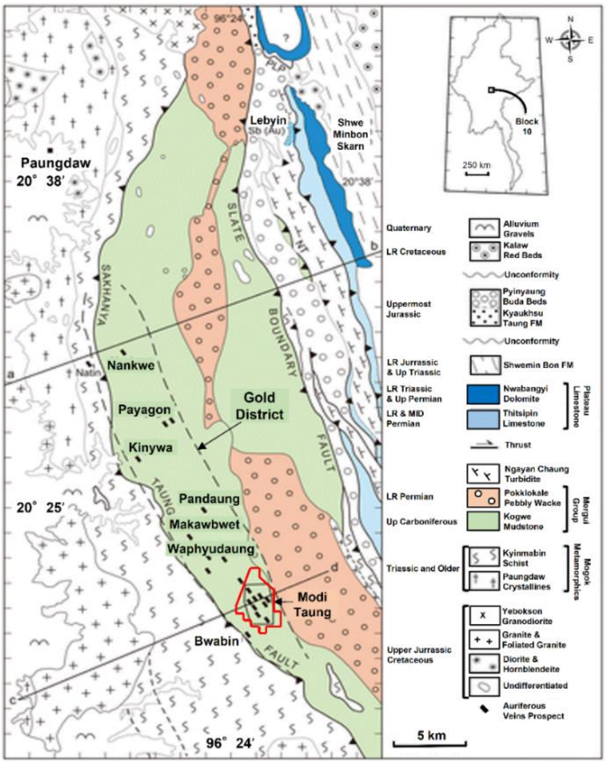

Figure 2.3 Geological map of Block 10 concession area (Mitchell et al. 2004).

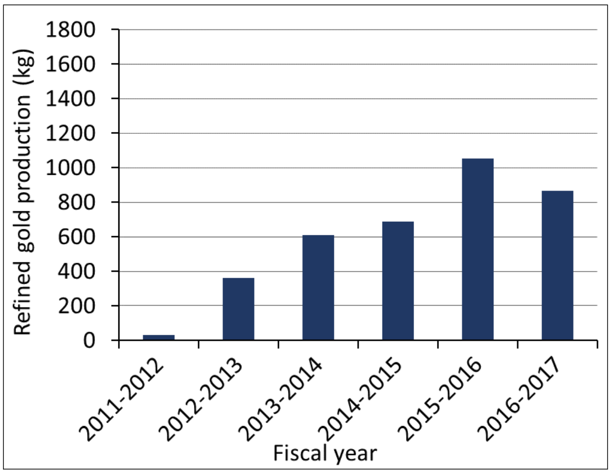

Figure 2.4 Refined gold production of Modi Taung gold mine during NPGPGL period (source: No (2) Mining Enterprise, Myanmar).

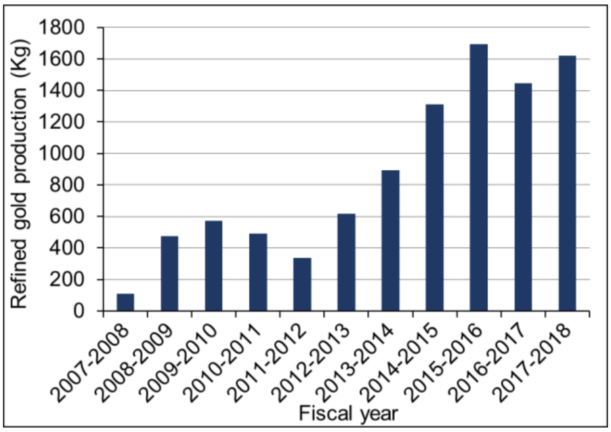

Figure 2.5 Refined gold production of Myanmar (source: No (2) Mining Enterprise, Myanmar).

2.3. Regional geology of Block 10

The Modi Taung – Nankwe gold district in Block 10 concession lies very largely within the slate belt of central Myanmar (Mitchell et al. 2004). This slate belt, consisting of a late Paleozoic slaty mudstone and sandstone succession, extends southwards through southern Myanmar, western Thailand and central Sumatra (see Figure 2.6). The Modi Taung – Nankwe mineralization is the first reported deposit of slate-hosted mesothermal quartz – gold veins in Southeast Asia, and it is interested due to the economic potential of gold district and potential for similar deposits elsewhere in the Slate belt (Mitchell et al. 2004). The deposit is hosted in the sedimentary units of the Mergui Group, which is composed of two dominant sedimentary formations. The lower part consists of massive to laminated mudstone, sandstone, rare limestones and channel-fill pebbly wackes while the upper part includes several polymict conglomerates.

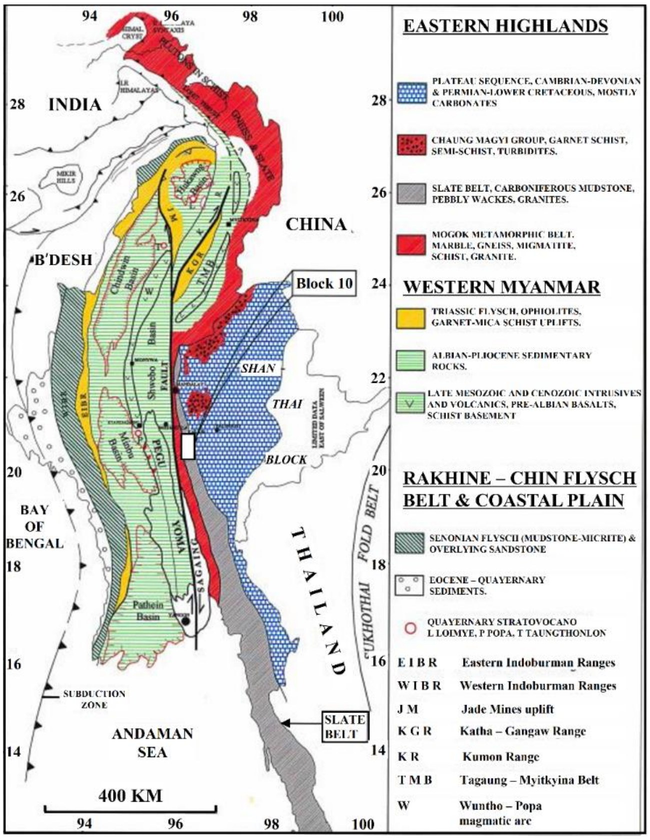

Figure 2.6 Myanmar regional tectonic map and location of Block 10 (IMHL 2003).

2.4. Local geology and mineralization of Modi Taung gold mine (area A)

The Modi Taung gold mine (area A) is situated in the southern part of Block 10 concession. Block 10 concession is divided into five areas, A, B, C, D and E by Myanmar government. The main areas of production are concentrated in area A, B and D with area A being the richest of ore deposit, most mined and the largest one.

There are three main veins in Modi Taung gold mine (area A) namely: Shwesin, Sakangyi, and Htonegyi (Mitchell et al. 2004). These veins are hosted by four main lithologies: mudstone, sandstone-siltstone, limey sandstone or limestone, and igneous intrusions. Mudstone is the predominant rock types in all vein systems but sandstone occupies short segments, and veins tend to occur along the inclined interface between sandstone and mudstone. Their competence and hardness increase with depth from soft clay immediately beneath soil cover to a hard rock that is tough and competent with the exception of moderate hardness near and below the base of the oxide zone. Sandstone and siltstone are mostly silicified and cut by quartz stockworks, forming quartzite. Ground conditions are poor in Shwesin vein system and within 60 m from surface where partial oxidation has occurred (IMHL 2003). The detail geological structure of Modi Taung gold mine (area A) is shown in Figure 2.7 (Erskine 2014) (Traynor 2015).

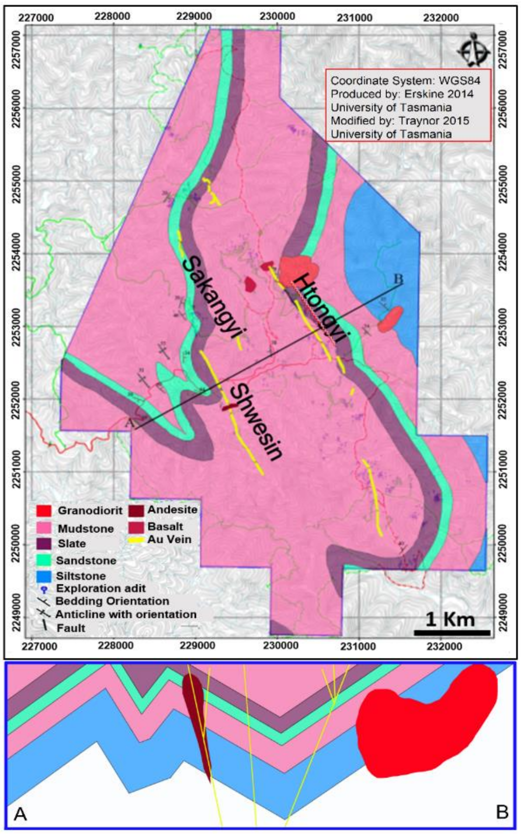

Figure 2.7 Detail geological map of Modi Taung gold mine (area A) (Erskine 2014).

Almost all of the auriferous veins in Block 10 lie within the 25 km x 5 km Modi Taung – Nankwe gold district. Even though there are three main vein system in Modi Taung region, other vein systems are also mineralized in the surroundings as shown in Figure 2.8 (IMHL 2003). Veins in east of least area are dipping steeply to the west, while veins in the west are dipping steeply to the east (Mitchell et al. 2004). Each vein system consists of either a single vein, or multiple parallel vein separated by host rock. Vein width varies with elevation and ranges between centimeter and meter scale (Mitchell et al. 2004) (Erskine 2014). At Shwesin vein system, the vein consists of two main veins: eastern vein and western vein, both are striking NNW (Figure 2.9). Eastern vein maintains a constant thickness of 40 cm and dips up to 80° to the west, and western vein dips 75° to the east and ranges in thickness from 60 cm to 140 cm (Erskine 2014). All veins are massive quartz and the oxidation is likely due to groundwater interacting with the ore through a leached zone. The Modi Taung gold deposit (area A) is economically viable due to three main reasons; it has concentrated high gold grades of 10-300 g/t Au and up to 3,000 g/t Au, large lateral extent and if further investigated may be proved to be even larger than expected, and steeply dipping veins that continue to depth which makes stoping and extraction of ore easier (Traynor 2015).

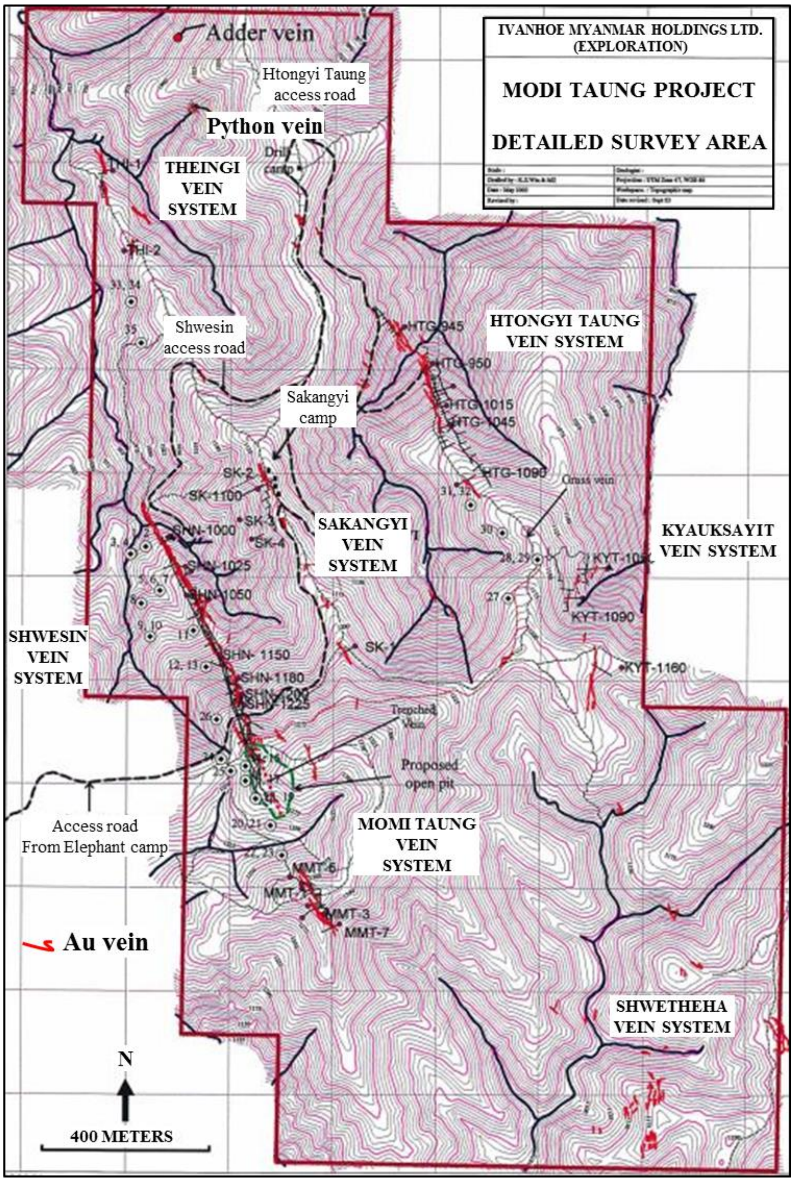

Figure 2.8 Au vein systems mineralized in Modi Taung region (IMHL 2003).

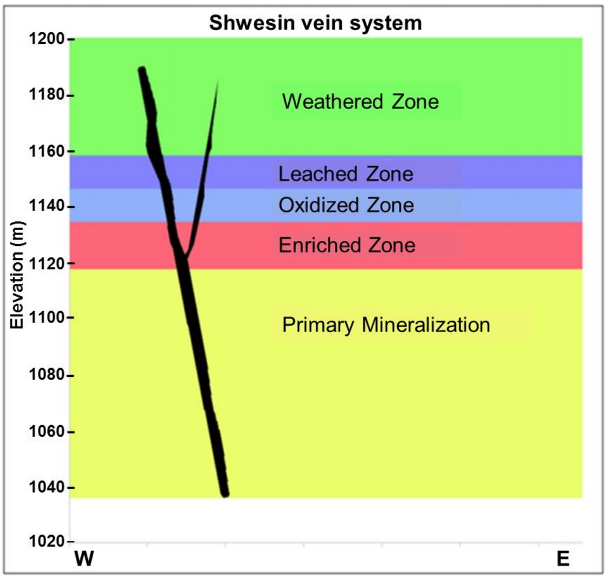

Figure 2.9 Schematic diagram of Shwesin vein orientations (Erskine 2014).

2.5. Rock mass evaluation from field observations and experiments

The instability of underground excavation depends on the behaviors of surrounding rock mass. Different rock types have different characteristics that can influence their mechanical behaviors. Therefore, knowledge and understanding of rock mass condition are essential for stability of underground excavations. The first step for describing rock mass is to examine rock mass properties determined by lithology, laboratory tests and field observation data. The second step is to determine the geotechnical information of rock mass for the purpose of rock engineering design such as numerical modelling, analytical calculation, etc. Considering the importance of rock stability in stope opening, field observation for lithology, geology and mining system was conducted, and laboratory experiments was carried out to get the physical properties of rock mass.

At Modi Taung gold mine gold mine, Htongyi Taung and Sakangyi vein systems are hosted by mudstone, while the host rocks in the Shwesin, Sakangyi and Momi Taung systems are predominantly mudstone or siltstone and the rest sandstone. As described in Section 2.4, host rocks from Modi Taung gold deposit are sedimentary units and rock mass conditions from shallow part are poor. From the borehole data, the RQD and depth from Modi Taung gold mine are shown in Figure 2.10 and the correlation between RQD value and rock mass quality is shown in Table 2.1 (Lucian.C. and E.M. 2013). According to this RQD data, the rock mass of Modi Taung gold mine can be classified as very poor condition within 30 m from the surface, and poor to fair condition more than 80 m depth from the surface. Furthermore, rock mass parameters that obtained from laboratory experiments are shown in Table 2.2 and the uniaxial compressive strength of intact host rock and vein from Modi Taung gold mine are 148 MPa and 140 MPa, respectively. However, some activities such as discontinuities, persistence, aperture, rock roughness, and weathering of rock are conducted to complete full estimation of rock mass condition.

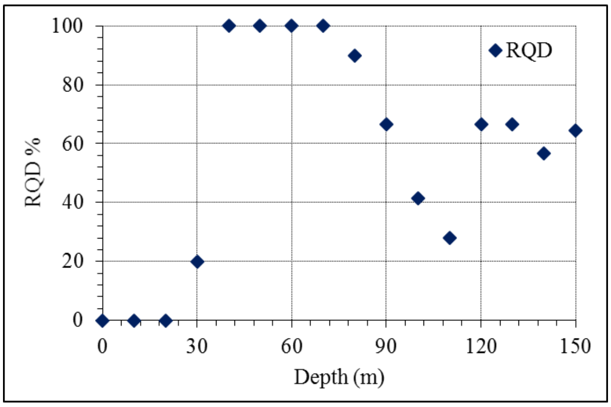

Figure 2.10 Relation between RQD data and depth of Modi Taung gold mine (source: NPGPGL).

Table 2.1 RQD classification index (Deere et al. 1967).

RQD	Rock mass quality
0 – 25 %	Very poor
25 – 50 %	Poor
50 – 75 %	Fair
75 – 90 %	Good
90 – 100 %	Excellent

Table 2.2 Intact rock parameters obtained from laboratory experiments.

	ρ [kg/m^3]	E [GPa]	v [-]	σ_t [MPa]	φ [deg]	C [MPa]
Host rock	2,717	19	0.25	10.4	58	18.5
Vein	2,667	12	0.22	3.8	71	11.1

According to these intact rock parameters, it can be seen that the rock mass strength of Modi Taung gold mine is strong. However, rock mass properties which do not consider geological structure are not proper if used into any form of analysis for the design of underground excavations. On the other hand, wrong engineering geological estimations will effect to all numerical analysis and calculations. For this situation, the Geological Strength Index (GSI) introduced by Hoek, E., Kaiser, P.K. and Bawden, 1995 is very essential to estimate the rock mass strength for different geological conditions (E. Hoek, Kaiser, and Bawden 1995). The Geological Strength Index (GSI) provides a number which, when combined with the intact rock properties, can be used for estimating the reduction in rock mass strength for different geological conditions (E. Hoek and Brown 1997). From the field observation at Modi Taung gold mine, many cracks and joints within rock mass are found in underground tunnels and stopes as shown in Figure 2.11. These rock mass conditions will be effected to the instability of underground excavations and should be paid attention to prevent the collapse of stope.

Additionally, heavy rainfall is one of the causes of weathering the rock mass. From the annual rainfall data of Modi Taung area recorded by IMHL, the data shows an average annual precipitation of 2,228 mm with a maximum annual precipitation of 2,900 mm recorded in 2002 (unpublished exploration report of IMHL, 2004). Conditions in adits at Modi Taung gold mine are very humid with meteoric water seeping through geological structures in the hanging wall as well as foot wall. The rate of water charge increases after periods of heavy rain which is common for this area. The seeping pathways for meteoric water flow appear to be through a complex system of fracture in the weathered host rock (see Figure 2.12). These meteoric water are interacting with surrounding rocks and can be affected to the weathering of host rocks, and then reduced the strength of host rocks. Consideration of rock mass condition including several factors such as RQD, joint spacing, the condition of joints, and weathering, the rock mass properties can be estimated as shown in Table 2.3.

Figure 2.11 Rock mass condition showing joints and cracks.

Figure 2.12 Underground adits condition seeping with meteoric water.

Table 2.3 Rock mass properties evaluated with geological conditions.

	ρ [kg/㎥]	E [MPa]	ν [-]	σ_t [MPa]	φ [deg]	C [MPa]
Hanging wall	2,670	3,786	0.23	0.035	44	0.76
Foot wall	2,717	3,786	0.25	0.065	40	0.69
Vein	2,667	3,786	0.22	0.034	45	0.77

2.6. Conclusions

In this chapter, the preliminary studies about mine background, geological conditions, rock mass properties, and mineralization of Modi Taung gold mine are conducted to evaluate the stability of underground mining for next research. From the previous researches of IMHL, the Modi Taung gold mine is interested due to the economic potential of gold reserve and potential for similar deposits elsewhere in this area. According to Traynor 2015, the Modi Taung gold deposit (area A) is economically viable for its high grade ore (10-300 g/t Au and up to 3,000 g/t Au), the deposit amount may be larger than expected, and veins are steeply dipping and can continue to depth which makes stoping and extraction of ore easier. All of these data can bring the interest for many investors in this gold mine.

Regarding the rock mass condition, the rock strength from Modi Taung gold mine is strong by seeing the intact rock parameters. However, according to the RQD data and field observations, the rock mass of Modi Taung gold mine is very weak in shallow regions and poor to fair rock mass condition in deeper regions. Therefore, these rock mass conditions have obvious impacts on the stability of slope/surface, accordingly it will affected to the slope slides and the instability of stope mining in shallow regions.

So far, the mine projects in Modi Taung gold mine are already developed in most of shallow ground parts, therefore the mine will be continued to the deeper levels under the previous mined-out regions. Accordingly, the influence of stress redistributions and potential of failure accumulation will be affected to the mine opening under the previous mined-out regions. Therefore, the investigations for the stability of underground mining should be conducted before the development of mining projects in Modi Taung area in the near future.

CHAPTER 3

INSTABILITY OF STOPE MINING UNDER PREVIOUS MINED-OUT ACTIVITIES

3.1. Background

Stability of underground openings is a major concern for the safety and productivity of mining operations. Mine development and/or production instability can cause production delay, loss of reserves, as well as injury to miners. Within the scope of this study, a series of open stope's instability under the influence of overlaying mined-out regions were carried out with different mining scenarios at Modi Taung gold mine. So far, most of the shallow parts of Modi Taung gold mine are already mined-out and mining activities are going to continue to progress to deeper levels in order to fulfill the demand of gold. It has been well known that stress condition in deeper mine will be greatly changed and the mining process would be more complicated accordingly, particularly when it is operated under overlaying mined-out regions. Creating a new stope under overlaying mined-out regions is not easy considering the instability of mined-out regions can affect the stope. The instability of new stope is not only due to its own induced stress but also the effect of the mined-out regions situated on upper part of the stope. Therefore, the understandings of ground behaviors and failure mechanisms of new stope due to the influence of overlaying mined-out regions are paramount to be studied.

3.2. Current mine condition of Modi Taung gold mine

As described in Chapter 2, there are three main vein systems in Modi Taung gold mine, and this study is focused in Shwesin vein system among these three main veins. At Shwesin vein system, the accessible shallow area has been already mined, hence, the deposits are left in deeper regions. Therefore, the mining plans to continue to deeper levels, in which separated into 6 blocks, namely from Block 1 to Block 6. The overall mine plan is illustrated in Figure 3.1. According to this mine plan, most of the blocks are located under previous mined-out regions, hence the new mine developments might be affected not only by its induced stress but also by the effects of induced stress from previous mined-out activities. Overhand cut-and-fill mining method is adopted to extract the minerals at Shwesin vein and the dimension of stope is 2.5 m in height and 2 m in width. The waste rocks from excavation are only used to both fill the stope and provide permanent wall support for the lower mine-out cavity.

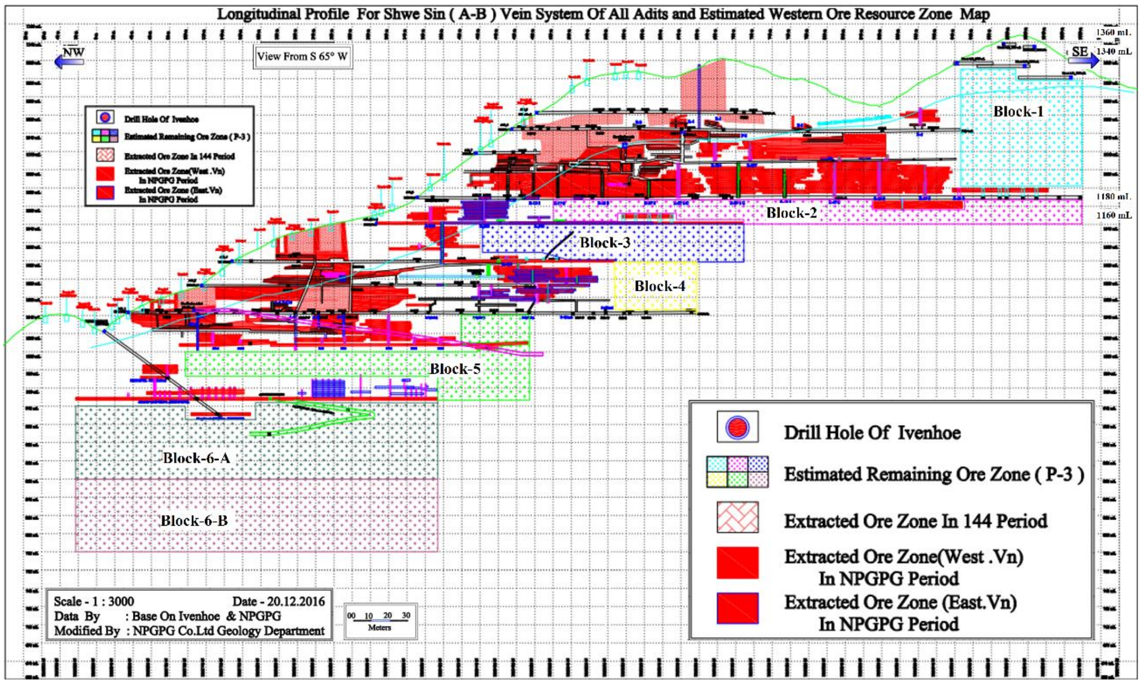

Figure 3.1 Mining plan at Shwesin vein system (source: NPGPGL).

3.3. Description of numerical modelling and mining plan

Numerical simulation is an effective way to examine the stability of a stope. In this study, the failure conditions of new stope because of overlaying mined-out regions were investigated by means of the 3D finite difference code (FLAC 3D) for preliminary study. FLAC 3D is a numerical software which is widely used for analyzing stress and deformation around the surface and underground openings conducted in both soil and rock (Itasca Consulting Group Inc. 2012). The size of the basic numerical model is 250 m × 250 m × 250 m with 70 degrees in vein dip as shown in Figure 3.2. The bottom of the model was fixed in the vertical direction, the sides were fixed in the horizontal direction, and the surface was free in all directions. The slaty mudstone is a predominant rock type in Modi Taung gold mine, and therefore the hanging wall and footwall are assigned as homogenous model for simplification. The mechanical properties of host rock and vein for basic model are given in Table 3.1. Moreover, to obtain more precise result of the rock failure distribution, the smaller mesh size was created around the excavation area. Overhand cut-and-fill stope mining in Block 2 was conducted with the stope dimension of 2.5 m in height and 2 m in width as shown in Figure 3.3. During stoping procedure, the stope 1 is firstly mined for overhand cut-and-fill method, and the model is computed for the stability of stope before backfilling. After that backfilling material is filled and analyzed for the stope 1, then the excavation is continued for stope 2. This sequence is advanced in the planned mining zone until the final stope 8. Block 2 is the planned mine zone with 24 m in height under overlaying mined-out regions with 100 m in height, and the total overburden above Block 2 is 150 m. This study is carried out for the stability of current stope in Block 2 in various mining conditions which is not only influenced by its own induced stress but also affected by overlaying mined-out regions.

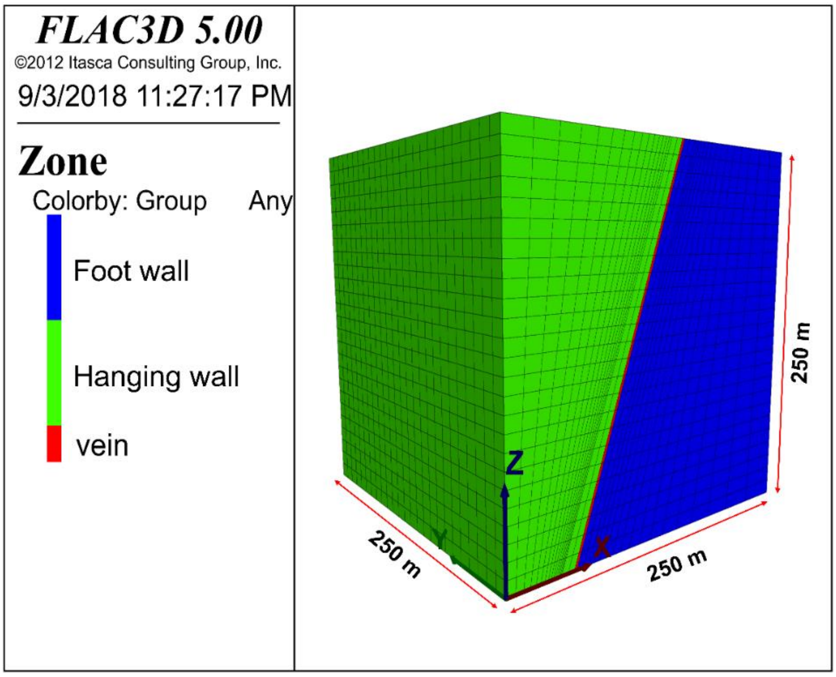

Figure 3.2 Basic numerical model for research study.

Table 3.1 Rock mass parameters of basic model.

	ρ [kg/m^3]	E [MPa]	ν [-]	σ_t [MPa]	φ [deg]	C [MPa]
Hanging wall	2,670	3,786	0.23	0.035	44	0.76
Foot wall	2,717	3,786	0.25	0.065	40	0.69
Vein	2,667	3,786	0.22	0.034	45	0.77
Filling rock	1,400	153	0.321	-	20.5	0.20

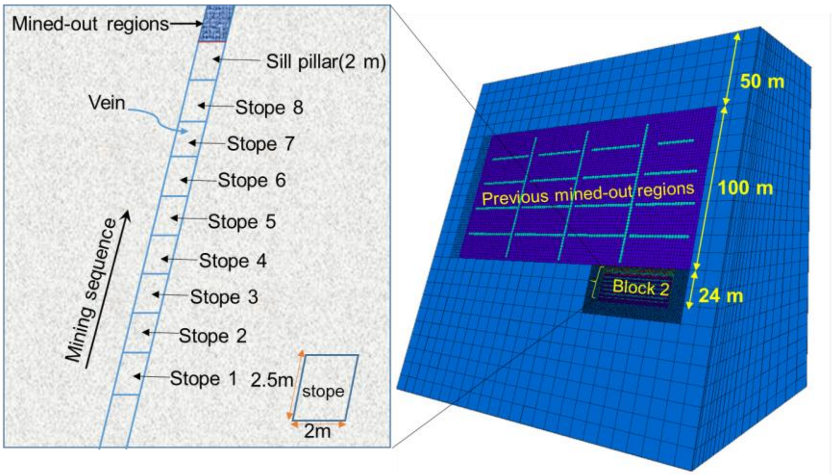

Figure 3.3 Overhand cut-and-fill mine plan at Shwesin vein.

3.4. Failure criterion

Mining objective is to recover ore as much as possible from the vein. However, safety of workers in the advancing stopes must be ensured. Potential hazards in the stopes are rock falls from the stope's roof and buckling failures in the hanging wall and footwall. In order to stabilize the stope, a failure criterion must be selected. A factor of safety of 1.3 would generally be considered as a stability standard for a temporary mine opening while a value of 1.5 to 2.0 may be required for a permanent one (E. Hoek, Kaiser, and Bawden 1995). However, the selection of an appropriate factor of safety is based upon engineering experience and field observation. In this research, the Mohr-Coulomb failure criterion is adopted as shown in Figure 3.4 and elasto-plastic behavior of the rock mass is used. The factor of safety (strength factor) is calculated by dividing the strength of rock mass by the induced stress of stoping activities to provide a basis of stability assessment as follows (W. Abdellah et al. 2014):

$$\text{Factor of Safety} = \{c \cos\varphi + [(\sigma_1+\sigma_3)/2 \text{ x } \sin\varphi]\} \,/\, [(\sigma_1-\sigma_3)/2]$$

where c is cohesion, φ is friction angle, σ_1 is major principal stress, and σ_3 is minor principal stress. In this research, a factor of safety (strength factor) of 1.3 is adopted for the stability of temporary stope mining. The stope was considered in a stable condition when Mohr-Coulomb strength factor is greater than 1.3 and unsatisfactory condition will meet when the safety factor is less than 1.3.

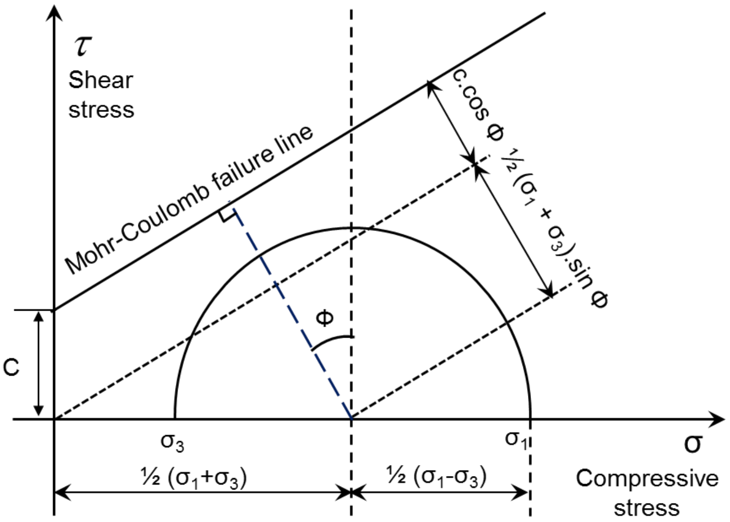

Figure 3.4 Mohr-Coulomb failure criterion.

3.5. Results for numerical analysis

In general, no one can estimate the rock mass is stable or not without numerical simulations. Determining the tunnels and stopes are stable or unstable should be based on yield zones from numerical simulations. Therefore, to understand the instability of stope opening under overlaying mined-out regions, numerical simulations were firstly conducted for the stope without overlaying mined-out effects compared with simulations for the stope with overlaying mined-out effects. Additionally, for more understanding of stope's instability under overlaying mined-out regions, numerical simulations have been observed with various mine conditions such as different vein dips, different vein widths, different stress ratios for vertical/horizontal, different geological conditions and different backfilling materials. The explanations of failure terms given in the legend in Flac 3D software are as follows (Yasitli and Unver 2005):

1) "none" indicates no-failure zone,
2) "shear-n" indicates the region failed under shear loading and failure process is still in progress,
3) "shear-p" indicates the region failed under shear loading and failure process is stopped due to lowered amount of shear forces.
4) "tension-n" indicates the region failed under tensile loading, and failure process is still in progress,

5) “tension-p” indicates the region failed under tensile loading, and failure process is stopped due to lowered amount of tensile forces.

3.5.1. Stability assessment for the influence of overlaying mined-out regions

First of all, numerical simulations are carried out with the aim to understand the influence of previous mined-out activities and the potential instability of stope due to the effects of previous mined-out activities. When an opening is excavated in rock mass, the stress field is locally disrupted and a new set of stresses are induced in the rock surrounding the opening (E. Hoek, Kaiser, and Bawden 1995). Moreover, if the underground mining operation is conducted under the previous mined-out regions, the excavation of later mining zone might be affected not only by its own induced stress but also by the effects of the redistribution stress from previous mined-out activities.

Figure 3.5 (A) shows the potential displacement of surrounding rock mass to stope opening without previous mined-out activities and Figure 3.5 (B) indicates the situation of new stope opening under previous mined-out regions. The result of Figure 3.5 (A) shows that the potential displacements to the stope can affect by its own induced stress while the other one suggests that the displacement to the new opening come not only by induced stress but also the influence of the redistributed stress from previous mined-out activities. Based on these results, it can be said that the rock mass are already disturbed by the induced stress of previous mined-out activities. This phenomenon can be proved by Figure 3.6. Figure 3.6 (A) shows the failure zone which occurred around the stope without the influence of overlaying mined-out. Whereas, Figure 3.6 (B) shows the failure zone developed with the influence of overlaying mined-out. Based on the simulation results which are shown in Figure 3.6 (A), the failure zone around the stope increases steadily as the stope progresses move towards the upper slices. These trends represent that failure zone of current stope is accumulated to the next stope. Compared with the failure condition under the influence of the overlaying mined-out effects as shown in Figure 3.6 (B), the failure characteristics of surrounding rock masses in Figure 3.6 (B) are larger than that of in Figure 3.6 (A). The statement can be addressed from these two results that the redistributed stresses from overlaying mined-out regions are surely propagated to the current stope mining activities. As a result, the development of failure zone of current stope is increased. This increment is not only due to its own induced stress but also affected by the redistributed stresses of overlaying mined-out regions.

Additionally, the instability of the sill pillar near the overlaying mined-out regions became more severe when the stope mining reaches to the uppermost level (i.e. the nearest stope to the upper mined-out regions) as shown in Figures 3.7 (A) and (B) that indicate the contour of safety factor propagated around the stope opening. Moreover, as roof stability is primarily determined in many underground mines especially in the presence of structural discontinuities, the monitoring points for the factor of safety are placed 0.5 m and 1 m above the center of stope's roof as shown in Figure 3.7.

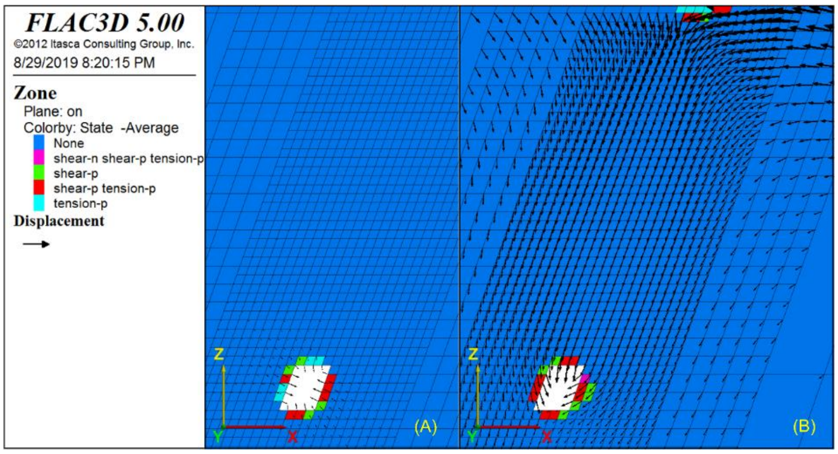

Figure 3.5 Displacement of rock mass (A) condition without previous mined-out effects (B) condition with previous mined-out effects.

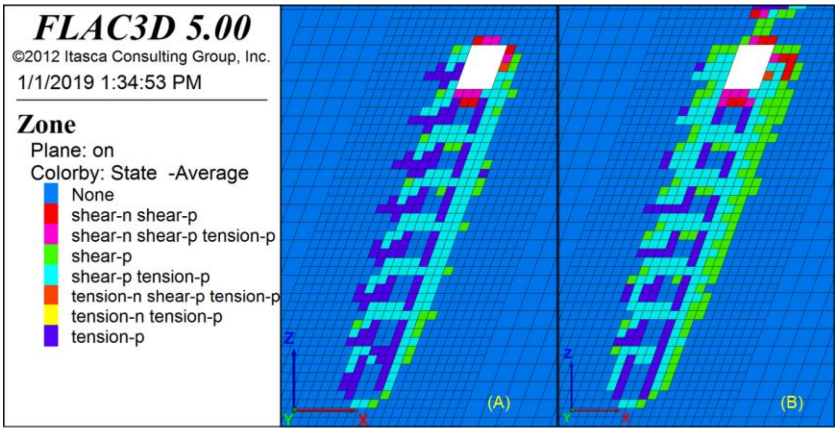

Figure 3.6 Failure zones occurring at the stope (A) condition without previous mined-out effects (B) condition with previous mined-out effects.

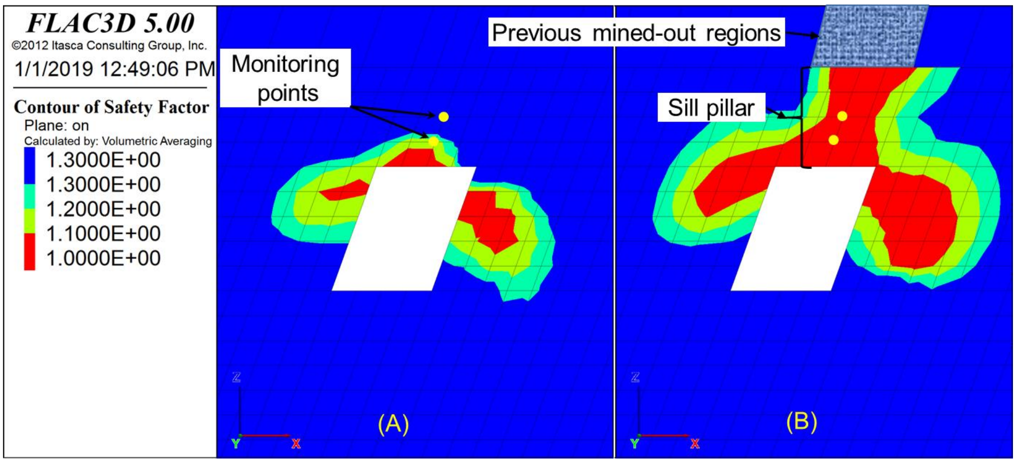

Figure 3.7 Unstable regions around the stope (A) condition without previous mined-out effects (B) condition with previous mined-out effects.

In this study, the thickness of sill pillar is set to 2 m between the final stope and upper mined-out regions considering the current condition of sill pillar at Modi Taung gold mine. Figure 3.8 proved the description by showing that the factor of safety gradually decrease as the mining steps increase. This result suggests that decrease in the safety factor makes increase in instability area around the stope. Based on the graph's result, the instabilities had appeared in 0.5 m above the stope's roof and it had been occurred a stable condition above 1 m from the stope's roof except the uppermost stope near the overlaying mined-out regions. For this sense, the numerical simulations for the stability of sill pillar are conducted to understand the possibility of unstable regions with different sill pillars and determine the appropriate thickness of sill pillar to maintain the stability of stope.

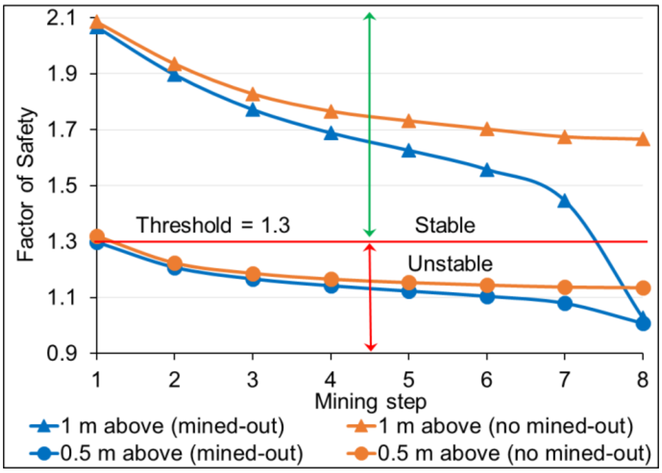

Figure 3.8 Safety factor indicators for various mining steps.

3.5.2. Stability of sill pillars with its heights

At Modi Taung gold mine, the sill pillar is broadly maintained 2 m thickness between the final stope and previous mined-out regions. As explained in previous section, the failure zone at the sill pillar had been propagating to previous mined-out regions. Therefore, numerical simulations with different pillar thickness (2 m, 2.5 m, 3 m and 3.5 m) were carried out to discuss the optimum thickness of the sill pillar.

Figure 3.9 shows that the failure conditions of sill pillar with different pillar thickness and Figure 3.10 indicates the unstable regions around the stope shown by contour color of safety factor. These results suggest that the failure zones and instability of final stope continue to the overlaying mined-out regions when the sill pillar is set to 2 m thickness, and decrease gradually when the thickness of sill pillar increases to 2.5 m, 3 m and 3.5 m continuously.

According to these results, the unstable regions are still propagated to the upper mined-out failure zone when the sill pillar is set to 2.5 m thickness. However, this condition did not continue when the sill pillars are set to 3 m and 3.5 m thickness. Therefore, these results suggest that the sill pillar between the final stope and overlaying mined-out regions should be maintained at least 3 m thickness to ensure stability of sill pillar. Additionally, the potential failures from the hanging wall and foot wall should be paid attention to prevent rock falls from the side walls of the stope.

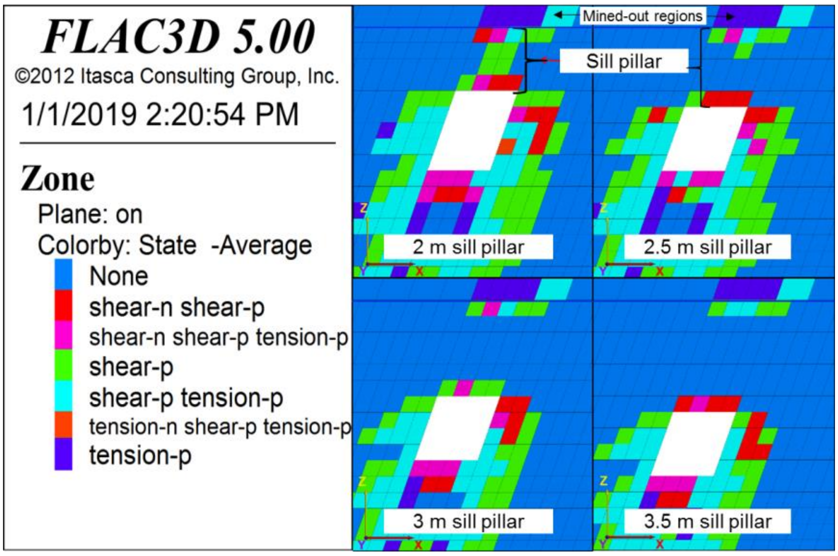

Figure 3.9 Failure zones of sill pillars with different pillar heights.

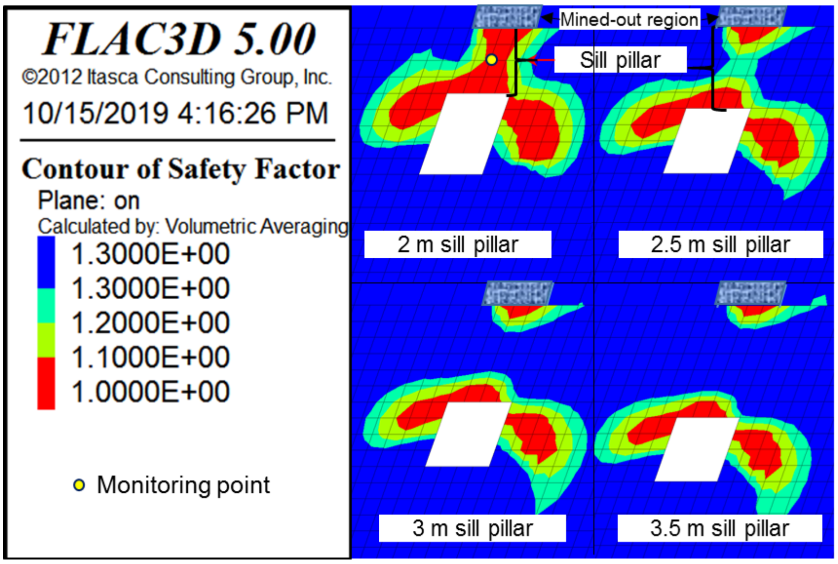

Figure 3.10 Contour of unstable regions of sill pillar with different pillar thickness.

3.5.3. Instability of sill pillar due to the effect of the deterioration of backfilling materials in previous mined-out regions

When the mining activities are carried out under the previous mined-out regions, it need to be taken account the influence of the deterioration of backfilling materials from the mined-out area. From time to time, the strength and deformability of backfilling materials might be getting worse, and this condition can effect to the stability of underground mining under previous mined-out regions. There are common understandings in some literatures that both strength and deformation characteristics of rock are changed when the meteoric and underground water is interacted with the surrounding rock mass. Generally, the strength of wet rocks is less than the strength of dry rock (Lajtai, Schmidtke, and Bielus 1987). In Modi Taung mining area, heavy rainfall is common during rainy season, therefore a large amount of surface water can seeping into the rock mass though discontinuities. As a result, the crushed backfilling waste rock in the previous mined-out regions can be deteriorated for a period of time, and this situation will influence to the stability of sill pillar under it.

Considering deterioration of backfilling waste rock in the previous mined-out regions, the numerical investigations are carried out with 25 %, 50 %, and 75 % deterioration of backfilling quality from the original properties, and the simulation results for the instability of sill pillar in each case are shown in Figure 3.11. By seeing these results, if the backfilling materials is deteriorated up to 50 %, the sill pillar can maintained at least 3 m thickness. However, if the backfilling properties is reduced up to 75 %, it is doubtful whether the sill pillar can maintain with 3 m pillar thickness because of a large potential of rock instability around the stope and pillar in shorter sill pillar condition. To make sure for safety, it is suggested to keep 3.5 m sill pillar thickness in case of 75 % deterioration in backfilling properties. The contour of safety factors monitored at 1 m above the final stope in each case are shown in Figure 3.12. The results show that the safety conditions of sill pillar are gradually decreased as the backfilling materials are deteriorated in the previous mined-out regions.

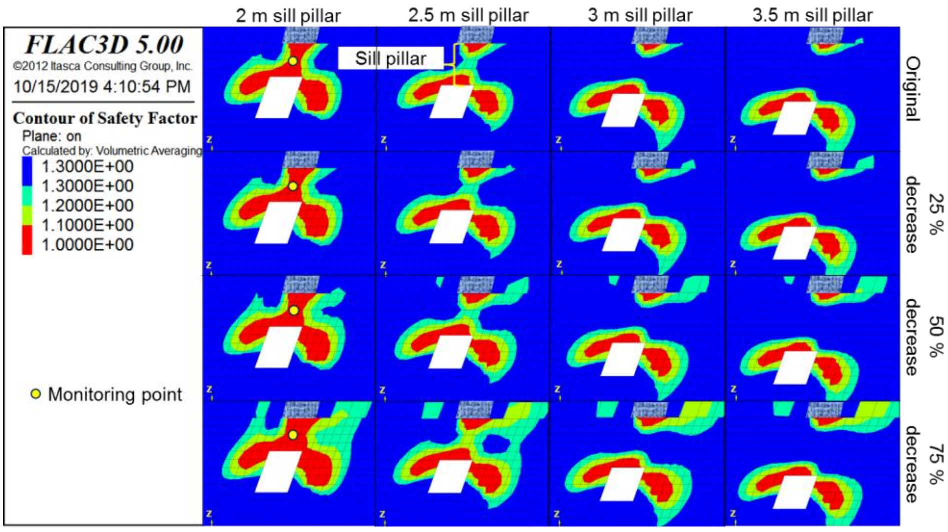

Figure 3.11 Instability of sill pillar due to the effect of deterioration of backfilling materials in previous mined-out region.

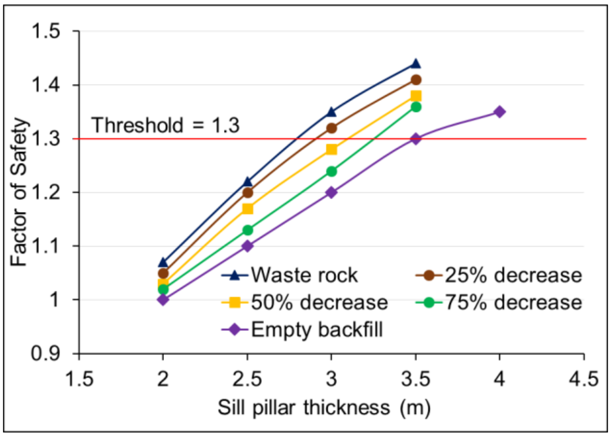

Figure 3.12 Contour of safety factor with different deteriorations of backfilling materials in previous mined-out regions.

In addition, during mining activities in previous mined-out regions, there might be a condition for empty backfilling material after mined-out the stope. For this condition, the simulation is carried out with empty backfill in previous mined-out regions by considering the worse condition for rock instability, and the results are shown in Figure 3.13. From a series of simulation results, it can be seen that the rock instability in the sill pillar is reached in serious condition when the thickness of sill pillar is small. For this condition, the optimum sill pillar is suggested to maintain at least 4 m thickness to ensure the stability of stope under the previous mined-out regions.

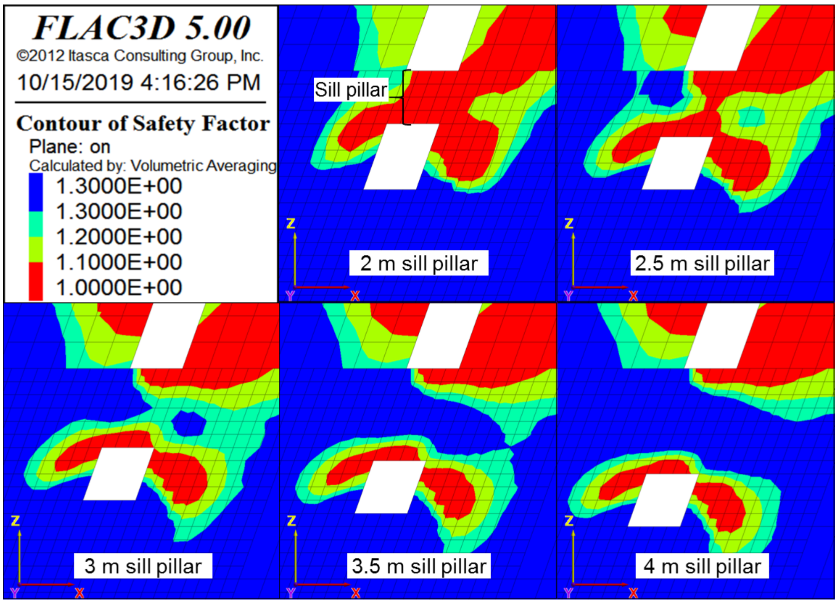

Figure 3.13 Instability of sill pillar with the influence of empty backfilling in previous mined-out region.

3.5.4. Stability assessment of stope under various geological conditions

Without a doubt, the strength of rock mass can differ from the variation of rock mass properties. Therefore, simulations for various geological conditions were carried out to understand the instability of open stope due to the influence of different geological conditions under overlaying mined-out regions. Geological conditions are represented by Geological Strength Index (GSI) parameter (Marinos, Marinos, and Hoek 2005). As mentioned above, the rock structure of Modi Taung gold mine consists of many cracks and joints. Rock mass properties are changed with geological condition as the basis, and all rock mass properties were determined for each GSI by using Hoek and Brown Failure Criterion (E. Hoek and Brown 1997). In order to understand the effect of different geological conditions, a series of numerical simulations were conducted with the various GSI numbers ranging from 39, representing blocky/disturbed/seamy rock mass folded with angular blocks formed by many intersecting discontinuity sets, to 49, representing very blocky rock mass with interlocked partially disturbed rock mass (Sonmez, Gokceoglu, and Ulusay 2004). The summary of rock mass properties, which are used in this simulation, is provided in Table 3.2.

Figure 3.14 shows the failure zone occurred around the stope with different GSI numbers. Whereas, Figure 3.15 indicates the unstable area which is shown by the contour color of safety factor around the stope with different sill pillars under previous mined-out regions, and Figure 3.16 shows the index of safety factor from the stope advancing activities that are monitored from 0.5 m and 1 m above the stope.

Table 3.2 Rock mass properties for simulations with various geological conditions.

Geological Strength Index (GSI)	Zone	Rock mass parameters				
		E [MPa]	v [-]	σ_t [MPa]	φ [deg]	C [MPa]
39	Hanging wall	3,190	0.23	0.027	43	0.70
	Foot wall	3,190	0.25	0.049	38	0.62
	Quartz vein	3,190	0.22	0.026	43	0.71
42	Hanging wall	3,790	0.23	0.035	44	0.76
	Foot wall	3,790	0.25	0.065	40	0.69
	Quartz vein	3,790	0.22	0.034	45	0.77
44	Hanging wall	4,250	0.23	0.042	45	0.81
	Foot wall	4,250	0.25	0.078	41	0.73
	Quartz vein	4,250	0.22	0.041	46	0.82
49	Hanging wall	5,660	0.23	0.067	48	0.93
	Foot wall	5,660	0.25	0.123	44	0.87
	Quartz vein	5,660	0.22	0.064	48	0.94

The results suggest that the geological conditions become more severe with decrease in GSI, the potential of failure zones around the stope is higher for lower GSI value. Accordingly, the unstable regions become larger as the rock mass structure characterizes in weaker condition. By seeing in Figure 3.15, the rock instability is propagated in the sill pillar under previous mined-out regions, and the optimum sill pillars can be maintained at least 3 m thickness for the model having stronger geological condition. However, in weaker geological condition, GSI 39, the optimum sill pillar is suggested to keep at least 3.5 m thickness to make sure for safety pillar and stope because of a large potential instability in smaller pillar thickness.

In addition, from Figure 3.16, the safety factor index of stope in every mining step that shows the unstable conditions are occurring in 0.5 m above the stope's roof and it will reach a stable condition in 1 m above the stope except the final stope. All these results pointed out that rock mass structure characterized with cracks and joints and meteoric water which result in weathering of host rock will give effects to the instability of underground excavations. Therefore, it should be paid attention for the stability of stope if the geological condition of rock mass is severe condition, not only for safety but also for production losses.

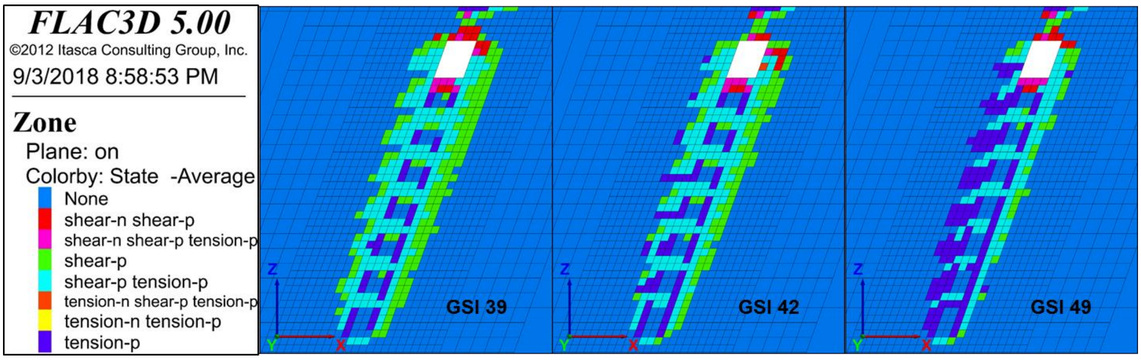

Figure 3.14 Failure zones around the stope with various geological conditions.

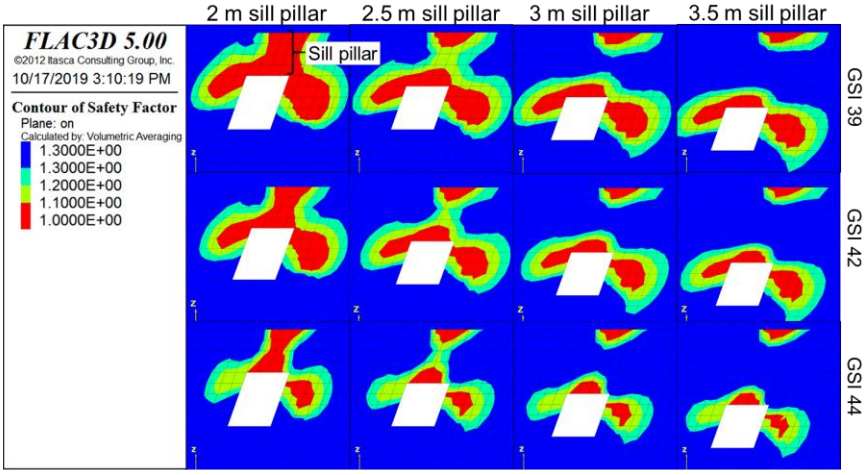

Figure 3.15 Contour of unstable regions around the stope with various geological conditions.

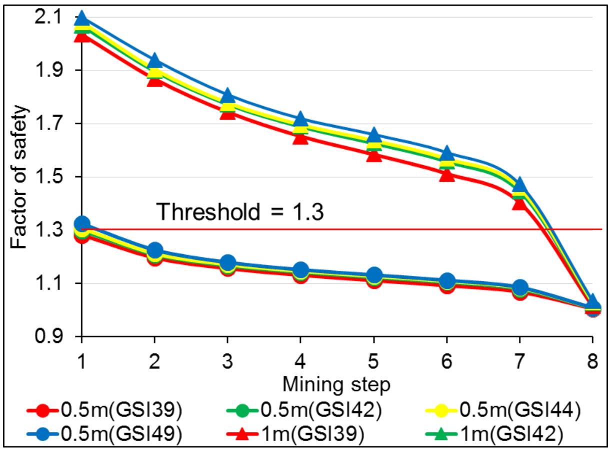

Figure 3.16 Safety factor indicators from stope advancing in various geological conditions.

3.5.5. Stability assessment of stope mining with different stope widths

At Modi Taung gold mine, most veins trend north-northwesterly and each vein system consists of either a single vein, or multiple parallel vein separated by host rocks (Mitchell et al. 2004). The vein systems have many splits, and in most crosscuts consist of two or more narrow veins or veinlets. For the extraction of all veinlets in this mine site, the wider stopes might be needed to increase the ore extraction. For this reason, numerical simulations are conducted with various stope widths (2 m, 3.5 m and 5 m). The simulation results shown by the failure zones and contour of safety factor are given in Figure 3.17 and Figure 3.18. Additionally, Figure 3.19 describes the indicators of safety factor occurred in the bound of stoping sequence.

The wider stope width can develop higher induced stress and instability which result serious consequences on the behavior of the stope. As shown in Figure 3.17, the potential of failure zones are more likely to occur when the stope become wider especially in the roof of the stope. On the contrary, Figure 3.18 shows that the unstable region occurs more severe at the roof of wider stope opening. Because of larger excavation face in wider stope width, the potential of rock instability in sill pillar is quite large. Therefore, the optimum sill pillar should be maintained at least 5 m thickness for 3.5 m stope and 6.5 m thickness for 5 m stope width.

Consequently, the graph of monitoring points collected at 1 m above each stope’s roof indicates that the wider stopes are unstable in all mining steps within 1 m above the stope's roof as shown in Figure 3.19. Thus, these results pointed out that the importance of stope stability when stoping activities carried out at wider vein width and the necessity of an appropriate support system around the wider stope since higher induced stresses are expected to redistribute to the surrounding rock mass.

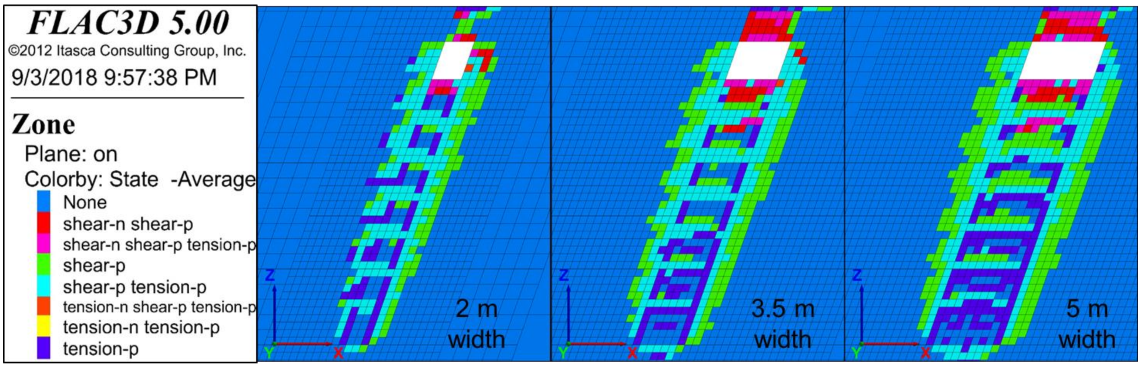

Figure 3.17 Failure zones around the stope due to different stope widths.

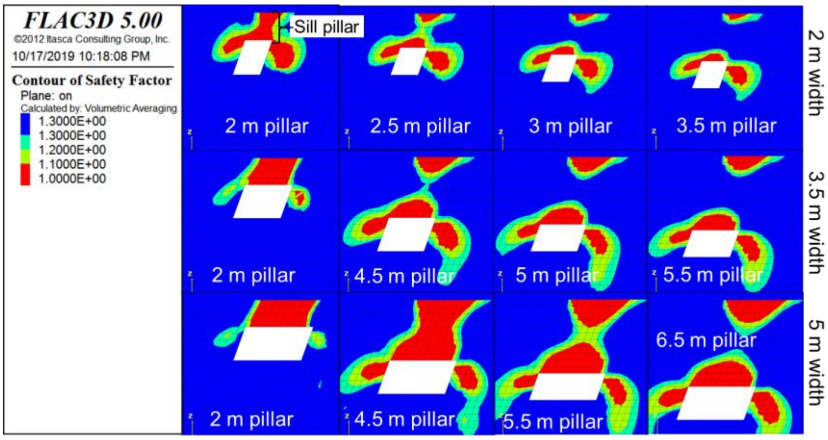

Figure 3.18 Contour of unstable regions due to different stope widths and pillars

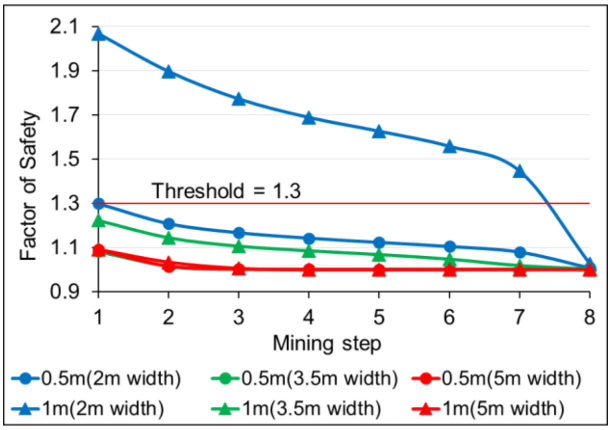

Figure 3.19 Safety factor indicators from stope advancing in different stope widths.

3.5.6. Stability assessment of stope with various vein dips

Another parameter that might influence the stability of stope mining is vein dipping. As the vein dips in Modi Taung gold mine are steep and mostly are larger than 50 degree, hence numerical simulations are carried out in various vein dips with the aim to understand the stope's stability under the overlaying mined-out regions in different vein dips. The simulation models are divided into three different vein dips including 60 degree, 70 degree and 80 degree. Apart from different vein dips, all models are simulated with the same properties and stress ratio as the previous model. The condition of stress flow and the potential displacement of surrounding rock mass is shown in Figure 3.20. Moreover, the condition of failure zones, unstable regions with different sill pillars, and safety factor index in the bound of stoping sequence from the simulations are shown in Figures 3.21, 3.22 and 3.23, respectively.

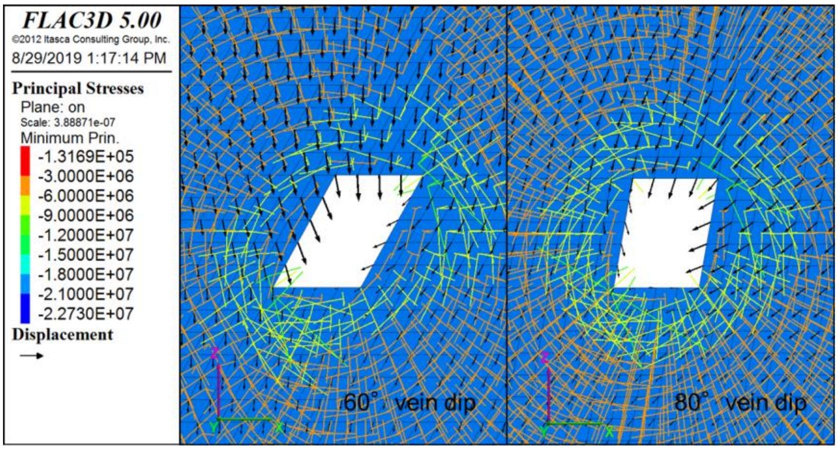

Figure 3.20 Stress flow and displacement from the surrounding rock mass in different vein dip.

According to the condition of stress flow and potential displacement, Figure 3.20 shows that the potential of rock failure might be developed from hanging wall and roof in lower vein dip. Diversely, these failure development come more from foot wall and roof in steeper vein dip. Regarding the failure conditions, Figure 3.21 indicates that the failure zones are more likely to occur with lower vein dip especially in hanging wall and foot wall region/area. On the other hand, the failure zones above the stope opening increase as the vein dip increase.

Figure 3.22 shows the contour of unstable regions around the stope and sill pillar due to various vein dips and Figure 3.23 demonstrates the graph of safety factor index monitored at 0.5 m and 1 m above the stope's roof in every mining steps. It can be seen that the unstable regions are more severe in hanging wall and foot wall with lower vein dip while the instability become more develop above the stope opening with steeper vein dip. By seeing rock instability in the different sill pillars, the results suggested that the optimum sill pillar can be maintained at least 3 m thickness in all vein dipping. From the safety factor indicator graph, the result tells that the stope will reached a stable condition at 1 m distance from the roof except the adjacent stope to the overlaying mined-out area. Thus, all results suggest that special attentions need to be given on hanging wall and foot wall when mining activities conduct at lower vein dip condition.

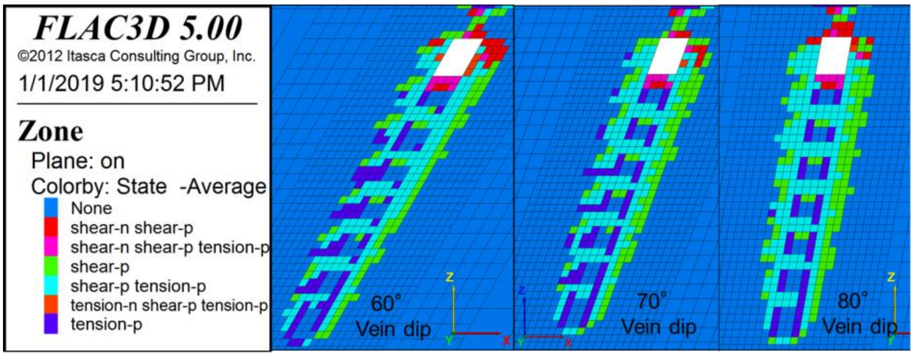

Figure 3.21 Failure zones around the stope due to various vein dips.

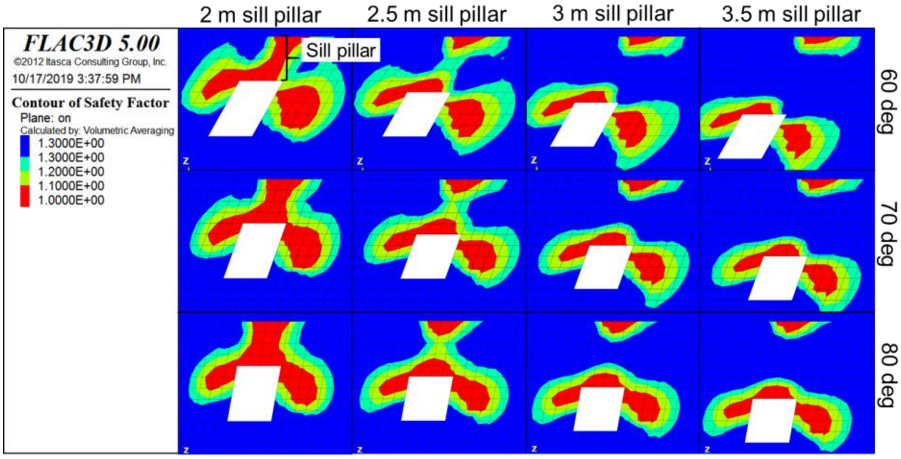

Figure 3.22 Contour of unstable regions around the stope and sill pillar due to various vein dips.

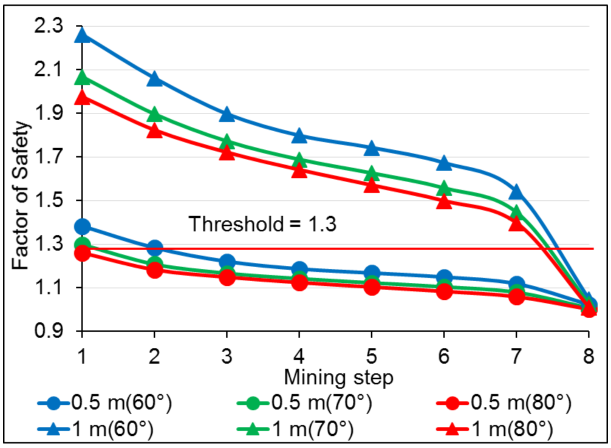

Figure 3.23 Safety factor indicators from stope advancing in various vein dips.

3.5.7. Stability assessment of stope with different stress ratios

In underground mining, the pre-existing stress state changes dramatically due to excavation, therefore load must be redistributed. As mining progresses and more stopes are mined, stresses will be transferred to the remaining unmined stopes. Notably, near-by mining activity has a significant on the state of stress prior to mining a stope. As described in mining condition of Modi Taung gold mine, the accessible shallow area is already mined out at Shwesin vein system and the mining activities plan to continue to the deeper area located under previous mined-out regions. If the rock masses are steadily disturbed by the induced stresses from previous mined-out activities, the intense redistributed stresses can be affected to the stability of rock masses in later mining zones. Therefore, analyzing over stress changing become one of the important issues for the stability of stope opening under previous mining condition in this underground mine. To assess this, three stress ratio (K) conditions are simulated as a preliminary assessment of stope's instability under overlaying mined-out regions. Stress ratio less than one indicate a high vertical stress condition, and a high horizontal stress condition will specify at the stress ratio above one.

It can be observed in Figure 3.24 that the failure zones around the stope opening in higher stress ratio are more propagated than that in lower stress ratio. Figure 3.25 shows more clearly that the strength of rock mass will have more severe condition around the stope opening and sill pillar in high horizontal stress state. Because of unequal stress conditions, the sill pillar thickness will be needed more for the stability of stope and pillar. From the results, the optimum sill pillar should keep at least 4.5 m thickness for higher vertical stress condition, and the pillar should maintain at least 5.5 m thickness for higher horizontal stress condition. In addition, the graph of safety factor index shows that the rock mass maintains satisfactory condition at 1 m above the stope's roof in lower stress ratio except the stopes near the upper mined-out area as shown in Figure 3.26. On the other hand, the stability of rock mass will have severe state since the 4th slice to the uppermost slice at higher stress ratio.

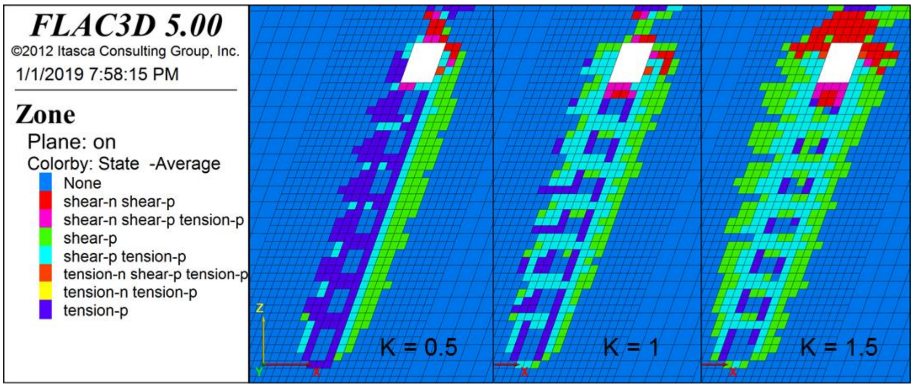

Figure 3.24 Failure zones around the stope due to different stress ratios.

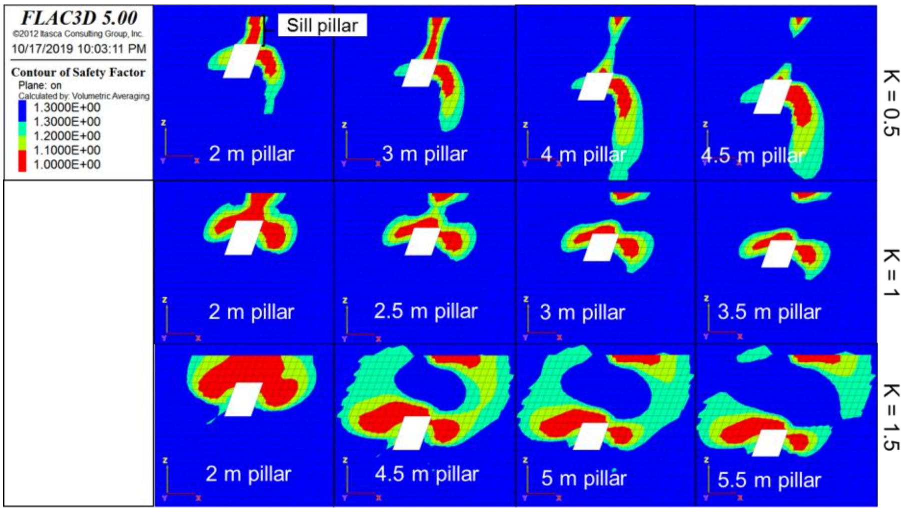

Figure 3.25 Contour of unstable regions around the stope and sill pillar in different stress ratios.

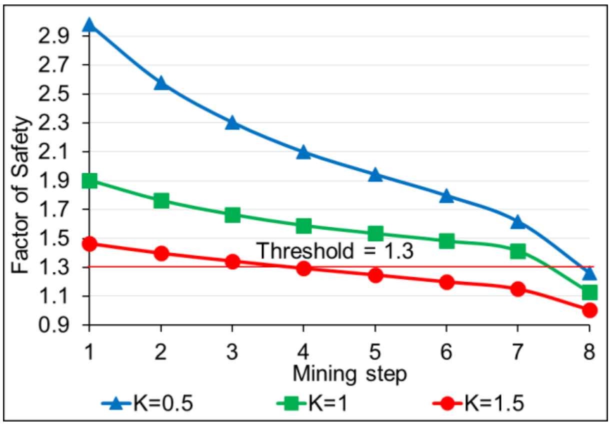

Figure 3.26 Safety factor indicators from the bound of stoping sequence in different stress ratios.

The source of stope's instability upon the variation of stress ratios is due to the effect of stress flow path in the surrounding rock mass as shown in Figure 3.27. Since stress cannot flow through an opening, it must flow around the stope's boundaries. The stope opening causes an induced stress in tangential direction around the stope. Thus, the expected trend have been made to link the flow path of the maximum field stress from the surrounding rock to the induced stress at the stope's boundaries. It can be seen in Figure 3.27 that the flow path of the maximum field stress pass through in line to the induced stress around the stope opening in lower stress ratio. When the stress ratio become larger, the tangential stress around the stope's boundaries are induced perpendicular to the maximum field stress, therefore the potential failures are expected more severe around the stope opening in high stress ratio. Because of these potential failure conditions, efficient countermeasure system is required in the stoping sequence with higher stress ratio.

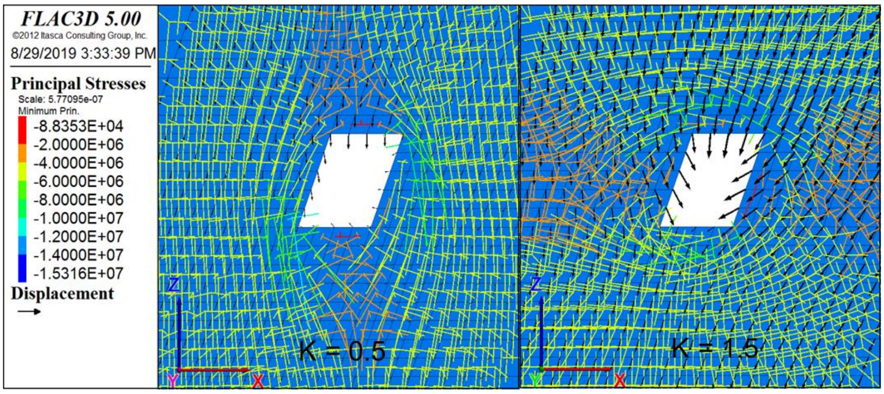

Figure 3.27 Stress flow and displacement for high vertical stress and high horizontal stress.

3.5.8. Stability assessment of stope with different backfilling materials

The placing of backfill underground has predominantly been a practice employed in cut-and-fill mines. Backfilling in stope tends to prevent the deformation of the surrounding rock mass into the mined-out space and to provide a working platform as well. The common types of fill material are waste rock, tailing materials, sand, gravel and some additives. At Modi Taung gold mine, waste rocks from the stope advancing are used to provide a working platform and localized support. In this study, the numerical simulations were conducted under overlaying mined-out area with four different backfilling materials which are waste rock fill, hydraulic fill (mixture of 60% - 80% solid in water), cemented paste fill (CPF) (cement-tailing ratio was 1:8) and cemented rock fill (CRF) (mixture of 5% cement with waste rock) in different properties as shown in Table 3.3 (Gonen 2011) (Purwanto 2015) (Yang et al. 2015) (Karian 2016). Figure 3.28 shows the result of failure zones with different backfilling materials. Figures 3.29 and 3.30 express the conditions of safety factor around the stope and pillar due to various backfilling materials.

Table 3.3 Properties of backfilling materials used in simulations.

	E [MPa]	v [-]	σ_t [MPa]	φ [deg]	C [MPa]
Hydraulic backfill	181	0.2	-	6.9	0.07
Waste rock backfill	153	0.321	-	20.5	0.10
Cemented paste fill (CPF)	1,130	0.16	1.01	48	1.16
Cemented paste fill (CPF)	2,850	0.34	0.7	25.4	1.40

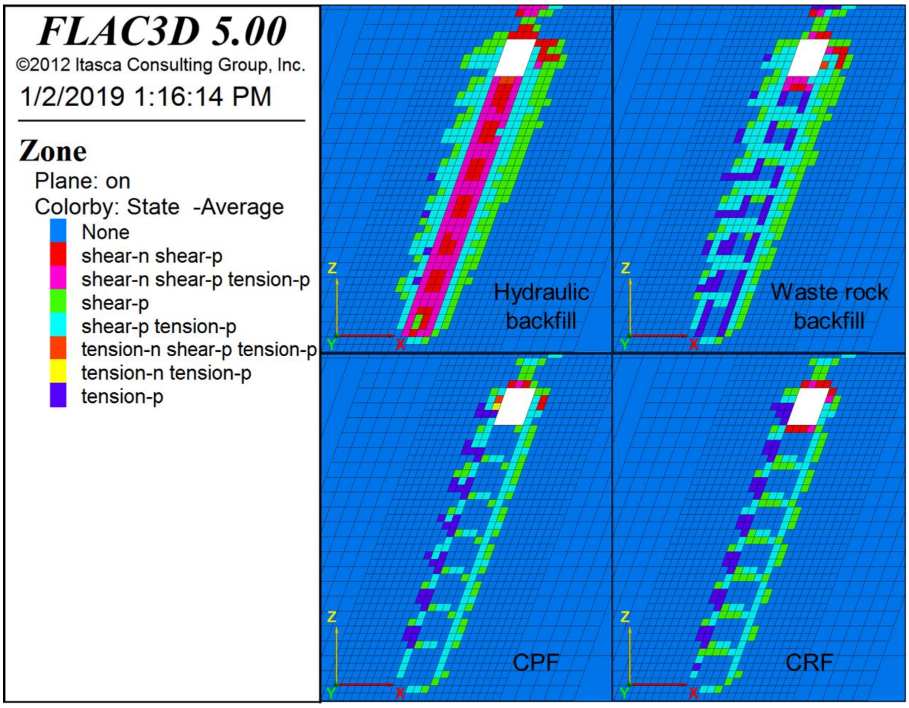

Figure 3.28 Failure zones around the stope with various backfilling materials.

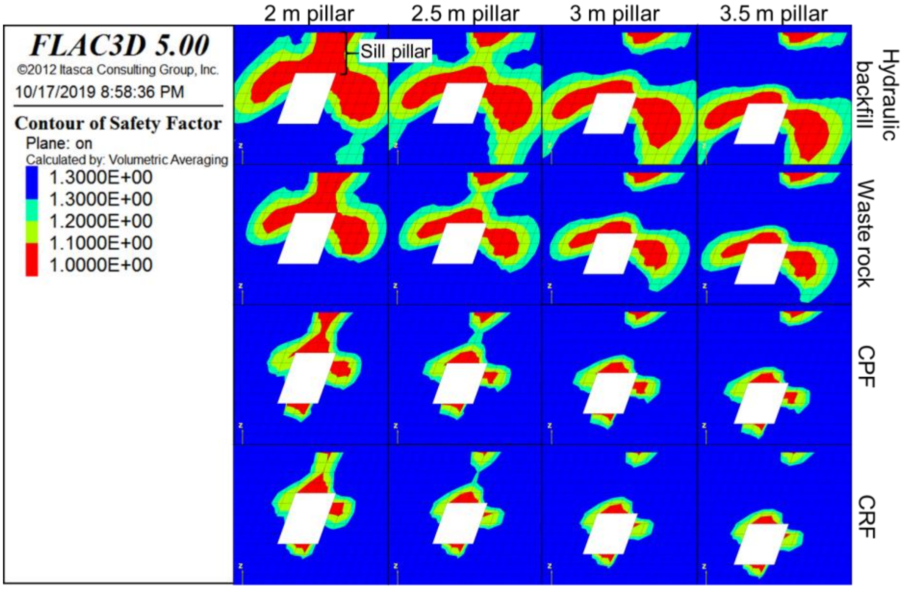

Figure 3.29 Contour of unstable regions around the stope and sill pillar with various backfilling materials.

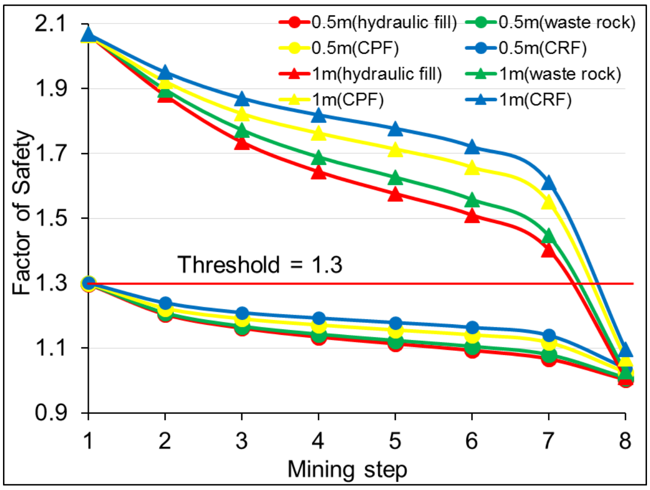

Figure 3.30 Safety factor indicators with various backfilling materials.

According to the results from Figure 3.28, it is noted that the failure zones around the stope decrease as the strength/competence of backfilling materials increase. However, the potential of failure cannot clear for the stope stability even though the backfilling materials have stronger properties. It can be obviously found in the contour of safety factor as shown in Figure 3.29 that the unstable zones still appear in the vicinity of the stope despite the open stopes are loaded with stronger backfilling materials. Moreover, the results showed that the optimum sill pillar can maintain at least 3 m thickness in all models except the model result with hydraulic backfilling. The sill pillar in hydraulic backfilling model is suggested to keep 3.5 m thickness by expecting a large potential instability occurred in 3 m pillar thickness.

In addition, Figure 3.30 indicates that the stopes will be unstable within 0.5 m above the roof and it will have a stable condition at 1 m above except the stopes near the upper mined-out area. According to these results, the deformation of the surrounding rock mass to the mined-out space can reduce depending on the properties of different backfilling materials, however, the unstable zones still occur. Therefore, it can be concluded that the stability of stope opening and sill pillar cannot rely completely on the properties of filling material in the bound of stoping sequence. Hence, the selection of backfilling material should be based on available backfill materials and economic factor only instead of stability reason.

3.6. Discussions

The accumulation of induced stresses from the stope opening and the redistributed stresses from previous mined-out regions are the main cause for the instability of stope advancements. All simulation results indicated that the failure zone and instability of the surrounding rock mass of stope increase steadily in overhand cut-and-fill mining as the stope progressing moves toward upper slices and it propagates to the previous mined-out regions. Thus, the sill pillar between the final stope and upper mined-out regions should be maintained the optimum thickness to stabilize the stope mining, and to prevent rock falls from the stope's roof and walls.

In the case for the deterioration of backfilling waste rock in the previous mined-out regions, the excavation should be paid attention at the final stope because the influence of the stress redistribution from the previous mined-out regions might be larger, and it can be affected to the current mining process. As mentioned in Section 1.3 of Chapter 1, most of the underground metal mines in Myanmar are already developed in shallow area so far, accordingly the mines are advancing towards the deeper level. Therefore, the effects of upper mined-out regions might be influenced more to the stability of advancing excavations because the strength and deformability of backfilling materials might be getting worse from time to time. In addition, in the case of no backfill condition during mining activities in previous mined-out regions, special attentions need to be paid for the stability of sill pillar and stope mining since the possibility of rock failures and redistributed stresses from upper mined-out regions are expected to have more severe conditions, consequently the appropriate countermeasure systems are suggested to be prepared for this conditions.

Moreover, the potential of instability is likely to occur in the lower vein dips, more severe geological condition, wider vein width and higher stress ratio. Furthermore, the backfilling materials cannot clear the instability in the vicinity of the stope. Therefore, the selection of backfilling material should be based on available backfilling materials and economic factor only instead of stability reason. According to all simulation results, suitable countermeasure arrangements are paramount to be prepared by considering the stability of stope mining activities under the influence of overlaying mined-out regions.

3.7. Conclusions

The mining activities of Modi Taung gold mine continue to progress towards deeper levels to fulfill the target ore production according to their mining plan. Therefore, the potential failures from the overlaying mined-out regions to the new stope opening should be investigated to stabilize the stope and sill pillar in various mine conditions because risk-indexes, such as stress redistribution from the previous mined-out area, rock fracture and weathering condition, etc. can be subjected to the current stope mining activities.

From this chapter, it can be concluded that the potential of displacement and rock mass failure to the working stope under the previous mined-out activities are developed not only by its own induced stresses but also the influence of the redistributed stresses from previous mined-out regions. Moreover, it need to be taken account the influence of the deterioration of backfilling materials from the upper mined-out regions. From time to time, the strength and deformability of backfilling materials might be getting worse, accordingly it can be affected to the stability of underground mining under previous mined-out regions. According to all simulation results from various mine conditions, suitable countermeasure arrangements are paramount to be prepared to the stope mining especially for the conditions of higher stress ratio and wider stope widths by considering the stability of working stope under the influence of previous mined-out regions.

CHAPTER 4

RISK ASSESSMENTS FOR THE STABILITY OF SLOPE SURFACE DUE TO STOPE MINING

4.1. Background

At Shwesin vein system of Modi Taung gold mine, there are many mountain slopes along the occurrence of ore mineralization, and hence underground mining have to operate under the slope surface (see Figure 4.1). In general, the occurrence of underground mine's instability comes from the stress redistribution of previous mined-out activities, the existence of discontinuities and void space, the influence of on-going mining activities, and opening geometry, etc. Moreover, if the underground mining is conducted near the mountain slope surface, the influence of slope surface should be taken into account to that mining activities. McTigue and Mei reported that the terrains had an obvious impact on the initial in situ stress field, even though the slope gradient was only 10% (McTigue and Mei 1987) (Li et al. 2017).

In the mountain slope, the in-situ stress values near the slope surface are generally considered to be quite small as the overburden depths are relatively shallow. However, on the horizontal plane of the same elevation, the maximum principal stresses increase at first and then decrease with the increment of the horizontal coordinate. In other words, the in-situ stress of the area within a certain distance to the slope surface exhibits a higher value owing to the mountain slope effect, especially within 100 m from the slope surface (Li et al. 2017).

If in-situ stresses are not equal from all directions under slope surface, then the differential stress will be appeared and affected to the surrounding rock mass. As well known, differential stress is the difference between the major and minor principal stresses, and rock mass can deform (break/flow) due to differential stress. Accordingly, it is necessary to make a full understanding of the distribution principles of initial in-situ stress field before selections of any engineering works and underground excavations in mountain slope areas. Two forms of instability are readily observed around underground openings: (1) structurally controlled gravity-driven processes (2) stress-induced failure or yielding (Martin, Kaiser, and Christiansson 2003). In many pieces of literatures, some instability indicators are usually defined in terms of failure zones, stress condition, displacement and extent of yield zones (W. R. Abdellah, Ali, and Yang 2018) (Karian 2016) (Purwanto et al. 2013). In this study, the evaluation on rock mass condition and mining under the sloping surface are described by the occurrence of natural and mining-induced differential stress and failure zones affected to mining activities. In this chapter, a series of numerical simulations are performed to analyze three different conditions including the evaluation on the strength of rock mass in the sloping surface, assessment on the stability of stope mining under the slope surface, and instability of stope in the near-ground region in various mine conditions.

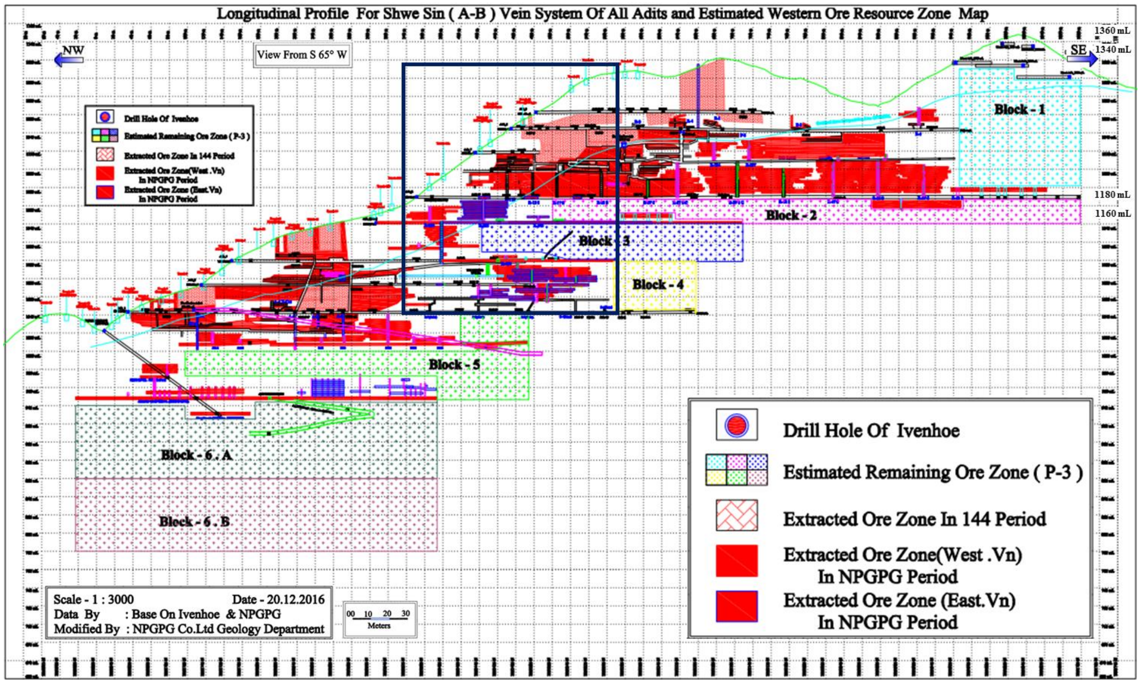

Figure 4.1 Topography and mining plan at Shwesin vein system (source: NPGPGL).

4.2. Numerical modelling for simulations

The evaluation on the rock mass condition before mining and probability of failure during mining under the slope surface in shallow depth is investigated by using FLAC 3D software. In this study, the size of the basic numerical model is 260 m × 260 m × 320 m with vein dip of 70 degree as shown in Figure 4.2. Except the model shape, rock mass properties, stope geometry, boundary conditions and failure criterion of the model are the same as previous Chapter 2.

The contents of numerical analyses consist of three parts in this chapter. Firstly, in order to understand the strength of rock mass near the slope surface, the simulation for slope stability is carried out with the basic numerical model as shown in Figure 4.2. Moreover, in order to analyze in-situ stress distributions under the slope surface, various monitoring points of differential stress (the difference between major and minor in plane principal stresses) subjected to the rock mass are recorded as shown in Figure 4.3. Secondly, in order to investigate the stability of stope mining under the slope surface, the simulations are carried out in the bound of stoping sequence. From this simulations, in order to evaluate the effect of induced differential stress to the stability of stope under the slope surface, the monitoring points for differential stresses and the occurrence of failure zones are recorded along the excavation length (middle of stope length and the two edges) as shown in Figure 4.4 and Figure 4.5. When the stope mining is operated in the near-ground part under slope surface, it results in higher differential stress and higher risk of stope failure due to the effects of mountain slope. Because of varying depths of overburden, mining activities along the sloping surface may experience different instabilities of the rock mass. Finally, the instabilities of stope in the near-ground part under the sloping surface are examined in various mine condition including the variation of rock mass properties and different stress ratio. Uncertainties on the estimation of rock mass properties and in-situ stress distribution have a significant impact on the stability of underground excavations and slope surface.

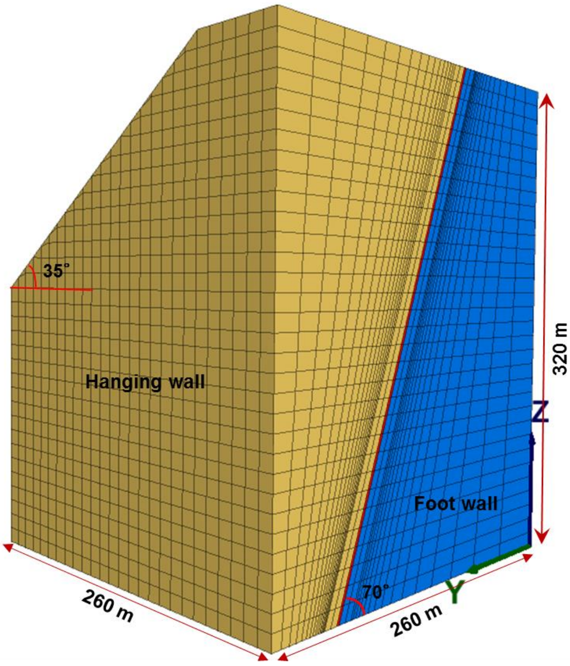

Figure 4.2 Basic model to analyze the effects of slope surface.

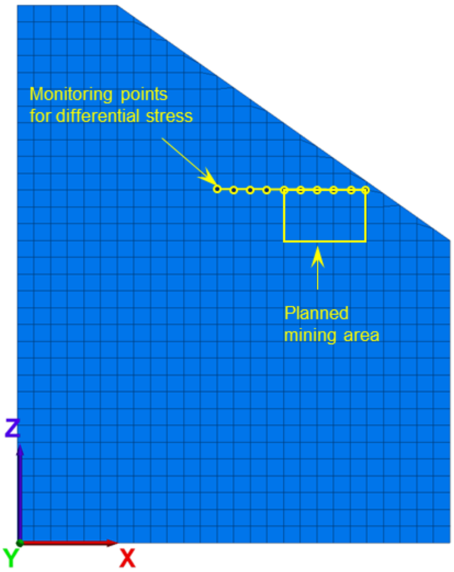

Figure 4.3 Monitoring points for differential stress under slope.

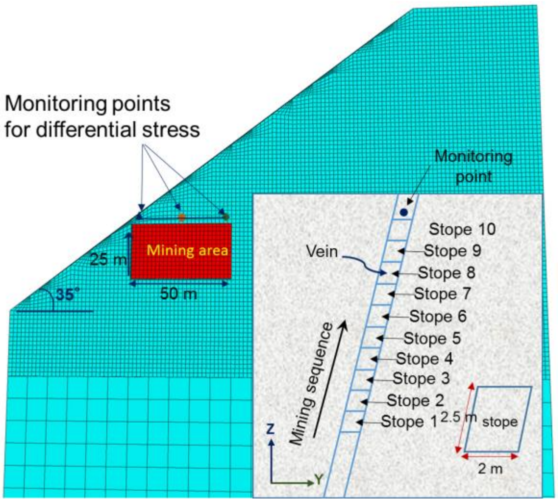

Figure 4.4 Stope mining and stoping sequence under slope surface.

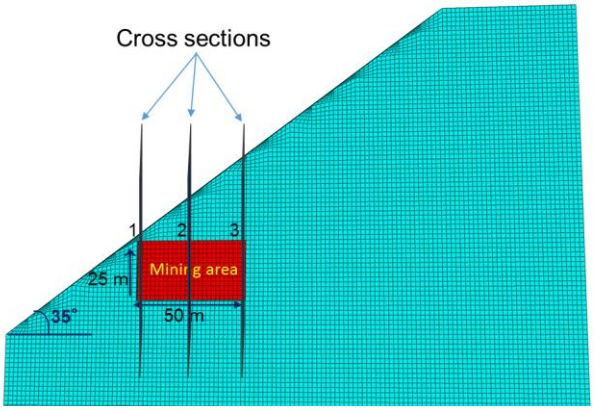

Figure 4.5 Monitoring planes recorded for failure zone in this study.

4.3. Results

4.3.1. Slope stability and strength of rock mass under the slope surface

In geotechnical field, stability analyses aim to support the safe and design of rock and slopes. A slope analysis is carried out by using an analytical or numerical model which can determine the movement of a potential unstable mass. In this research, firstly, slope stability analysis is carried out for the purpose of assessing slope mass strength by means of numerical simulations with different slope angles which can be influenced by in situ stress distribution. Figure 4.6 shows the contour of maximum shear strain increment with the description of factor of safety (FoS), and Figure 4.7 indicates the graph of safety factor index with different slope angles. Based on the analyzed results, the factor of safety is found as 3.45 in 35 degree slope, 2.88 in 45 degree slope, and 2.47 in 55 degree slope. According to the safety factor description in Section 3.4 of Chapter3, it can be said that the mountain slope of Modi Taung gold mine is in a stable condition before the development of the underground excavation. However, attentions should be paid when the underground mining is carried out under steeper slope condition since the stability of slope is decreasing as the slope degree is increased.

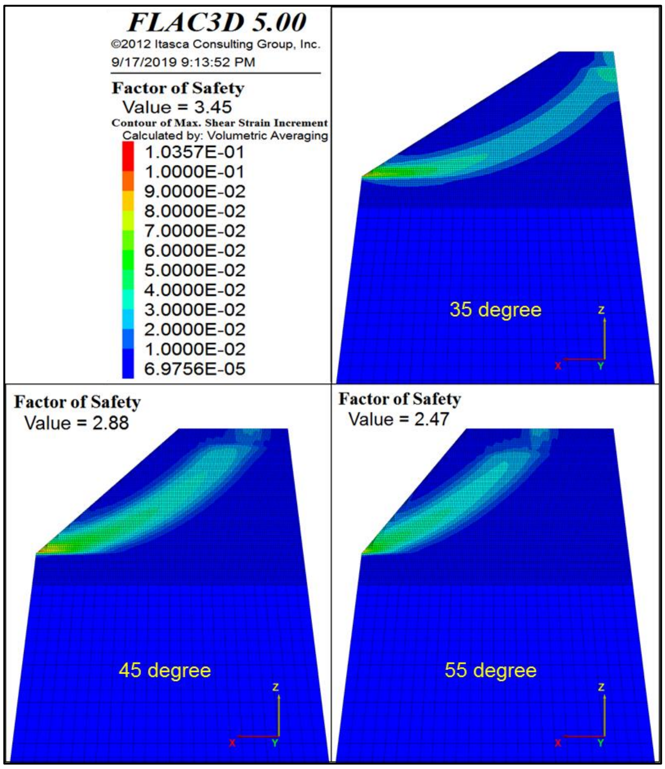

Figure 4.6 Condition of slope stability with different slope angles.

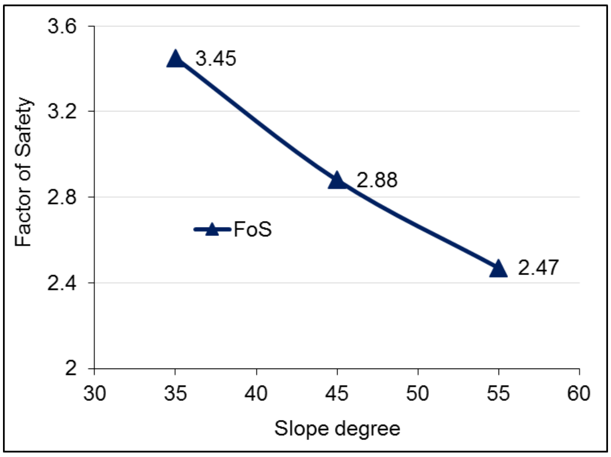

Figure 4.7 Safety factor index with different slope angles.

In addition, the distribution of unequal differential stress have an obvious impact on the stability of surrounding rock mass near the slope surface. Because of unequal differential stresses acting on surrounding rock mass, the instability arises and failure can be developed. In order to validate the in-situ stress distribution principle near the slope surface, evaluation on the condition of rock mass under slope surface are conducted before any excavations. By seeing the strength of rock mass to the stress applied at the mountain slope as shown in Figure 4.8, even though the most unstable part of the rock mass is experienced near the slope foot, the potential of instabilities are gradually propagated to the inner part of rock mass from the slope surface. The result proved that the rock mass closed to the slope surface can be more subjected by the influence of in-situ stress distribution than inner part of rock mass.

This result is demonstrated in Figure 4.9 that shows the illustration of differential stresses measured in sloping surface. The distribution characteristics of differential stress have been investigated on the same horizontal level from the slope surface. It can be seen that the differential stresses are progressively increased from the inner part of the rock mass as the monitoring points move towards the slope surface. The increment value of differential stress is much greater than the predicted value according to the vertical overburden depth especially within 13 m from the slope surface. That trend indicates that the stability of rock mass near the slope surface can meet more severe than that of the places far from the slope surface due to the strong influence of in-situ differential stress. Therefore, it should be noted that the mining activities near the slope should be paid attention to avoid unexpected rock mass failure.

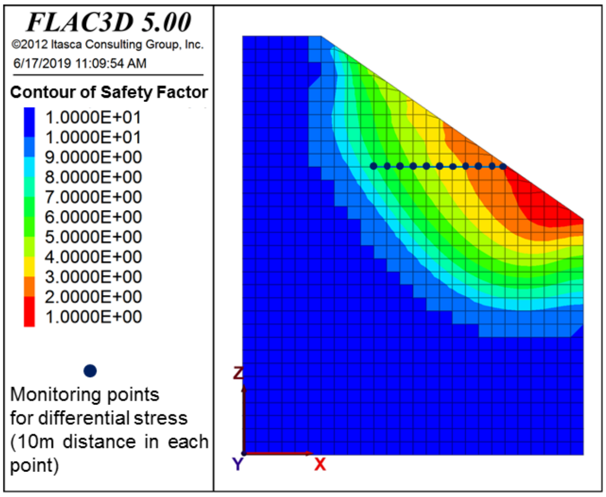

Figure 4.8 Potential of rock mass instabilities at slope surface.

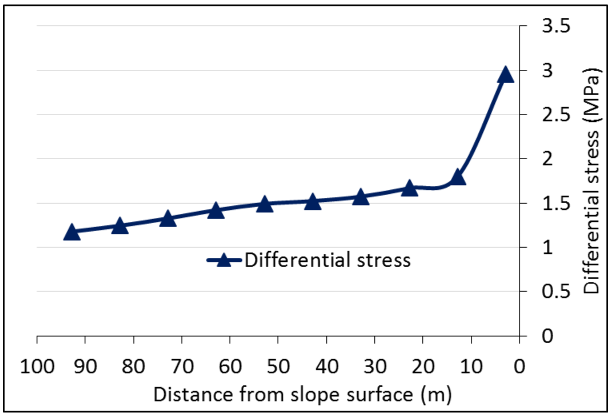

Figure 4.9 Differential stress measured under slope surface.

4.3.2. Assessment on the stability of stope mining near the slope surface

Secondly, the condition of stope mining under the slope surface and stoping sequence is conducted as shown in previous Figure 4.4. When the stope mining is operated in the near-ground part under slope surface, the variation of stress and failure of the stope is affected more than the deeper part. Because of varying depths of overburden, mining activities along the sloping surface may experience different instabilities of rock mass based on the distance from the slope surface. Thus, three monitoring points at the same horizontal elevation for induced differential stresses were recorded 3.5 m, 26.5 m, and 49.5 m distances from slope surface in order to evaluate the effect of induced differential stress to the stability of stope under the slope surface.

The results for the occurrence of differential stress in the bound of stoping sequence are shown in Figure 4.10. All of these monitoring points are placed at 1 m above the final stope during stope mining. According to the result, when the mining sequence is advancing towards the slope surface, the induced differential stress is gradually increased due to the accumulation and redistribution of stresses from the lower excavations. Moreover, it can be seen that the highest differential stresses are developed in the nearest part to slope surface compared to the other two places. That trend indicates that the instability of rock mass occurred in the nearest place to slope surface is more severe than that of the rock mass far from the slope surface because of not only from the stress accumulation of lower stope mining but also the influence of higher differential stress at the slope surface.

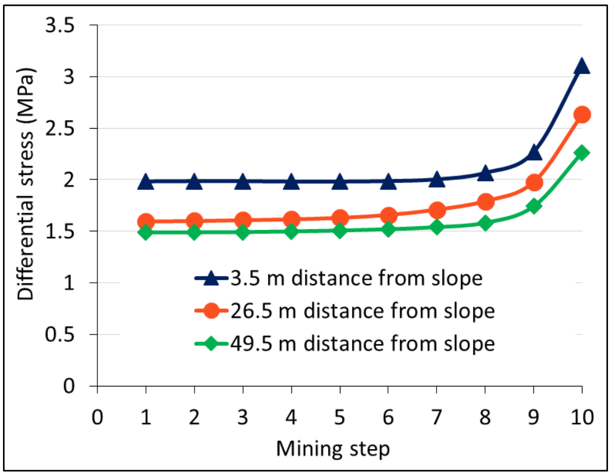

Figure 4.10 Differential stresses measured in the bound of stoping sequence under the slope surface.

This statement can be proved in Figure 4.11 that shows the propagation of failure zones occurred in the bound of stoping sequence under the slope surface. These results are also collected at the horizontal level of 3.5 m, 26.5 m and 49.5 m distances from the slope surface respectively as described in Figure 4.5. According to results, the distribution of failure zones is more propagated to the excavation near the slope surface. On the contrary, the mining activities far from the slope are likely affected by its own induced stress, therefore the failure zone only occurs around the stope mining without propagating to the surrounding rock mass. By seeing induced differential stress and failure zone condition as shown in Figures 4.10 and 4.11, if the mining activities become closer to slope surface, the potential of stope failure become higher due to the strong influence of increasing differential stresses. Thus, large instabilities of rock mass near the slope surface are expected, accordingly the stability of stope mining need to be monitored in the stope cavity especially at the nearest part to the slope surface. This explanation can be clearly seen in Figure 4.12 which demonstrates the propagation of unstable region around the stope under slope surface. The results suggest that the stope instability are more likely to occur at the stope near to the slope surface. The rock instabilities are more propagating at the stope near the slope than the place far from the slope surface. Therefore, according to this result, it can be proved that the rock mass near the slope surface can be more affected by differential stress than that of inner part, and this result highlights the importance of rock stability when the underground mining activities operate near the slope surface.

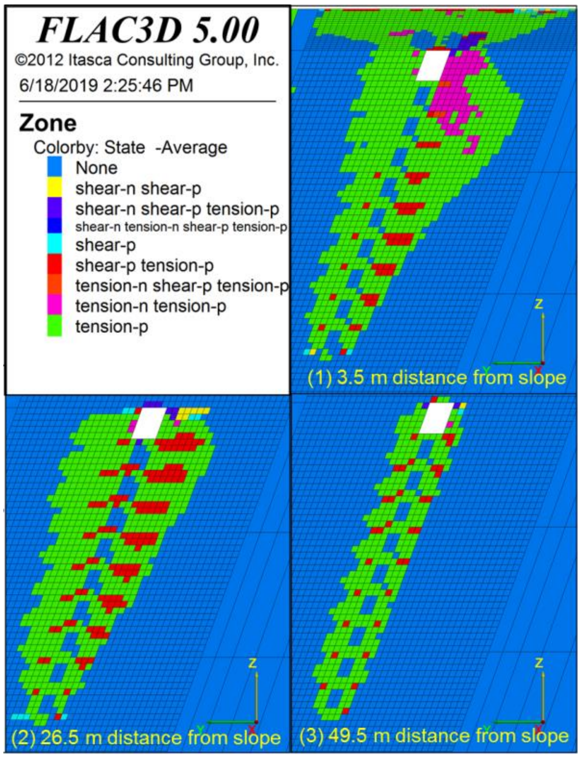

Figure 4.11 Failure zones occurred in the bound of stoping sequence in different places from slope surface.

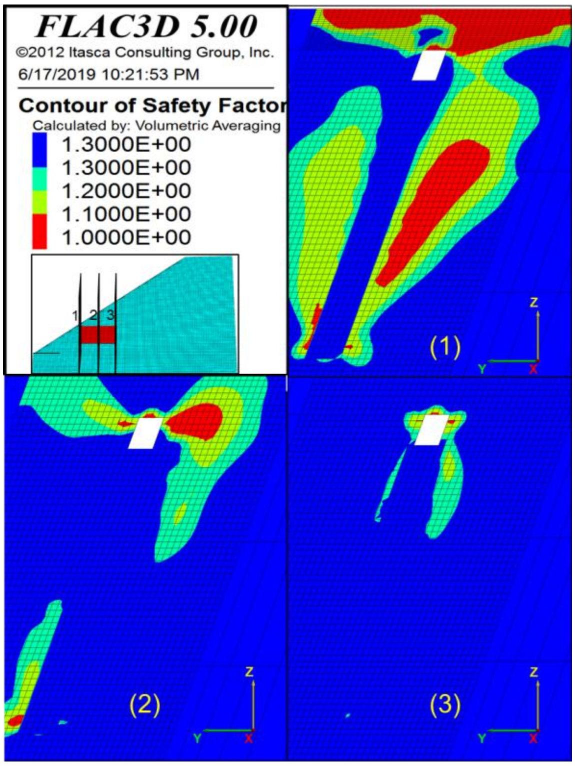

Figure 4.12 Rock instabilities occurred in the bound of stoping sequence in different places from slope surface.

4.3.3. Parametric study on the stability of stope under slope surface in various mine conditions

Parametric study in this session is carried out to understand the influence of slope surface to the stope mining in various mine conditions. The strength of rock mass can primarily differ from the variation of rock mass properties and the stress condition acting on that rock mass. Hence, the investigations on the stability of stope mining in the near-ground part under the slope surface are conducted on the variation of different stress ratios and various rock mass properties. As described in previous section, the most unstable region of stope cavity is the nearest part to the slope surface, accordingly the results from that part are recorded and discussions are carried out for various mine conditions.

4.3.3.1 Parametric study on the influence of different stress ratios

Firstly, investigations are carried out to understand the instability of stope mining under the mountain slope surface due to the influence of different horizontal to vertical stress ratio. In the mountain slope, the horizontal tectonic stresses are always larger than the vertical in situ stress (Li et al. 2017). Therefore, it is necessary to make a full understanding of the effect of stress distribution to the stope mining under the slope surface. To assess this, the model with three stress ratios are simulated as a preliminary assessment of stope's instability under the sloping surface.

Figure 4.13 shows the condition of differential stress acting on rock mass in the bound of stoping sequence with different stress ratio under the slope surface, Figure 4.14 demonstrates the occurrence stress flow around the stope excavation, and Figure 4.15 presents the distribution of failure zones which is located at the nearest place to the slope surface in different stress ratio. In Figure 4.13, when the stope is opened in higher horizontal stress condition, the differential stresses acting to rock mass are obviously larger than that of lower stress ratio. One of the reason is the cause of stress flow which passes through around the excavation as described in Figure 4.14. As the stress ratio becomes higher, the trend of gravitational field stress goes perpendicular to the induced tangential stress, therefore it can be expected the rock mass around the excavation might be more disturbed and the potential of rock deformation might be large in stope opening. This condition is proved in Figure 4.15 which shows the model with higher horizontal stress ratio meets more severe condition with shear and tensile failures than the one with higher vertical stress ratio.

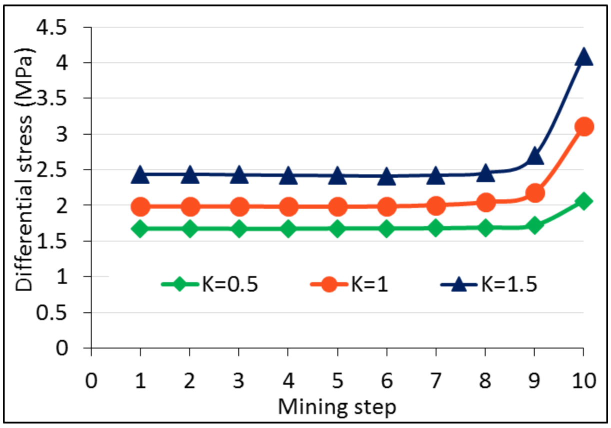

Figure 4.13 Differential stresses measured in the bound of stoping sequence with various stress ratios.

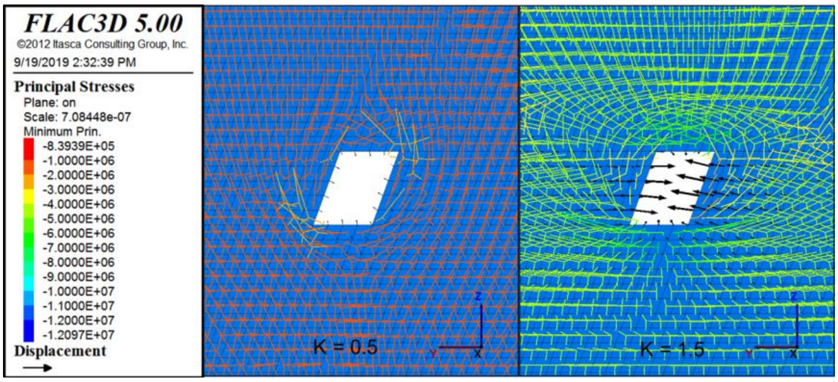

Figure 4.14 Stress flow around the stope in different stress ratio under slope surface.

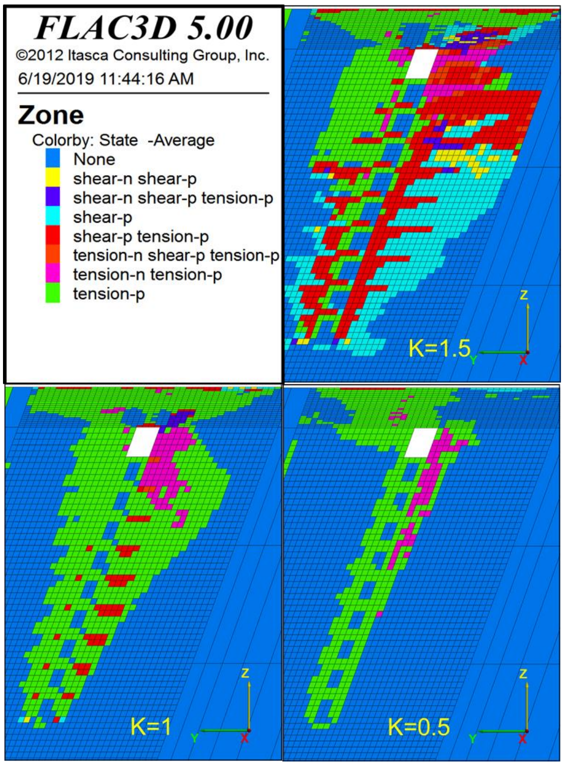

Figure 4.15 Failure zones occurred in in the bound of stoping sequence with different stress condition.

4.3.3.2 Parametric study on the influence of different rock mass properties

Apart from the stress distribution, another parameter that might influence to the stability of underground excavation is the variation of rock mass properties. In this study, numerical simulations are conducted with the different rock mass properties described in Table 3.2 from previous chapter. The results in Figure 4.16 show that the model with higher GSI value, represents better geological condition, has a lower differential stress under the slope surface, and higher differential stress had developed in lower GSI value.

By seeing these results, it can be expected that the rock mass condition near the slope surface can be more affected in lower GSI, and the potential of stope failure might be experienced more in lower GSI than that of higher GSI. The result of this assumption can be found in Figure 4.17 that demonstrated the occurrence of failure zones in the different geological condition under the slope surface. Even though both of the tensile and shear failures occur in all rock mass properties, more severe conditions are experienced as the geological condition become worse.

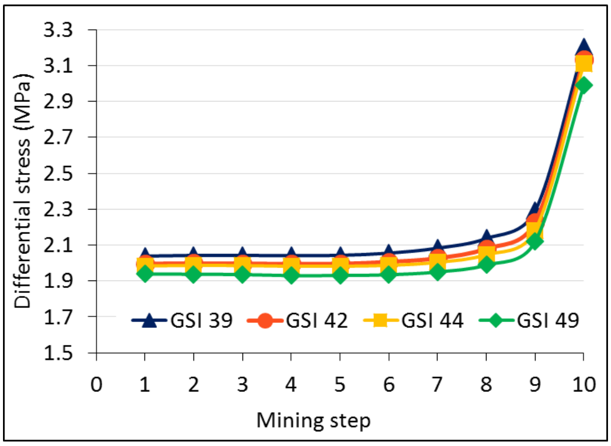

Figure 4.16 Differential stresses measured in the bound of stoping sequence with different geological condition.

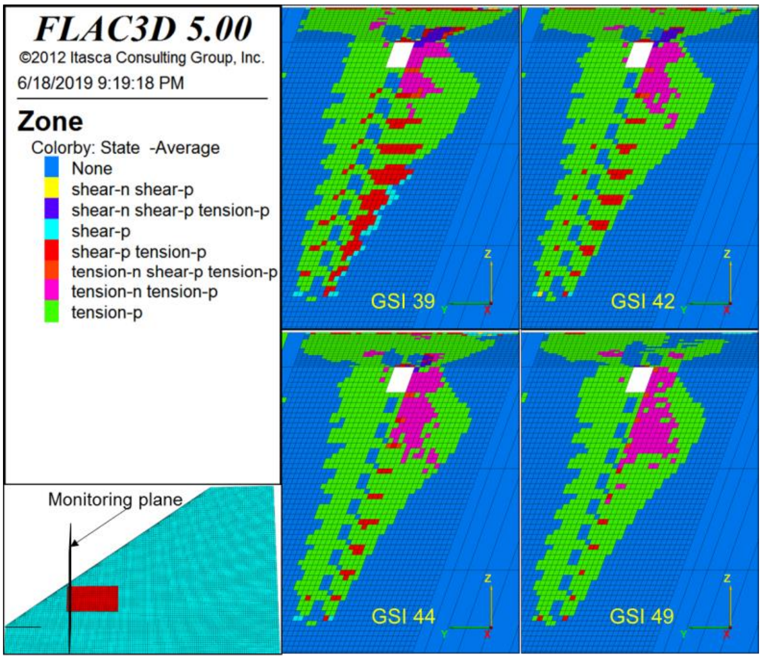

Figure 4.17 Occurrence of failure zones in the different geological condition under the slope surface.

4.4. Discussions

In this chapter, the study on the stability of rock mass in mountain slope before the development of underground excavations, and the potential rock failures of stope mining under the slope surface are carried out. Based on the investigations of rock mass condition near the slope surface, the results indicate that the stability of rock mass in mountain slope is decreasing as the mountain slope is steep, consequently attention should be paid when the underground mining is carried out under steeper mountain slope surface. Moreover, the differential stresses are progressively increased from the inner part of the rock mass towards the slope surface. This results point out that the rock mass closed to the slope surface can be more subjected by the influence of in situ stress distribution than the inner part of rock mass, accordingly the rock mass strength can gradually decreased towards the slope surface.

When the mining sequence is developed under the slope surface, the occurrence of failure zones is more propagated to the excavation near at the slope surface. On the contrary, the mining activities far from the slope are likely affected by its own induced stress, therefore the failure zones only occur around the stope mining without propagating to the surrounding rock mass. From this results, it can be pointed out that the rock mass in the nearest place to slope surface is more affected by not only from the stress accumulations of lower stope mining but also the influence of higher differential stresses near the slope surface.

Form the simulation with different stress ratio, it can be expected the rock mass around the excavation might be more disturbed by induced differential stresses, accordingly the possibility of failure zones might be large in higher stress ratio. Therefore, more supporting capacity is needed to stabilize the stope mining near the slope surface in higher stress ratio condition. In addition, from the simulation with different rock mass properties, the results show that both tensile and shear failures occur more severe as the geological condition become worse. Therefore, mining in poor ground should be paid more attention to avoid stress-induced rock mass failure.

4.5. Conclusions

There are many mountainous slopes along the occurrence of ore mineralization in remote area, hence underground mining have to operate under the slope surface. Mining activities under slope topography may affect more on the variation of stress and failure zones than the other places of rock mass due to the risks of the sloping condition.

From this chapter, it can be said that the strength and deformability of rock mass is gradually decreased towards the slope surface as the differential stresses are progressively increased from the inner part of the rock mass towards the mountain slope surface. Additionally, when the mining sequence is developed under the slope surface, the rock mass in the nearest place to slope surface is more affected by not only from the stress accumulation of lower stope mining but also the influence of higher differential stress near the slope surface. As the rock instability is more severe and propagated up to the slope surface, special attention needs to be given in the bound of stoping sequence under the mountain slope.

CHAPTER 5

COUNTER MEASURES FOR THE STABILITY OF STOPE OPENINGS

5.1. Background

The potential for instability in the rock surrounding underground mine openings is an ever-present threat to both the safety of men and equipment in the mine (E. Hoek, Kaiser, and Bawden 1995). Hence, optimization on the stability of underground mine opening becomes the most important issue to avoid injuries and fatal accidents during the life of mine. For improvement of underground mine's stability, rock support is widely used to improve the stability and maintain the load bearing capacity of rock near the boundaries of an underground excavation. Before support to the underground opening, it is required to decide whether the support system is temporary or permanent type. Temporary support is that support or reinforcement installed to ensure safe working conditions during mining. On the other hand, if the excavation was required to remain open for an extended period of time, permanent support was installed subsequently.

The selection of the type of support installed in a particular underground excavation depends upon the extent of the zone of loosened or fractured rock surrounding that excavation. Rock support is mainly categorized into active and passive type (Evert Hoek and Wood 1987). Active support forces a load onto the rock surface in order to support broken rock mass and ensure its stability. Passive support on the other hand is reactive to the ground's movements.

Active support system is usually required when it is necessary to support the gravity loads imposed by individual rock blocks or by a loosened zone of rock (Brady and Brown 2004). This system forces a load onto the rock surface in order to support broken rock mass and ensure its stability (Queen's University 2011). Mechanically anchored rock bolts, grouted or friction anchored dowels, grouted cables are common rock support which are categorized as an active type rock support. Each type of active type rock support offers different practical advantages and disadvantages along with suitability for a different rock mass and stress condition. From economical point of view, an active type rock support is preferred to the passive type support system due to its lower cost (Karian 2016).

On the other hand, if the rock mass surrounding underground openings seems to be poor, passive type rock support with higher supporting capacity is needed. The passive type rock support has a different working principal with the active one. Its system is external to the rock and responds to inward movement of the rock surrounding the excavation (Evert Hoek and Wood 1987). Rock support with a passive support often includes the use of mesh, straps, shotcrete, timbered set, and steel sets. Being more expensive than the active ones, a passive type rock support has very limited application such as in very loose ground. Another reason to apply a passive type rock support is high stress condition occurred around the opening (Karian 2016). In high stress condition, large displacement may occurs at boundary of opening and the rock around stope will continue to move even after supported by active type rock support.

5.2. Countermeasure systems for stope instability

The countermeasure support systems are conducted for stope instability in two parts; firstly, for the stope instability due to the effects of previous mined-out regions as described in Chapter 3, and secondly, for the stope instability due to the influence of mountain slope surface as shown in Chapter 4. In order to understand the effectiveness of rock support for stope failure in the above mentioned Chapters 3 and 4, a series of parametric studies are carried out with different geological condition, stress ratio, vein dip and stope width.

In this research, two types of countermeasure systems are applied to the stope for optimizing the stability of its opening. Firstly, active type support, grouted cables are applied to the simulations of stope openings. The main advantage of these cables is that they can be installed in mine openings with very low headroom (Evert Hoek and Wood 1987). Regarding the cable length, Potvin et al. (1989) suggested that the length of the cable bolt should be approximately equal to the span of the opening (Potvin, Y., Hudyma, M.R, Miller, H.D.S 1989) (E. Hoek, Kaiser, and Bawden 1995). In addition, the cable bolt spacings need to decide depending upon joint spacing, joint orientation, and overall ground conditions (E. Hoek, Kaiser, and Bawden 1995). In this research, 3 m length of grouted cables with 1 m x 0.5 m spacing are used as an active support to the stope openings, and its properties are as shown in table 5.1. As the stopes in Modi Taung gold mine are narrow and low heights, the use of grouted cables can effective in applications such as the reinforcement of ore, host rock and waste passes.

Secondly, as a passive type countermeasure system, shotcrete support is considered to make sure the stability of stope in severe condition due to its effective support capacity. Shotcrete has found increasing use in underground mining practice, initially for the support in permanent excavations but now increasingly for the support of stopes and stope accesses (Brown 1999). As described in the practical rock engineering book of Evert Hoek, highly jointed metamorphic or igneous rock mass can apply 50 mm thickness of reinforced shotcrete to rock surface for preventing rock mass failure (Evert Hoek 2000). In this simulations, the shotcrete with 50 mm thickness is supported for passive countermeasure system with the properties as shown in Table 5.2. In this research, active type support will be firstly installed to the stope opening, and if the unstable condition is found, the stope will be modified by passive type because of its higher supporting capacity.

Table 5.1 The properties of cable bolt used in analysis (Karian 2016).

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
Pretension force (kN)	100

Table 5.2 Properties of shotcrete (Karian 2016).

Shotcrete Properties	
Young’s modulus (GPa)	21
Poisson’s ratio	0.15
Compressive strength (MPa)	35
Tensile yield (kN)	20
Residual yield (kN)	10

5.3. Optimization of stope stability under previous mined-out regions

5.3.1. Optimize the stope stability in different geological conditions

Firstly, numerical investigations are carried out in order to understand the instability of stope due to the influence of different geological conditions and how to optimize the stability. The basic numerical model used in the Chapter 3 will be used and the results are compared with each other to obtain a full understanding of rock support effectiveness in different geological condition. Stress ratio in the numerical simulations with different geological conditions is set to be one. The thickness of sill pillar is set to be 2 m considering the current condition of sill pillar at Modi Taung gold mine. Figure 5.1 shows the comparison of failure zone occurred with different support system in various geological conditions. Figure 5.1 (a) shows the potential stope failure without any support, Figure 5.1 (b) describes the stope failure condition after installing with cable support, and Figure 5.1 (c) shows the stope failure condition after installing with shotcrete support.

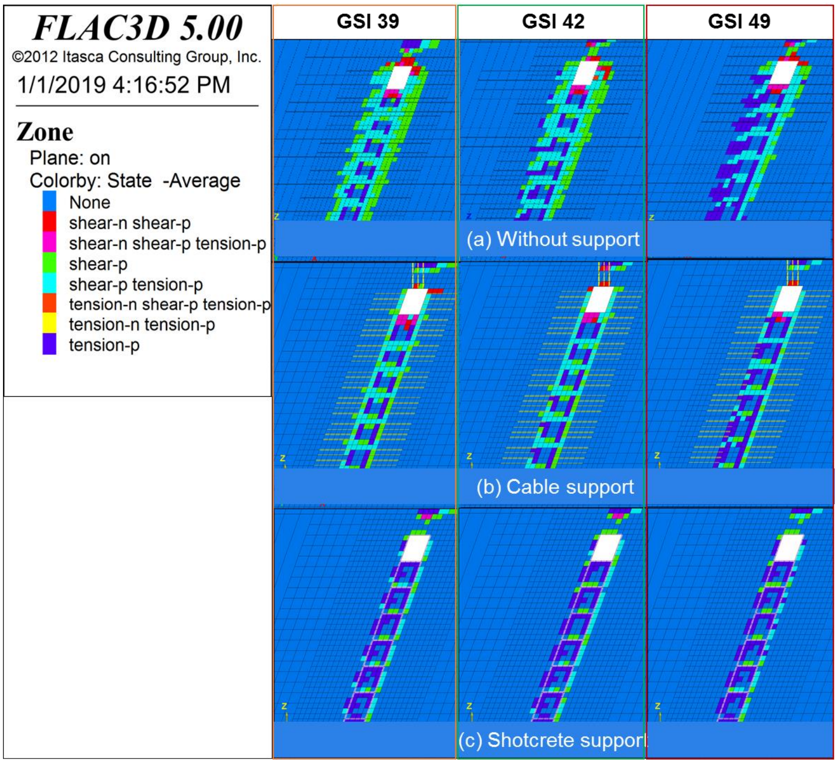

Figure 5.1 Occurrence of failure zone after support system with different GSI under previous mined-out regions.

The results of figure 5.1 (b) describe that the failure zones are reduced around the stope opening up to certain amount after installing countermeasure support with the cable bolts.

However, shear failures around the stope are still occurring in all geological conditions. After applying shotcrete support to stope opening as shown in Figure 5.1 (c), all the current failure conditions are removed from the stope opening, but potential of failure condition is still occurred around the stope. All of these conditions are clearly seen in Figure 5.2 that shows the potential of rock instability around the stope.

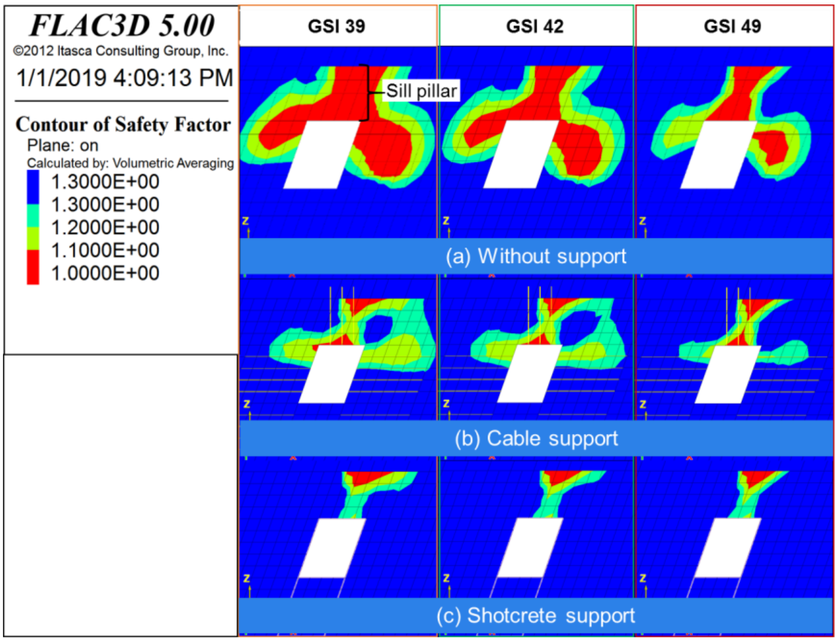

Figure 5.2 Contour of safety factor after support systems in different GSI under previous mined-out regions.

By seeing in Figure 5.2 (b), it can be found that stope opening cannot stabilize by using cable support especially in the sill pillar near the previous mined-out regions, and these results suggested that higher support capacity is needed to stabilize the stope. When the shotcrete support is applied to stope opening as shown in Figure 5.2 (c), all stope openings are reached under stable condition except the final stope excavation. Because of having rock instability in final stope even though the stope is supported by shotcrete, the optimum sill pillar is needed to maintain for stability of stope opening. In order to maintain for the optimum sill pillar, safety factor indexes from each stoping sequence of different geological conditions are recorded and the results are shown in Figure 5.3, Figure 5.4 and Figure 5.5, respectively. As roof stability is primarily determined in many underground mines especially in the presence of structural discontinuities, the monitoring points for these indicators is recorded at 0.5 m above the center of stope's roof.

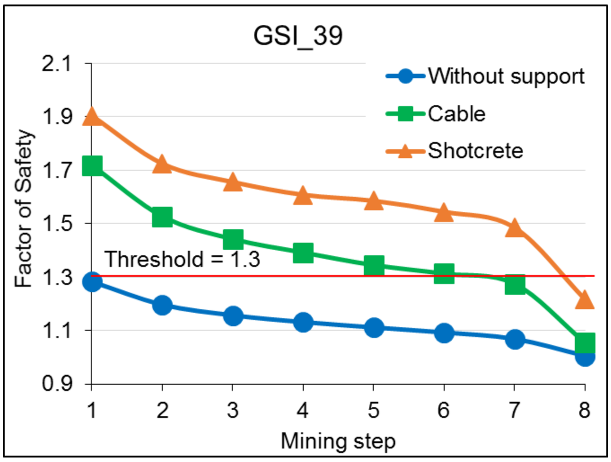

Figure 5.3 Safety factor index recorded from the bound of stoping in GSI 39.

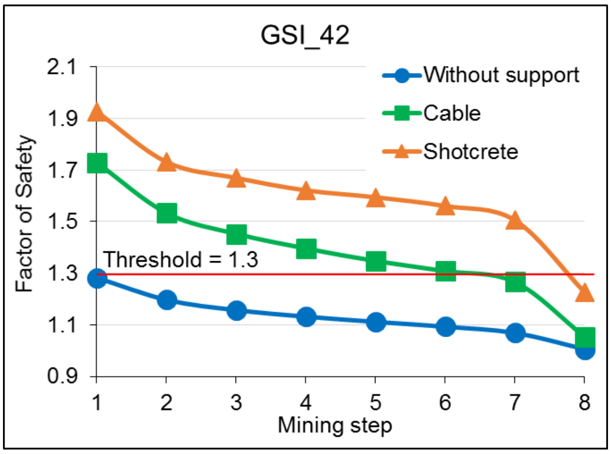

Figure 5.4 Safety factor index recorded from the bound of stoping in GSI 42.

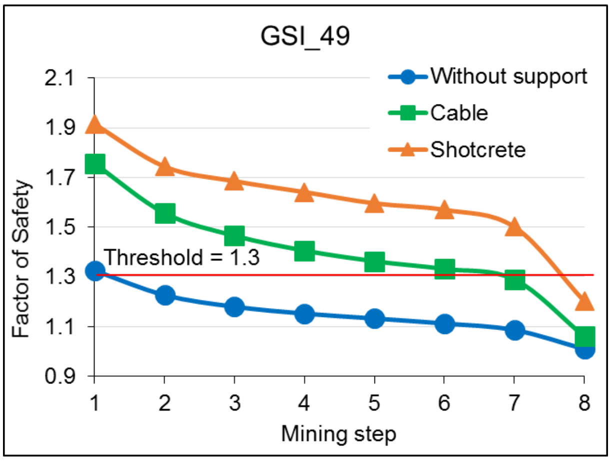

Figure 5.5 Safety factor index recorded from the bound of stoping in GSI 49.

The results of these graphs depicted that the safety factor indicators gradually decrease as the mining steps increase (i.e from the base to top of the stoping sequence). All results suggested that the stope cannot stabilize 0.5 m above the roof without any support systems and become stabilize after supported by cable and shotcrete except the stopes near the overlaying mined-out regions. When applying countermeasure support to the stope opening, economical aspects must be considered especially when the ore grade is low. On the other hand, the stability of stope must also be ensured to avoid unnecessary accident.

Form the Figures 5.3 and 5.4, by considering economic aspects for supporting cost, it is recommend that the stope should be installed with cable bolt up to 5th slice and the next 3 slices should be continued with shotcrete support to make sure for stabilizing the stope in GSI 39 and 42. In addition, in the simulation of GSI 49, the result shows that the stope can stabilize up to 6th slice with cable bolt support, therefore the shotcrete support should be installed after 6th slice in GSI 49. Apart from the roof failure, the potential of gravity driven failures from hanging wall and foot wall should be paid attention to prevent rock falls from the side walls of the stope.

Moreover, in order to keep an optimum sill pillar under previous mined-out regions, numerical simulations are carried out by increasing the height of sill pillar step by step. The results from these simulations suggested that the yield condition will not continue to previous mined-out regions at 2.5 m in height of sill pillar in all geological conditions. Therefore, it is also recommend that the optimum sill pillar should be maintained at least 2.5 m in height for all geological conditions in order to stabilize the stope opening under the previous mined-out regions.

5.3.2. Optimize the stope stability in different stress ratios

Near-by mining activity has a significant influence on the state of stress prior to mining a stope. The initial stope in situ stress state is directly affected by mining sequence and the location of a stope relative to adjacent mined stopes (Wang 2004). As mining progresses and more stopes are mined, stresses will be transferred to the remaining unmined stopes. In order to understand the effectiveness of support systems for stope instability under different stress ratio, analyses over stress changing are investigated as a preliminary assessment of stope’s instability under overlaying mined-out regions and optimize the stability by installing different support systems. To assess this, the K, ratio of horizontal stress to vertical stress was changed ranging from 0.5 to 1.5 in the simulation.

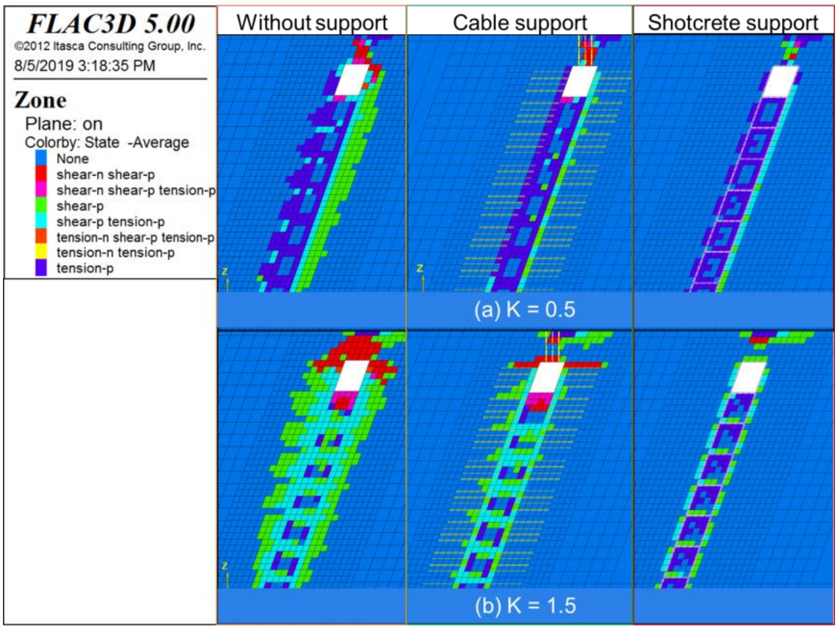

Figure 5.6 Occurrence of failure zone after support system with different stress ratios.

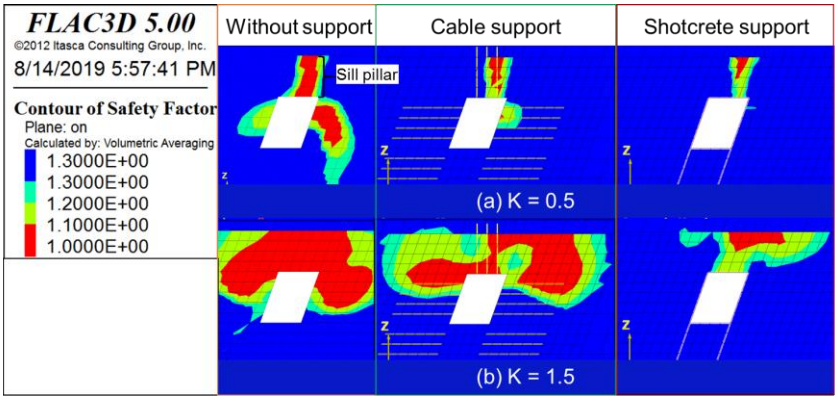

Figure 5.7 Contour of safety factor after support systems in different K ratios.

Figure 5.6 gives the occurrence of failure zone, and Figure 5.7 describes the potential of rock instability before and after supporting countermeasure system. Based on the results of stress ratio of 0.5, the unstable area is reduced by the effectiveness of both active and passive type supports. However, the instabilities of stope opening are left at the final stope and these unstable areas are more occurred at the foot wall region. Different from stress ratio of 0.5, the simulation results of stress ratio of 1.5 show that the unstable areas are more propagated around the stope opening even though the countermeasure supports are applied. In higher stress ratio, the cable support seemed cannot apply effectively near the sill pillar under previous mined-out regions. Therefore, shotcrete support is applied in the bound of stoping sequence, and the results show that the application of shotcrete is proven effective to stabilize stope in higher stress ratio. Nevertheless, yielded elements in the sill pillar are not reduced significantly from the model supported by shotcrete. In order to know stability of stope and optimum sill pillar, the indexes of safety factor collected in the bound of stoping sequence are described in Figure 5.8 and Figure 5.9.

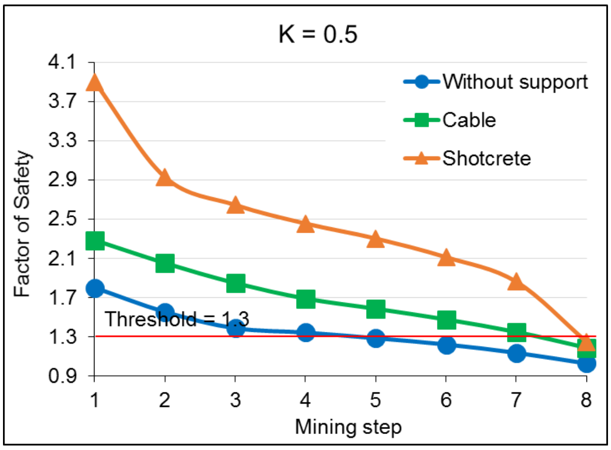

Figure 5.8 Safety factor index recorded from lower stress ratio, K = 0.5.

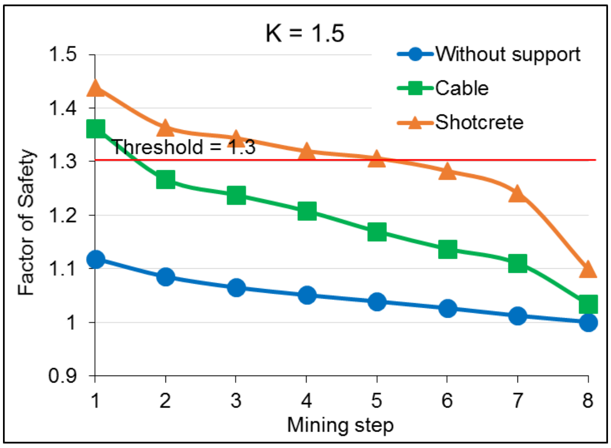

Figure 5.9 Safety factor index recorded from higher stress ratio, K = 1.5.

By seeing the graph in smaller K ratio, the index of safety factor recorded from cable support are located in stable region up to 6th slice and instability will occurred after 6th slice. This result suggests that cable bolt can supported up to 6th slice of stoping sequence, and shotcrete support should continue to make sure for the stability of stope and sill pillar. After shotcrete support to the final stope opening, the simulation results suggested that the optimum sill pillar should keep 2.5 m in height for smaller K ratio.

From the results of Figure 5.9, almost all of the safety factor index from cable support in larger K ratio are found in unstable region, and these indexes from shotcrete support are located in stable region up to 4th slice in larger k ratio. From these results, since cable support cannot anchored efficiently, shotcrete support alone is recommended for countermeasure system in larger K ratio. In addition, mining activities should paid attention since 6th slice in the bound of stoping sequence and the optimum sill pillar should keep at least 3 m in height for larger K condition.

5.3.3. Optimize the stope stability in different vein dips

Effects of different vein dipping under previous mined-out regions in Chapter 3 suggest that the potential of rock failure might be developed from hanging wall and roof in lower vein dip, and these failure development come more from foot wall and roof in steeper vein dip. In order to optimize the stability of stope and sill pillar under previous mined-out regions, the simulations for the effectiveness of countermeasure system with cable and shotcrete support are conducted. The results are described in Figure 5.10 for failure zone development and Figure 5.11 for rock instability around the stope opening.

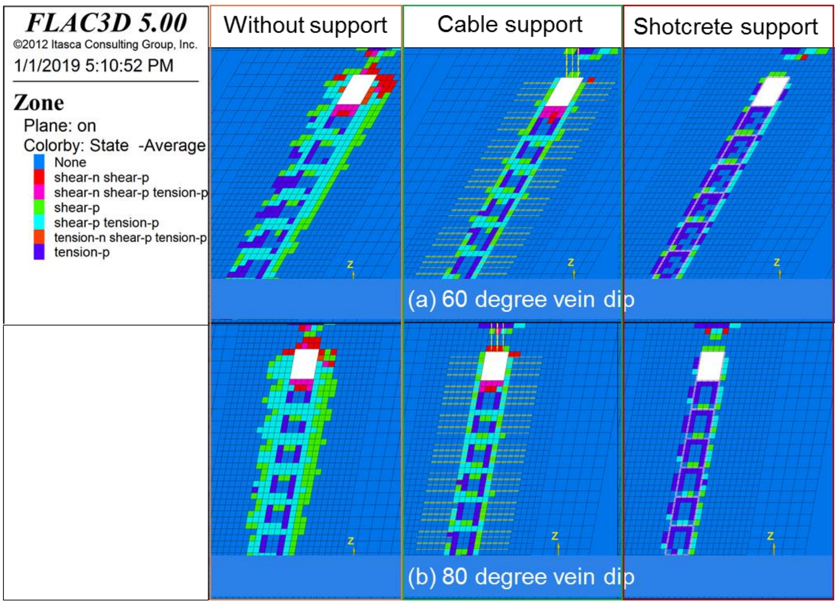

Figure 5.10 Occurrence of failure zone after support system in different vein dips.

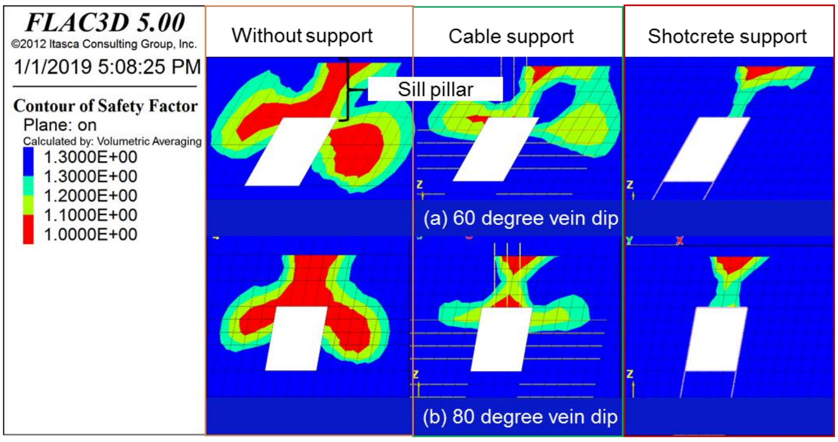

Figure 5.11 Contour of safety factor after support systems in different vein dips.

All the results from these two figures show that rock instabilities especially from hanging wall are still propagated around the stope opening in the model with lower vein dip after cable support is applied. On the other hand, the potential of rock instabilities are spread more above the stope roof in steeper vein dip. After shotcrete support is applied to the stope, most of yield elements are removed from the stope opening in both models except very few amount of yield element in sill pillar. To describe the condition of stope stability, the safety factor index collected from each stope opening are shown in Figure 5.12 for lower vein dip and Figure 5.13 for steeper vein dip.

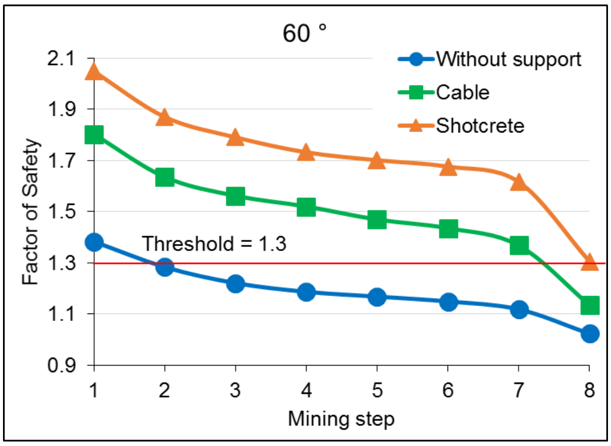

Figure 5.12 Safety factor index recorded from lower vein dip.

From the graph of Figure 5.12, the results suggest that the cable support can be installed up to 7th slice, and shotcrete support should apply at the final stope to remove instability in the sill pillar. Diversely, the cable can be supported up to 6th slice and shotcrete should apply after 6th slice to stabilize the stope and sill pillar. From the simulations, appropriate sill pillar should be maintained at least 2.5 m in height for all vein dip models. In addition, attention should be paid the failure potential from hanging wall in lower vein dip because few amount of yield elements might be developed from that side during the stoping sequence.

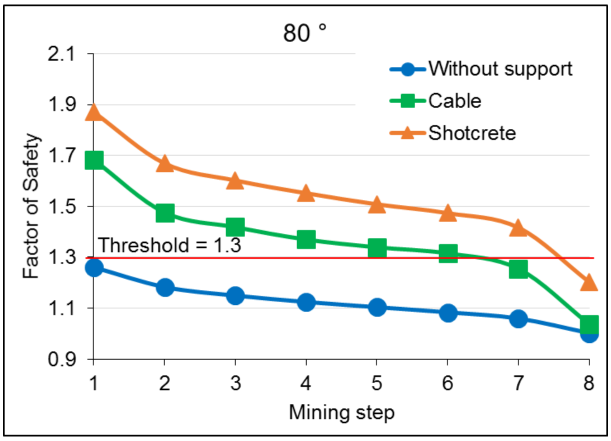

Figure 5.13 Safety factor index recorded from steeper vein dip.

5.3.4. Optimize the stope stability in different stope widths

According to the simulation results of different stope widths from Chapter 3, the necessity of an appropriate support system is suggested in wider stope opening since higher induced stresses are expected to redistribute in the surrounding rock mass in wider stope condition. For this sense, the capability of countermeasure system by using cable bolt and shotcrete support to wider stope opening is investigated in this section. Figures 5.12 and 5.13 show the condition of stope opening after support systems are applied. The results from these figures show that the rock instabilities around the stope and sill pillar cannot stabilize clearly by using cable support, and these results suggest that higher supporting capacity is needed for more stabilizing around the stope opening. After shotcrete support is applied to the stoping sequence, the results show that the yield elements are removed some amount from the stope and pillar. However, few amount of yield elements are still propagated to the previous mined-out regions especially in wider stope due to the prospect of higher redistributed stress and induced stress. In order to determine the rock stability around the stope opening, Figure 5.16 and Figure 5.17 describe the value of safety factor index collected from the stoping sequence of 3.5 m and 5 m stope width, respectively.

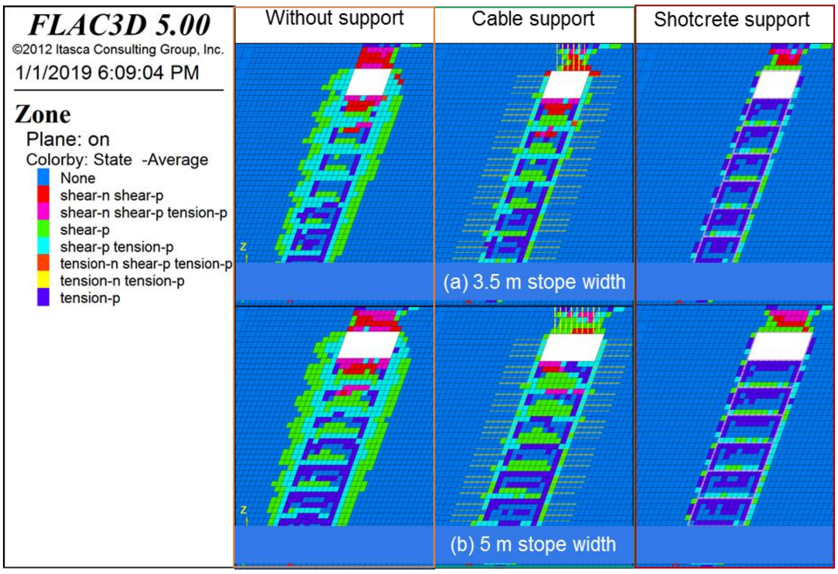

Figure 5.14 Occurrence of failure zone after support system in different stope widths.

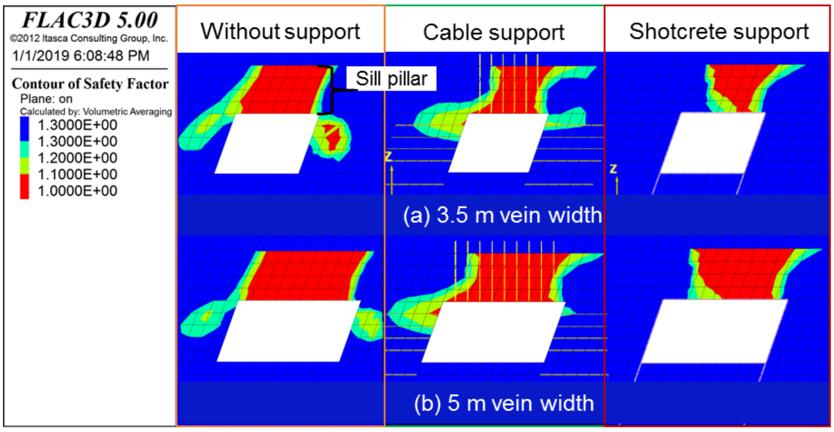

Figure 5.15 Contour of safety factor after supporting systems in different stope widths.

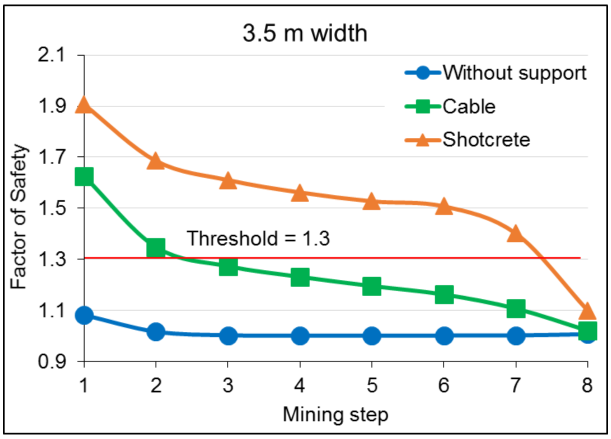

Figure 5.16 Safety factor index recorded from 3.5 m stope width.

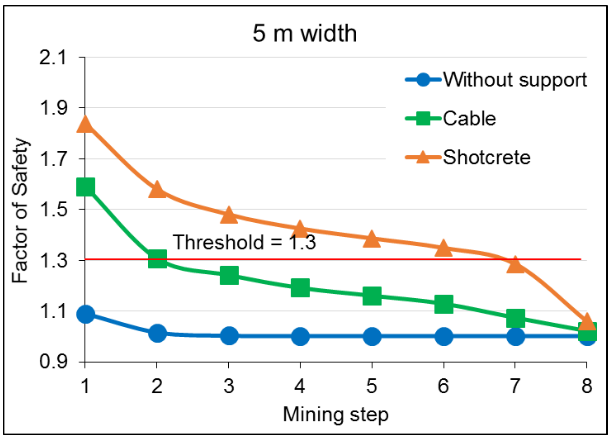

Figure 5.17 Safety factor index recorded from 5 m stope width.

From the simulation results of Figure 5.16 and Figure 5.17, the stope opening with cable support seem to be unstable after 1st slice is excavated. By supporting with shotcrete in 3.5 m stope width, the stope can maintained in stable condition up to 7th slice, and the final stope will be reached in unstable region near the sill pillar. For maintaining the stable sill pillar, from the simulation results, the sill pillar should keep at least 3.5 m thickness. On the other hand, according to the safety factor index in Figure 5.17, the stope with 5 m width can support up to 6th slice with shotcrete, and the other two stopes are reached in unstable region. For the need of stable stope, the optimum sill pillar for 5 m stope width model should keep at least 4.5 m in height. For the simulations with different stope width, it can be seen that special attention needs to be paid on the stope roof in wider stope roof because the potential of rock instability are mostly developed at the stope roof in wider stope width.

5.4. Optimization of stope stability under mountain slope surface

In Chapter 4, the assessments on the instability of stope mining under the mountain slope surface are carried out, and the occurrence of failure zones are recorded at three cutting planes during the stoping sequence in Modi Taung gold mine. From the simulations, the results pointed out that the rock mass strength can gradually decreased towards the slope surface, and the occurrence of failure zones is more propagated to the stope near the slope surface because of not only by the effects of stress redistribution from lower stope mining but also by the influence of higher differential stress occurring near the mountain slope surface.

In this section, the investigations on the optimization of stope stability under the mountain slope effects are conducted with countermeasure supports. The basic numerical model from Chapter 4 is carried out in this simulation with the rock mass parameters of GSI 42 and the stress ratio of 1. As a countermeasure support, the 3 m cable bolt with 1 m x 0.5 m spacing and the shotcrete support with 50 mm thickness are applied to the stope opening in the bound of stoping sequence. The material properties of cable bolt and shotcrete are the same as the one described in Tables 5.1 and 5.2. From the numerical simulations, the monitoring results are collected from three cutting planes of stope mining near the slope as shown in Figure 4.5 in Chapter 4.

The results from numerical simulations are shown in Figure 5.18 for failure condition around the stope and Figure 5.19 for rock instabilities propagated from the stoping sequence after applying with different support system to the stope under the mountain slope surface.

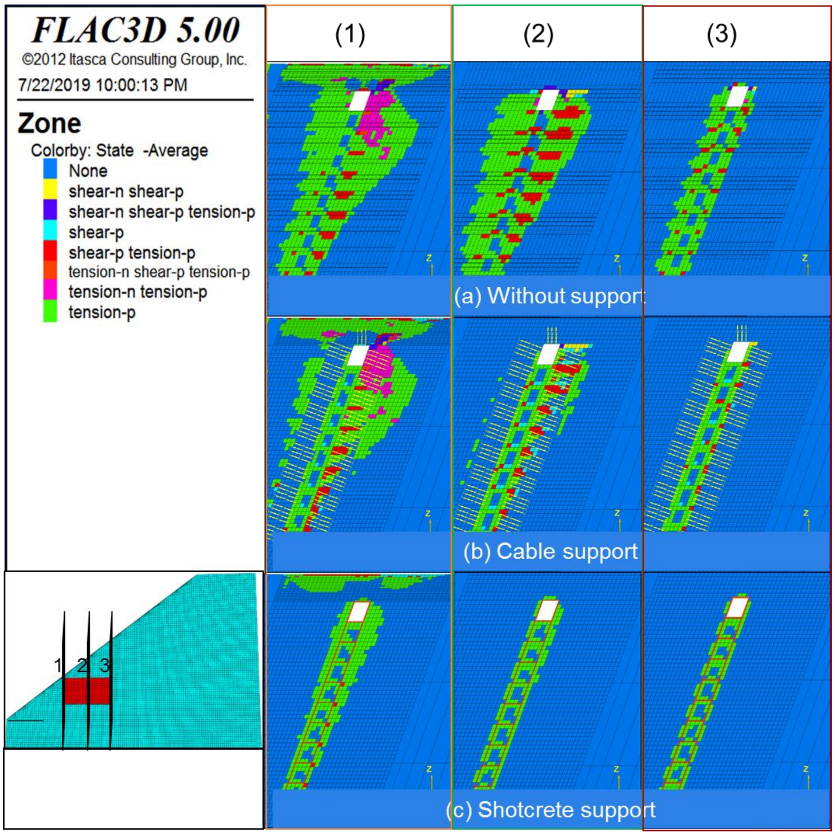

Figure 5.18 Failure zone in the stope mining under mountain slope surface with different support systems.

When the cable support is applied to the stope opening, the failure zone are slightly decreased especially in footwall side as can be seen in Figure 5.18. This result shows that the cable support is not efficient for stope instability when the underground excavation is closing to the slope surface. Diversely, when the shotcrete is support, the rock instability are removed from the stope opening, however it is still doubtful whether the shotcrete support can be efficient or not at the nearest part of stope to the slope surface. Even small amount of yield zone is appeared at the middle of stope, it is seem to be fine for the stability of stope as shown in Figure 5.19. Therefore, the monitoring records for the length of optimum crown pillar are required to stabilize the stope not to be collapsed.

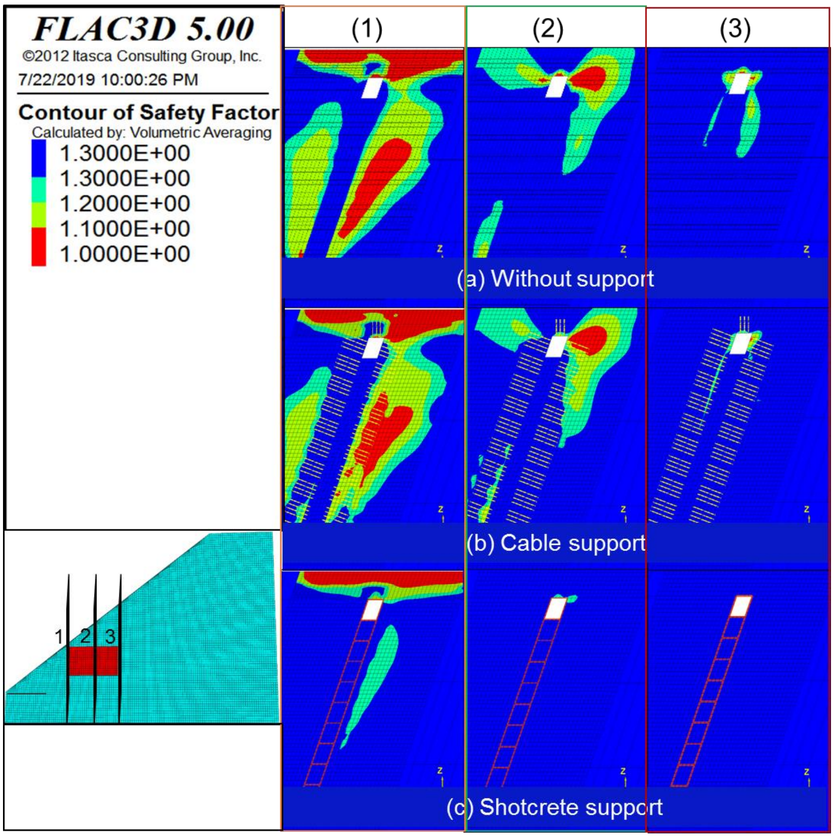

Figure 5.19 Rock mass instability of stope under slope surface with different support systems.

5.4.1. Optimum crown pillar at the mountain slope surface

In general, increasing the thickness of crown pillar could be a way to prevent the rock mass failure to the stope excavations under the mountain slope surface. However, valuable ore body might be formed in the crown pillar, hence stope mining should be carried out as much as possible. At the same time, special attention should be paid for the crown pillar failure since in situ differential stresses are exhibited a higher value owing to the mountain slope effect. Increasing the crown pillar thickness will keep more stable to the stope opening, however it make a reduction of mining recovery since large amount of ore body might be left in the pillar. Therefore, numerical investigations need to be carried out on the stope and crown pillar stability with the aims to optimize the stope stability as well as maximize the mining recovery.

Because rock instabilities around the stope are propagated to the slope surface from the final slice as described in previous Section 5.4, the numerical investigations for the optimum crown pillar are carried out with different countermeasure support at the mountain slope surface. The simulation results are shown in Figure 5.20 for the rock mass instability propagated to the slope surface with different supporting capacity, and Figure 5.21 shows for the comparison of optimum crown pillar height with different support system.

During the simulations, the results show that the yield elements are propagated since 14.7 m below the slope surface without any support systems as shown in Figure 5.20 (a). After the cable support is applied in Figure 5.20 (b), the rock mass instability are still induced from the stope at 14.7 m below the slope surface without decreasing the potential of stope failure. This result suggests that cable support is not effective for the stope stability near the slope surface and more supporting capacity will be needed to stabilize stope and crown pillar. When the shotcrete is applied to the stope opening, Figure 5.20 (c) shows that the yield elements are decreased obviously from the stope opening and the instability are generated from the stope at 8 m below the surface. In order to increase mining recovery, the combination support with cable and shotcrete support is applied to the stope opening. However, the yield elements are slightly decreased from the crown pillar and it is still propagated at 8 m below the slope surface as shown in Figure 5.20 (d).

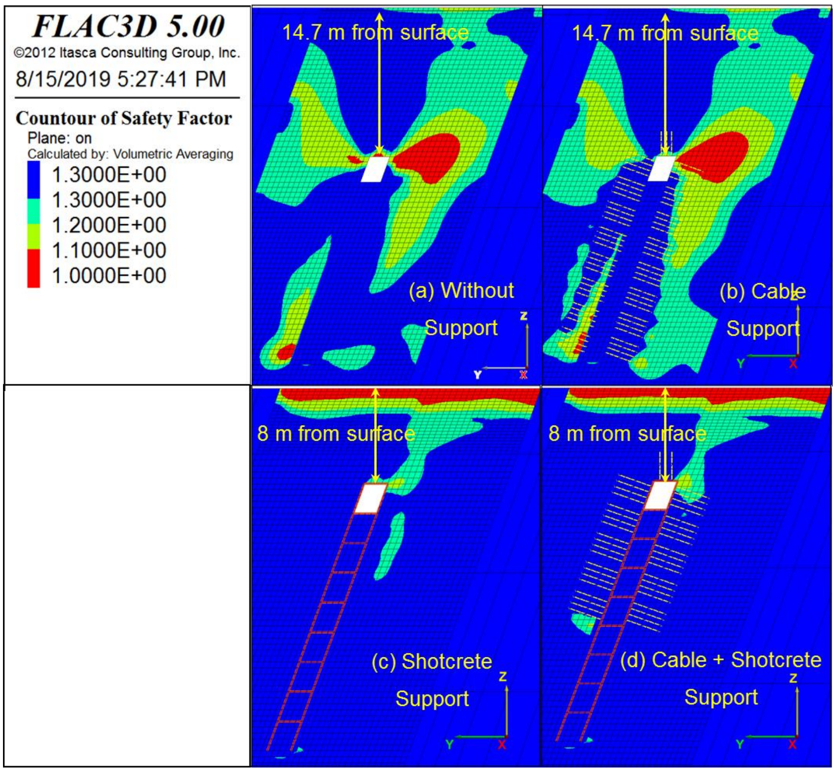

Figure 5.20 Rock mass instability propagated to the slope surface after installing different support systems.

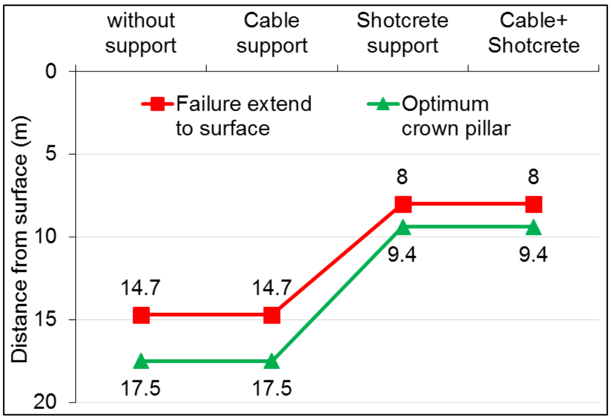

Figure 5.21 Comparison of optimum crown pillar with different support systems.

As can be seen in Figure5.21, the points on the red line with different support systems are described that the yield elements are propagated to the slope surface from these points. For maintaining the stability of crown pillar, the yield elements should not be propagated to the slope surface, hence numerical simulations are conducted in order to bring the optimum crown pillar. From the simulation results, the points in the green line are described as the length of optimum crown pillar which need to be maintained to stabilize under the slope surface. In addition, according to the results of Figure 5.20 and Figure 5.21, as the cable bolt cannot anchored effectively to stabilize the crown pillar near the mountain slope surface, therefore the use of shotcrete support alone is recommended near the slope surface considering installation cost. Furthermore, economic analysis needs to be carried out to compare its cost whether it is worthwhile to be mined or not.

5.4.2. Optimum crown pillar with different geological condition under the mountain slope surface

In order to keep the optimum crown pillar of stope mining under the slope surface with different geological conditions, the effectiveness of rock support to the stope mining is investigated in the numerical model with different GSI value. The rock mass properties for all geological conditions are the same as shown in Table 3.2 mentioned in Chapter 3. As cable bolt cannot support effectively for anchoring near the slope surface as described in previous Section 5.4.1, shotcrete support alone is applied to the stoping sequence. From the simulations, the results for the optimum pillar thickness to different GSI are described in Figure 5.22, respectively.

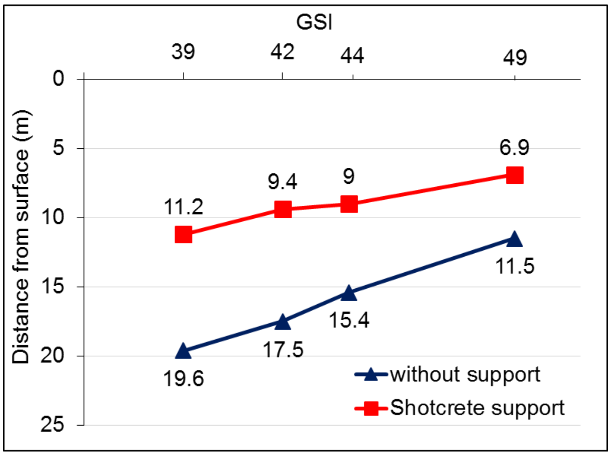

Figure 5.22 Comparison of optimum crown pillar without support and shotcrete support with different geological conditions.

By seeing the results, if the stope mining under the mountain slope surface is carried out with no support, the optimum crown pillar needs to maintain 19.6 m in GSI 39, 17.5 m in GSI 42, 15.4 m in GSI 44, and 11.5 m in GSI 49, respectively. When the shotcrete support is applied to the stope opening, the optimum crown pillar can maintain 11.2 m in GSI 39, 9.4 m in GSI 42, 9 m in GSI 44, and 6.9 m in GSI 49, respectively. From these results, it can be seen that the lengths of optimum crown pillar are improved approximately 40 % in all geological conditions after shotcrete is supported which mean more mineral can be mined out from the vein in safe condition under slope surface. At Modi Taung gold mine, especially in Shwesin vein system, the high grade ore are often mineralized to the host rock in shallow level, the stope mining might be worthwhile to be mined with shotcrete support near to the slope surface.

5.4.3. Optimum crown pillar with different stress ratio under the mountain slope surface

In previous Section 4.3.3.1 in Chapter 4, the results showed that the occurrences of failure zones and rock instability in stope and crown pillar are severe to the stope mining near the slope surface due to the presence of high in situ differential stress. Because of this conditions, the effectiveness of rock support as a countermeasure system is needed to investigate for keeping the optimum crown pillar and stable stope opening in different stress ratio. Hence, the passive support, shotcrete, is installed to the stoping sequence under the slope surface for improving the stability of stope and crown pillar. From the simulations results, the optimum crown pillars that need to be maintained for different stress ratio are shown in Figure 5.23.

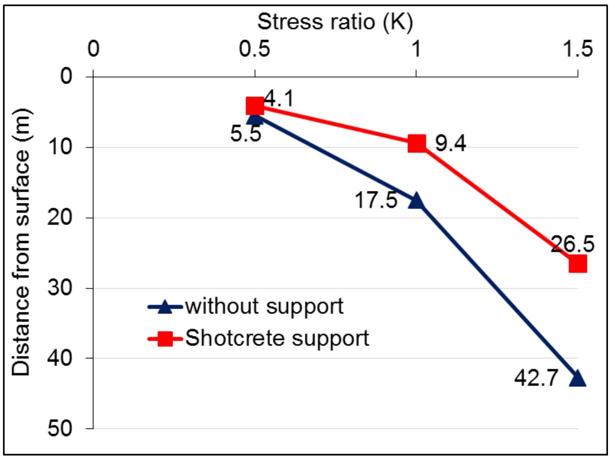

Figure 5.23 Comparison of optimum crown pillar without support and shotcrete support with different stress ratios.

The results showed that the crown pillar thickness from simulation of stope supported by shotcrete for smaller stress ratio is 4.1 m while 5.5 m thickness is needed for no support condition. Contrarily, when the stope mining is operated under high stress ratio condition, larger crown pillar is required for stability as shown in Figure 5.23. Therefore, stope mining in high horizontal stress condition should be paid attention under the slope surface, and the optimum crown pillar for this condition should keep at least 26.5 m from the surface with shotcrete support. If the high grade ore is mineralized in the crown pillar, the higher supporting system to the stope opening such as combination of shotcrete and H beam, and/or alternative countermeasure method such as surface grouting should be considered to improve the stability of stope and to maximize the crown pillar recovery. Hence, the economic analysis for stope mining become one of the important issues in this mine to compare the cost whether it is worthwhile to be mined or not.

5.5. Economic analysis for stope mining

The economic aspect of a mine is based on the evaluation of benefits and costs ratio which can be expressed in equation below by dividing the total cash inflows with the total cash outflows.

$$B/C = \sum PV \text{ of net cash inflows} / \sum PV \text{ of net cash outflows}$$

where B is benefit, C is cost, and PV is present value. If B/C value is greater than one, the stope can be economically mined. However, it is not recommended to develop the mine if the value is less than one. Based on the production sharing proposal of previous exploration company (IMHL), the total operating cost (including off site costs, underground mining costs (exclude support system), processing costs, and site G & A costs) is 111.13 US$/ton in 2003 (IMHL 2003). However, an appropriate discounted rate is needed to reflect the timing of net benefits. According to World Bank data for the inflation of Myanmar, the GDP deflator in 2003 is 20.497 and 8.279 in 2018. Therefore, the interest rate in this period for Myanmar is 5.4 % annually. By calculating discounted cash flow (DCF), the total operating cost in 2018 is 244.59 US$/ton.

According to previous mining company, the average Au grade in Modi Taung is 10 g/ton, and ore density is 2.67 ton per cubic meter. Hence, the ore tonnage for one stope (2m in width x 2.5m in height x 50 m length) is 667.5 ton, and the Au amount in one stope will be 214.6062 tr.oz. According to London Metal Exchange (LME), the gold price is 1503.2 US$/tr.oz on October 2019, and the benefit cost ratio for the stope mining can be described as follow:

$$\mathrm{B/C} = \frac{Au\,price\ *\ Au\,amount}{[(total\ operating\ cost\ *\ ore\ amount)\ +\ \text{supporting cost}]}$$

For each stope, in the case of cable bolt supporting system with the vertical spacing of 0.5m and horizontal spacing of 1m and weld mesh (350 m^2) covering the wall of stope, the calculation of B/C value for stope mining with cable bolt countermeasures system is shown in follow:

$$\mathrm{B/C} = \frac{1503.2\frac{\$}{tr.oz}\ *\ 214.6062\,tr.oz}{\left(244.59\frac{\$}{\text{ton}} * 667.5\ \text{ton}\right) + \left(63\frac{\$}{\text{unit}} * 50\ \text{unit}\right) + \left(86.63\frac{\$}{m^2} *\ 350\,m^2\right)} = 1.64$$

For the case of shotcrete supporting method with 50 mm thickness and weld mesh (350 m^2) covering at the wall of stope, the calculation of B/C value for stope mining with shotcrete countermeasures system is shown in follow:

$$\mathrm{B/C} = \frac{1503.2\frac{\$}{tr.oz}\ *\ 214.6062\,tr.oz}{\left(244.59\frac{\$}{\text{ton}} * 667.5\ \text{ton}\right) + \left(671.05\frac{\$}{\text{m}} * 50\ \text{m}\right) + \left(86.63\frac{\$}{m^2} *\ 350\,m^2\right)} = 1.42$$

Since the benefit cost ratios with two types of countermeasures systems for each stope mining are more than one, therefore it can be concluded that the stope mining in Modi Taung gold mine can be carried out economically with the adoption of both supports. On the other hand, the safety in the working stope is the most important concern in underground mining. As a result, in the zone where the potential of rock failure is high, it is plausible to adopt the higher support capacity such as multiple support combination system.

5.6. Discussions

Because of the instability occurred from the stope and the sill pillar under previous mined-out regions in Chapter 3 and crown pillar near to the mountain slope surface described in Chapter 4, the effectiveness of a support system for optimizing the stability of sill pillar and crown pillar is conducted in different mine condition in this chapter.

From the simulations for the optimization of stope and sill pillar stability under the previous mined-out regions, the results suggested that the cable support can be applied in the stoping sequence, however rock instability is still developed around the stope opening.

Therefore, shotcrete support should choose in some stoping process due to its higher supporting capacity especially in the final stope near the previous mined-out regions. The countermeasure supports for stope openings with higher stress ratio and wider stope are recommended by shotcrete support alone since cable support cannot effective to these conditions. In addition, mining activities in those conditions should be paid attention since 6th slice in the bound of stoping sequence to make sure the stability of stope and sill pillar under the previous mined-out regions.

From the simulations of stope optimization with the influence of the mountain slope surface, the results pointed out that the cable bolts cannot anchored effectively to stabilize the stope opening and crown pillar near to the slope surface. Therefore, the use of shotcrete support alone is recommended to the stope openings near the slope surface. Because of higher differential stresses affected near the slope surface, the stability of crown pillar should be optimized by using higher quality of rock supports. After shotcrete support is applied to the stope opening at the nearest part of slope surface, the results show that the stope can be mined-out more below the slope surface. By seeing the economic analysis in Section 5.5, as the stope mining in Modi Taung gold mine can be economically carried out, a countermeasure system with higher supporting capacity should be installed to the working stope to ensure the safety of mine workers.

5.7. Conclusions

In common sense, increasing the thickness of pillar could be a way to prevent the rock mass failure to the stope excavations. However, valuable ore body might be formed in the pillar. Therefore, if the ore grade and ore deposits are seem to be large in the pillar, stope mining should be carried out as much as possible, accordingly some considerations upon safety issues and ore recovery are needed to be revised. At the same time, special attention should be paid for the stability of pillar to avoid unexpected rock mass failure.

Based on the results from this Chapter, it can be suggested that the countermeasure system with higher supporting capacity should be applied to the final stope near the previous mined-out regions since the rock instability is still developed around the stope mining near the upper mined-out activities even though the countermeasure support system is applied. In addition, if the stope mining is developed under the condition of higher stress ratio and wider stope width, the countermeasure systems are recommended by shotcrete support alone since cable support cannot effective to these conditions under the previous mined-out activities.

Furthermore, the rock mass strength can gradually decreased towards the slope surface, and the occurrence of failure zones is more propagated to the stope opening near the slope surface because of not only the effects of stress redistribution from lower stope mining but also the influence of higher differential stress occurring near the mountain slope surface. Therefore, if the underground excavation is operated under the mountain slope surface, the countermeasure system with shotcrete support alone is recommended to the stope openings near the slope surface since the cable bolt cannot anchored effectively to stabilize the stope opening and crown pillar near to the slope surface.

CHAPTER 6

CONCLUSIONS

Stope mining is the most common mining method adopted in underground metal mines of Myanmar. Most of the underground mines in Myanmar are being mined-out or still mining at the easily accessible shallow places. Therefore, underground mining activities are going to continue to progress into deeper levels to fulfill the mineral supply. However, the assessments on the stability of mine opening still remain quite limited, and there are not so many recorded data regarding rock mass failures accidents. Hence, the study on the stability of underground mining become one of the important issues in Myanmar mining industry to mitigate the unpredictable nature of rock failures. In this research, the study is carried out for the occurrence of rock mass instability and optimize the stability of stope mining, sill pillar under previous mined-out regions, and crown pillar near the mountain slope surface in Shwesin vein system of Modi Taung gold mine, one of the largest underground gold mine in Myanmar.

To investigate the instability of stope mining and prevention of stope failure, firstly, it is important to understand the geological and geotechnical characteristics of rock mass in research mine site. The Modi Taung gold mine has three main vein system namely: Shwesin, Sakangyi, and Htonegyi (Mitchell et al. 2004). These veins are hosted by four main lithologies: mudstone, sandstone-siltstone, limey sandstone or limestone, and igneous intrusions. Each vein system consists of either a single vein, or multiple parallel vein separated by host rock. All veins are massive quartz and the oxidation is likely due to groundwater interacting with the ore through a leached zone. The Modi Taung gold deposit (area A) is economically viable due to three main reasons; it has concentrated high gold grades of 10-300 g/t Au and up to 3000 g/t Au, large lateral extent and if further investigated may be proved to be even larger than expected, and steeply dipping veins that continue to depth which makes stoping and extraction of ore easier (Traynor 2015).

From the field observations of Modi Taung gold mine, many cracks and joints within rock mass are found in underground tunnels and stopes. These rock mass conditions will be effected to the instability of underground excavations. Additionally, conditions in adits are very humid with meteoric water seeping through geological structures. These meteoric water are interacting with surrounding rocks and can be affected to the weathering of host rocks, and then might be reduced the strength of host rocks. From the laboratory experiments, the uniaxial compressive strength of intact host rock and vein from Modi Taung gold mine are 148 MPa and 140 MPa, respectively. According to the intact rock parameters, the rock mass strength from Modi Taung gold mine is strong. However, to complete full estimation of rock mass, RQD from borehole data, discontinuities in rock mass, heavy rainfall, and other geotechnical factors are needed to consider for the stability of underground opening in this mine since these parameters can also have a great effect on the strength and deformability of rock mass. Understanding the rock mass conditions, appropriate mining methods should be considered before starting the mine operations.

In order to understand the influence of previous mined-out activities to the instability of stope, numerical simulations are conducted for the stope mining without overlaying mined-out effects compared with simulations for the stope with overlaying mined-out effects. From the simulations, the results indicated the potential displacements to the stope opening can only affected by its own induced stress when it is mined without previous mined-out activities. On the other hand, the potential of displacement and rock mass failure to the new opening are developed not only by induced stress but also the influence of the redistributed stress from previous mined-out activities. Based on the simulations in the bound of stoping sequence, the results showed that the instability of rock mass around the stope mining obviously increases as the stope progressing move towards the upper slices and it propagates to the previous mined-out regions. For this sense, the numerical simulations are conducted to understand the possibility of unstable regions with different sill pillars, and determine the appropriate sill pillar to avoid rock falls from the previous mined-out regions. Based on the results, the sill pillar between the final stope and upper mined-out regions should be maintained at a certain thickness to stabilize the stope and to prevent the rock falls from the previous mined-out regions. Additionally, the potential failures from the hanging wall and foot wall should be paid attention to prevent rock falls from the side walls of the stope.

When the mining activities are carried out under the previous mined-out regions, it need to be taken account the influence of the deterioration of backfilling materials from the mined-out area. From time to time, the strength and deformability of backfilling materials might be getting worse, and this condition can effect to the stability of underground mining under previous mined-out regions. Based on the simulations, the results suggested that it needs to be maintained a proper sill pillar due to a large potential of rock instability around the stope and pillar due to the deterioration of backfilling waste rock in the previous mined-out regions. In addition, during mining activities in previous mined-out regions, there might be a condition for empty backfilling after mined-out the stope. Therefore, in the case of no backfill condition during mining activities in previous mined-out regions, the optimum sill pillar should be maintained more thickness to ensure the stability of stope under the previous mined-out regions.

In order to fully understand the stability of stope and sill pillar due to the influence of previous mined-out activities, the simulations of stope mining without any support system in various mine conditions have been conducted under the overlaying mined-out regions. From the simulation results, the potential of instability is likely to occur in the lower vein dips, more severe geological condition, wider stope width, and higher stress ratio condition. Moreover, the backfilling materials cannot clear the instability in the vicinity of the stope. Therefore, the selection of backfilling material should be based on available backfilling materials and economic factor only instead of stability reason. The results in this simulations are pointed out the potential instability of stope mining by the influence of the previous mined-out activities and the necessity of appropriate countermeasure system to optimize the stability of stope and sill pillar in different mine conditions.

At Modi Taung gold mine, there are many mountain slopes along the occurrence of ore mineralization, and hence underground mining have to operate under the slope surface. Mining activities under slope topography may affect more on the variation of stress and failure zones than other places of rock mass due to the risks of the sloping condition. From the investigation of rock mass condition near the mountain slope surface, the results indicate that the differential stresses are progressively increased from the inner part of the rock mass towards the mountain slope surface at the same level. The results point out that the rock mass strength are gradually decreased towards the slope surface.

When the mining sequence is developed under the mountain slope surface, the simulation results show that the occurrence of failure zones is more propagated to the excavation near the slope surface. This results suggested that the rock mass in the nearest place to slope surface is more affected by not only from the stress accumulation of lower stope mining but also the influence of higher differential stress at the slope surface. As the rock instability are more severe and propagated up to the slope surface, special attention needs to be given in the bound of stoping sequence below the slope surface.

When the stope mining under the mountain slope is carried out in higher horizontal stress, the failure occurrence are developed more severe condition as the in situ differential stress acting to the rock mass is more concentrated near to the slope surface. Moreover, the results suggested that the potential of stope and crown pillar failure might be experienced more in the stoping sequence with lower geological condition and weaker backfilling materials due to higher differential stress affected to the surrounding rock mass.

Because of rock instability by the influence of previous mined-out regions and mountain slope effects, two types of support systems which are cable bolt and shotcrete support are supported in order to optimize the stability around the stope and pillars, and to see the effectiveness of countermeasure support systems. From economic point of view, cable support is preferred to the shotcrete support due to its lower cost, however the countermeasure system with higher supporting capacity should considered where the potential of rock failure is large.

Firstly, from the simulations for the optimization of stope and sill pillar stability under the previous mined-out regions, the results suggested that the cable support can be applied in the stoping sequence, however rock instability is still developed around the stope opening and sill pillar. Therefore, shotcrete support should choose in some stoping process due to its higher supporting capacity especially the final stope near the previous mined-out regions. Moreover, the results suggested that the countermeasure systems for stope openings with higher stress ratio and wider stope width are recommended by shotcrete support alone since cable support cannot effective to these conditions.

Secondly, from the simulations of stope optimization with the influence of the mountain slope surface, the results pointed out that the cable bolt cannot anchored effectively to stabilize the stope opening and crown pillar near to the slope surface. Therefore, the use of shotcrete support alone is recommended to the stope openings near the slope surface. As the stope mining in Modi Taung gold mine can be economically carried out, a countermeasure system with higher supporting capacity should to be installed to the working stope to ensure the safety of mine workers.

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