Project Report

On

Risk assessment and reinforcement design for a mine stope at depth

TGB4212 - Advanced Rock Mechanics

NTNU

Kunnskap for en bedre verden

Group No: F

28 November, 2023

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1 Introduction

This report presents an analysis of a deep metal mine operation employing the cut-and-fill (C&F) mining method and rill miming method. The project aims to investigate the reasons behind a hanging wall collapse and assess the feasibility of re-mining the ore between two stopes. This report outlines the description of project, methodologies used, results, discussion followed by conclusion and limitations in the project.

The main aim of mining is to extract ore from the ore body. Underground excavation is highly affected by three factors: quality of the rock mass, in-situ stress in the rock mass and size and shape of excavation.

1.1 Types of mining used in the project.

Cut and fill: The cut and fill mining method are a highly selective open-stope mining and is considered ideal for steeply dipping tabular mine deposits in weak host rocks. An ore body or block is mined from the bottom and then progressed upward using this method. The ore is drilled, blasted, loaded and removed from the proceeding stope. Personnel and equipment are protected from injury by backfill, which provides mild excavation support. Raising is driven upward through the ore body to facilitate progression between stops.

Rill mining: A mining technique in which ore are driven down by long hole drilling and blasting by slice of ore to two or three normal slices.

2 Description of the Project

2.1 Project Background

The deep metal mine under investigation is located at a significant depth, and the primary mining method employed is the cut-and-fill (C&F) method. In this method, each stope is divided into five slices or cuts for excavation, with each slice being 5 meters in height and 6 meters in width. The mining process involves the extraction of ore from each slice, followed by backfilling with waste rock and tailing sands to maintain the structural integrity of the stope. Figure 1 shows the length profile (vertical) of the ore body which shows the cut in every stope in each level, type of mining used in the project and collapse happened after the mining operation.

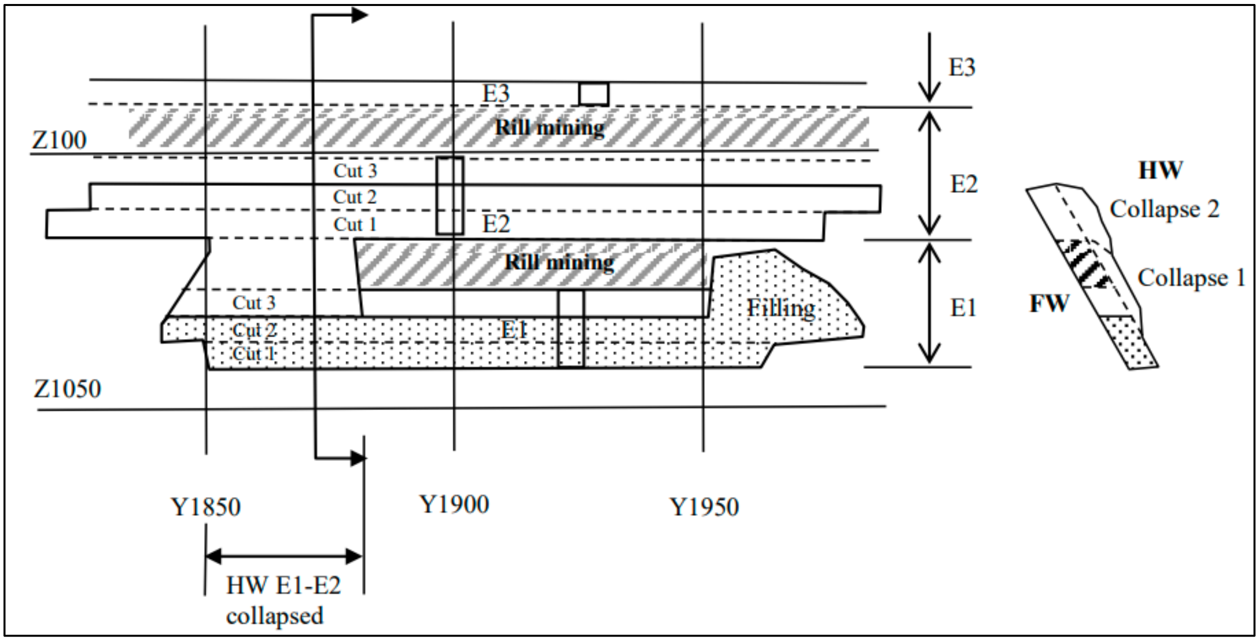

Figure 1: The length profile (vertical) of the ore body.

2.1.1 Initial Mining Process

The mining operation initially followed a well-structured plan, where each slice (Cut 1 through Cut 5) was excavated and immediately backfilled after completion. This approach aimed to ensure that the voids left behind after ore extraction were adequately supported by the backfill, reducing the risk of hanging wall instability.

2.1.2 Deviations from the Plan

However, a critical deviation from the mining plan occurred during the excavation of Cut 3 in stope E1. Instead of proceeding with the planned backfilling after Cut 3, the mining operation continued with rill mining. The voids created by these blasts were not immediately backfilled as per the plan, leading to the formation of unsupported voids of approximately 27 meters in length between stopes E1 and E2.

2.1.3 Hanging Wall Collapse

The consequences of this deviation became evident when the hanging wall (HW) of the mine collapsed into the unsupported void twice, as shown in Figure 1. These collapses halted mining operations and raised concerns about the safety and feasibility of resuming mining activities in this area.

2.2 Geological Context

To understand the factors contributing to the hanging wall collapses, it is essential to consider the geological context. The rock mass in this region is primarily composed of sericitic quartzite, with the presence of chloritic quartzite and chloritic schist as shown in Figure 2. The uniaxial compressive strength of these rock types varies, with sericitic quartzite having a mean strength of 140 MPa ± 70, chloritic quartzite 100 MPa ± 54, and chloritic schist 68 MPa ± 35. The structural geology of the area is characterized by foliations and complex folds, which vary in size and orientation. The ore body dips at about 65º to the horizontal plane, and the foliations are parallel to the ore body's dip, also at 65º.

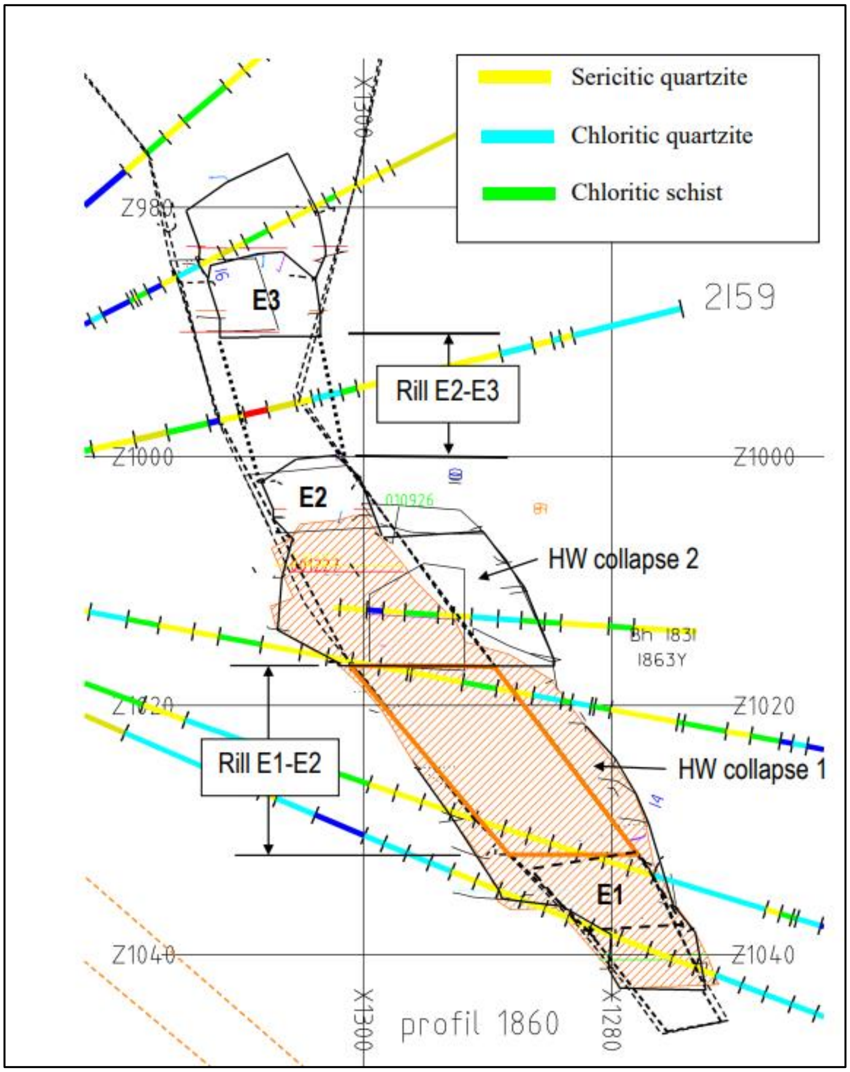

Figure 2:Geological mapping of drill cores in profile Y1860.

2.3 In-Situ Stresses

In this project the in-situ/virgin stress like vertical stress, major principal stress, minor principal stress are estimated by relations given in the problem statement as in (2-1), (2-2), (2-3). The value of Z is estimated as the mean depth of mining, i.e., 1025m.

$$\sigma_v = 0.0265 * Z \tag{2-1}$$

Where σ_v in MPa is the vertical stress, Z is the depth from the ground surface in m.

$$\sigma_1 = 2.5 * \sigma_v \tag{2-2}$$

Where σ_1 in MPa is the horizontal stress parallel with the strike of the tabular ore body in W-E orientation.

$$\sigma_2 = 1.4 * \sigma_v \tag{2-3}$$

Where σ_2 in MPa is the horizontal stress perpendicular to the strike of the ore body in N-S orientation.

2.4 Objectives of the Project

The primary objectives of this project are as follows:

- Investigate the reasons behind the hanging wall collapses between stopes E1 and E2.
- Assess the feasibility of re-mining the ore between E1 and E2 using the rill mining method.
- Provide recommendations for rock support strategies to prevent similar hanging wall collapses in the stope.

This detailed description sets the stage for the project, outlining the critical aspects of the mine's operation, geological context, and the deviations that led to the hanging wall collapses. These insights provide the foundation for the subsequent analyses and recommendations presented in this report.

3 Methods

3.1 Numerical modelling

Computer program RocScience (RS2) was used for modelling the influence of mining operations. RS2 is a finite element modelling software that is used to effectively analyze stress deformation, stability, rock support design and staged excavation with ease. The staged construction of the mined stopes was excavated in the model as in the given task. The mined-out stopes were not backfilled in the model as they could bear little load, so they were left open as void in each of the staged construction. The failure zones were also outlined and excavated. The failure zones are named as collapse E1 and E2 respectively in the problem statement.

3.2 Model Setup

The 65º incline 6m width and 5 m of stages are defined in the model. The external boundary is placed such that the in-situ stress is not disturbed by excavation of the mining stope. The stope with cut slice of 5m high and 6m wide is made (Figure 3). For simplicity in model generation, the tabular ore body extends throughout the boundary of the model.

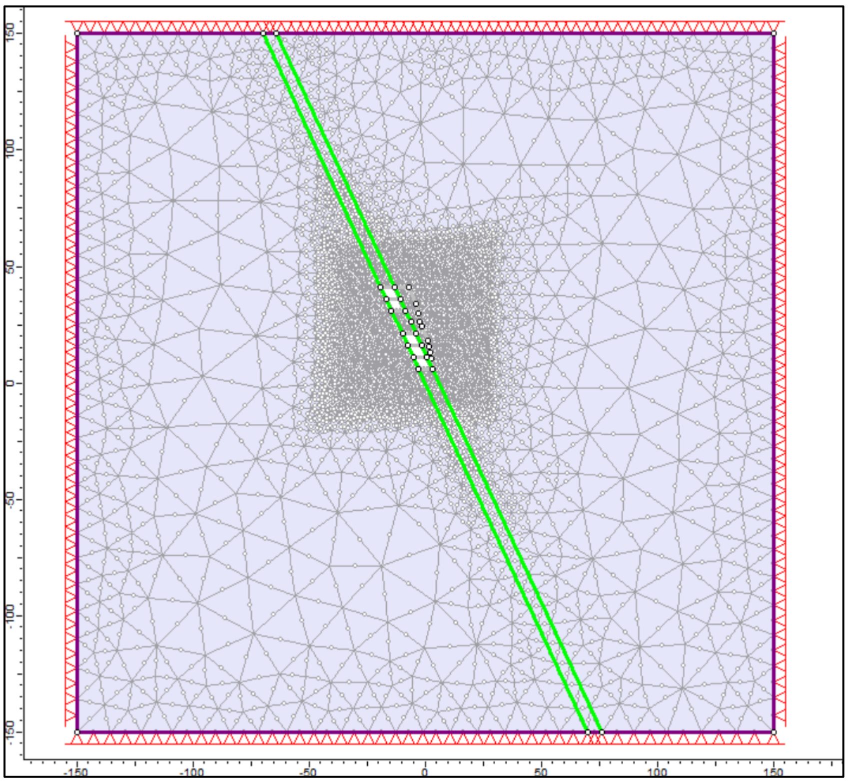

Figure 3: Model for numerical modelling.

3.2.1 Stage construction

<u>Stage 1:</u> Excavation of stope E1 is 25 m high, and it is located between 1025 to 1050 m. The stope is divided into five parts C1, C2, C3, C4 and C5 each 5m high. C1, C2 and C3 which were excavated by cut and fill method. C4 and C5 were decided to be excavated by rill mining method.

When C3 was excavated from stope 1, C1 and C2 were excavated from cut and fill method from stope 2.

Stage 2: Rill mining was done on stage 2 for stope 1. The rill mining was done for section C4 and C5 for stope 1. The depth of the C4 and C5 is 10m. They decided to excavate the blast at 3m. For this they drilled the blast holes from C3 towards C4 and C5. The blast created the retreat of 3m with every blast sequence. After 3 blasts, that is equivalent to 9m retreat backfilling of material from C1 to the blasted void was planned.

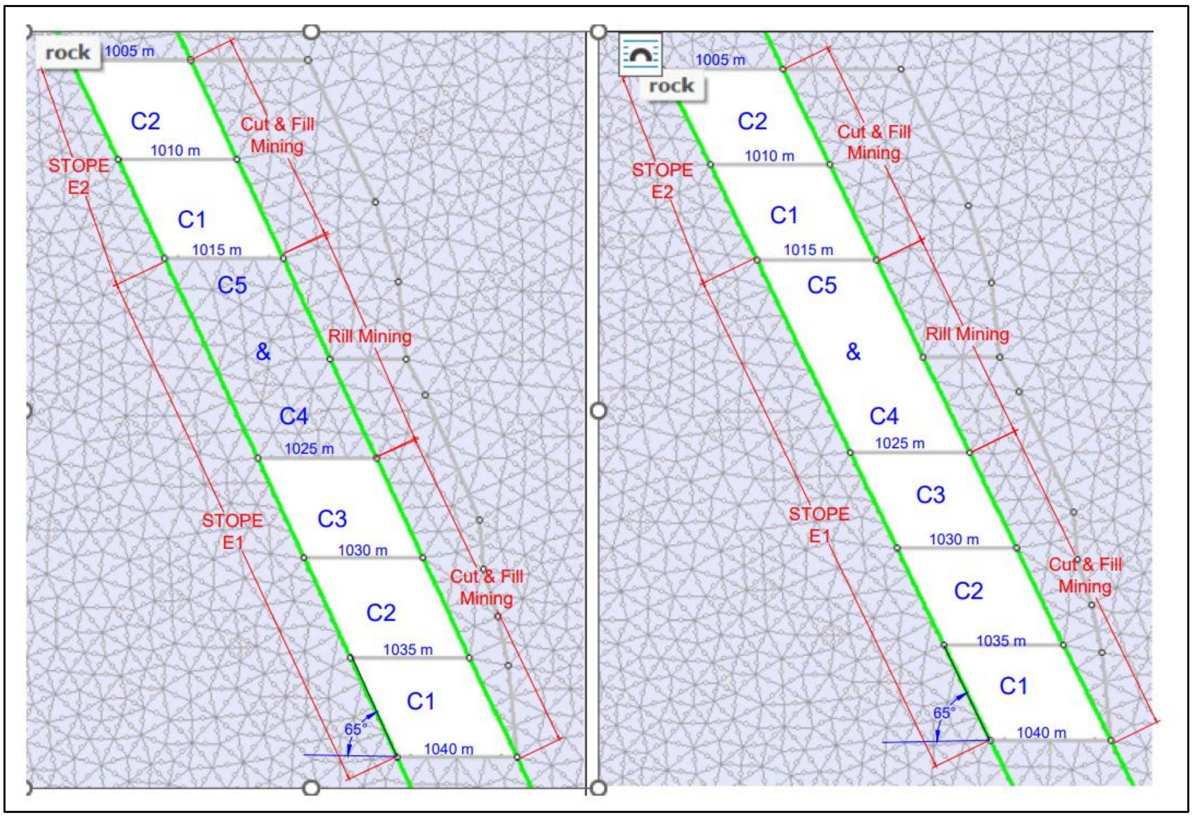

Figure 4: Stage construction of stage 1 and stage 2.

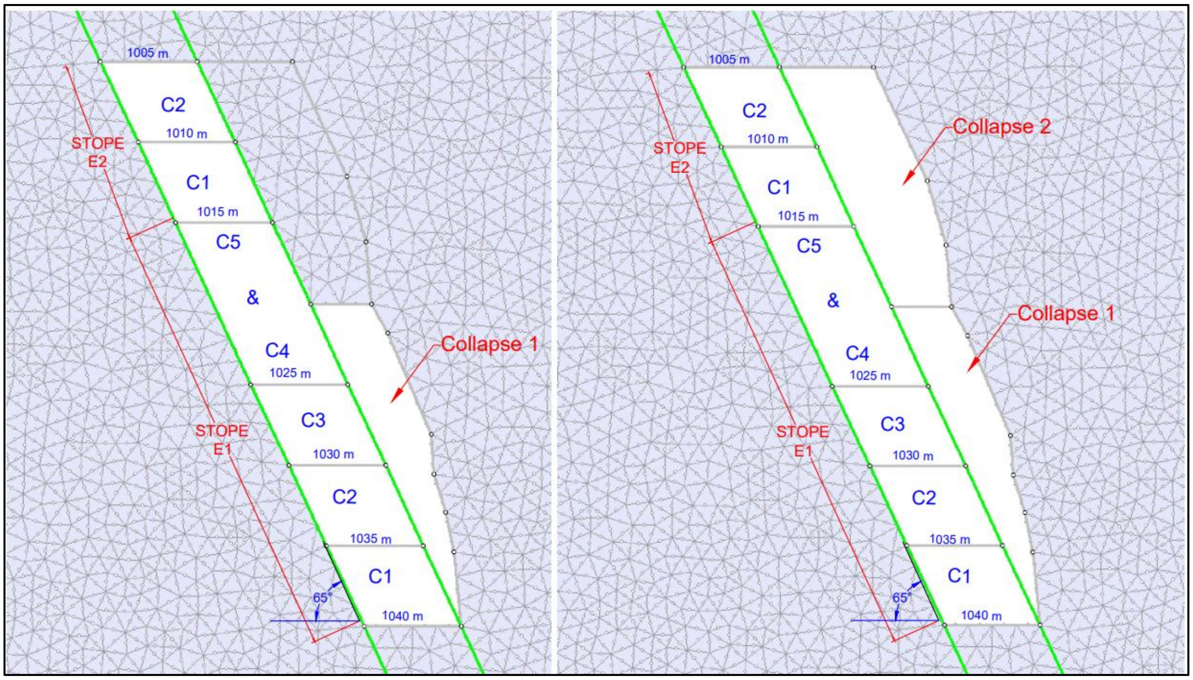

Figure 5: Stage construction of stage 3 and stage 4.

Stage 3: Even after the 27 m of retreat backfilling was not done according to the plan i.e after every 3 blasts (approximately after 9m of retreat), hanging wall was collapsed in the stope E1 as shown in Figure 5 at stage 3.

Stage 4: After the collapse 1 of hanging wall of stope E1 collapse 2 happened in stope E2. Figure 5 shows stage construction for stage 3 and stage 4 respectively.

3.2.2 Boundary Condition

The boundary condition is fixed at all ends because the in-situ stress is applied in all directions as constant type. So, all fixed boundary conditions are appropriate for this condition.

3.2.3 Meshing

6 noded triangular meshing is implemented in the model. The area near failure and the area around the mining area is meshed with higher resolution to optimize the computational time and increase the accuracy in stress modelling. Figure 6 shows the meshing setup in the model. Figure 3 shows the refinement of mesh around excavation zone.

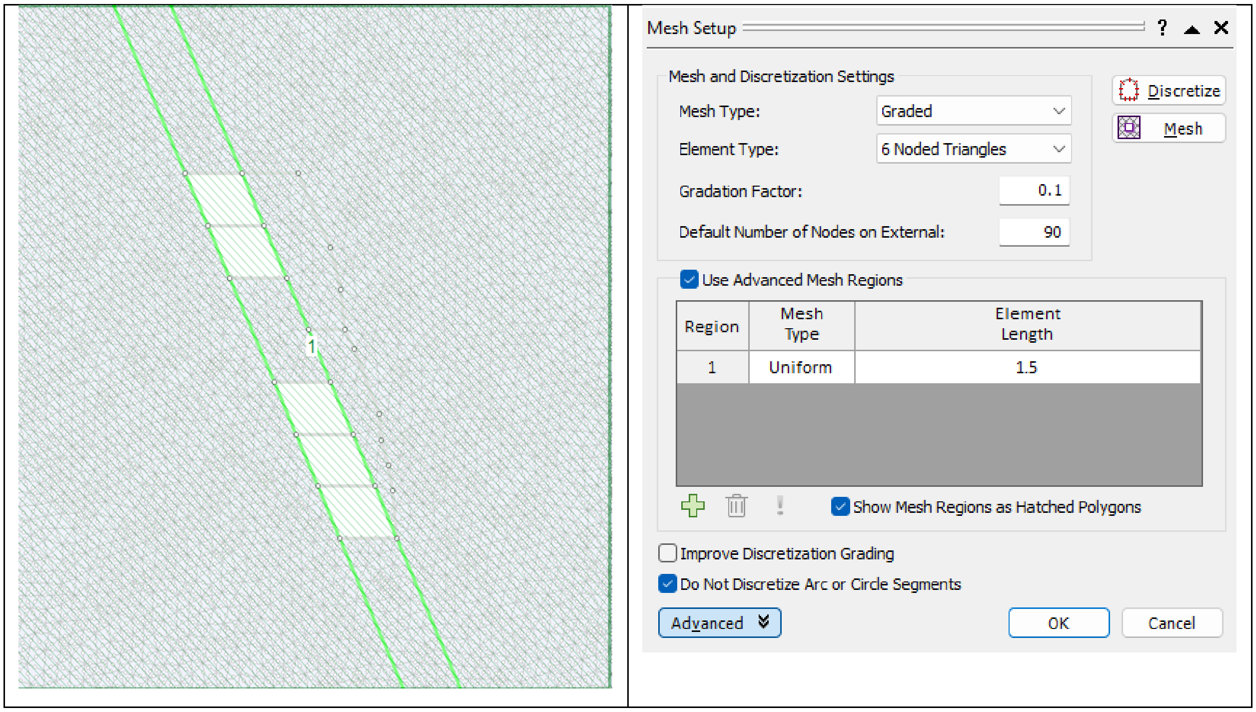

Figure 6: Mesh setup in model, the fine meshing is done around the failure zone.

3.2.4 Field Stress

Field Stress is inserted as constant stress. The in-situ stress is applied as calculated in Section 2.3. The assignment of field stress in RS2 is shown in Figure 7.

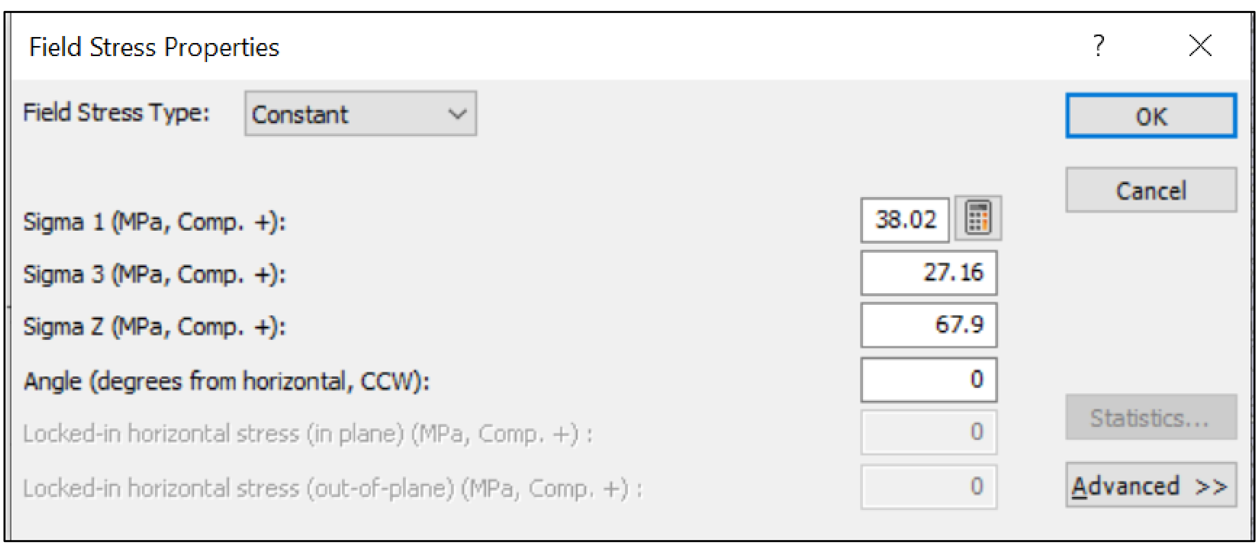

Figure 7: Assignment of field stress in model as constant type.

The geological data contains several information about the rock mass in the mined area. For simplicity in modelling, sericitic quartzite is considered as the rock mass of the area and the properties is taken from literature (Brandon, 1974).

The elastic property considers that the material will not fail, and the stresses are always uniquely related to strains. The analysis is governed by Young's modulus. As for plastic modelling, elastic modelling does not require many parameters, but still helps assess the excavation performance and potential zones of failure. Here, the strength factor of the material and the redistribution of the stress can be analyzed. It is also computational-time friendly (Hammah, 2022). Failure of the rock mass is defined by the depletion of load-carrying capacity of the material, and it can be predicted by different constitutive models and failure criteria.

In the analysis, Mohr-Coulomb criteria is used for analysis.

The material properties used for modelling are given below:

Poission's ratio: 0.3

Young's Modulus: 64000 MPa

Strength parameters

Frictional angle degree: 35°

Peak cohesion: 10.2 MPa

In our context, elastic modelling is used as we are interested in failure zones and not the actual deformation of failure.

3.3 Analysis and Interpretation

After constructing geometry and assigning all the required model properties, the model is analyzed and then interpretation is done according to the output of the model.

4 Results

The major principal stress distribution at different stages shows the redistribution of in situ stresses when the excavation progresses. The orientation and magnitude of in situ stresses changed significantly. There was stress concentration in the middle pillar, top and bottom of the openings. Tension was seen on the walls of openings (stope E1 and E2); however, it is not so severe (Figure 8).

When the cut of C4 and C5 of stope 1 was excavated by rill mining method, there was huge stress concentration in top and bottom of the opening. Large excavation area created tensile forces on the walls on stage 2 (i.e., after blasting by rill mining method). At this stage the headwall collapsed twice which is represented by stage 3 and stage 4 in Figure 10 and Figure 11 respectively.

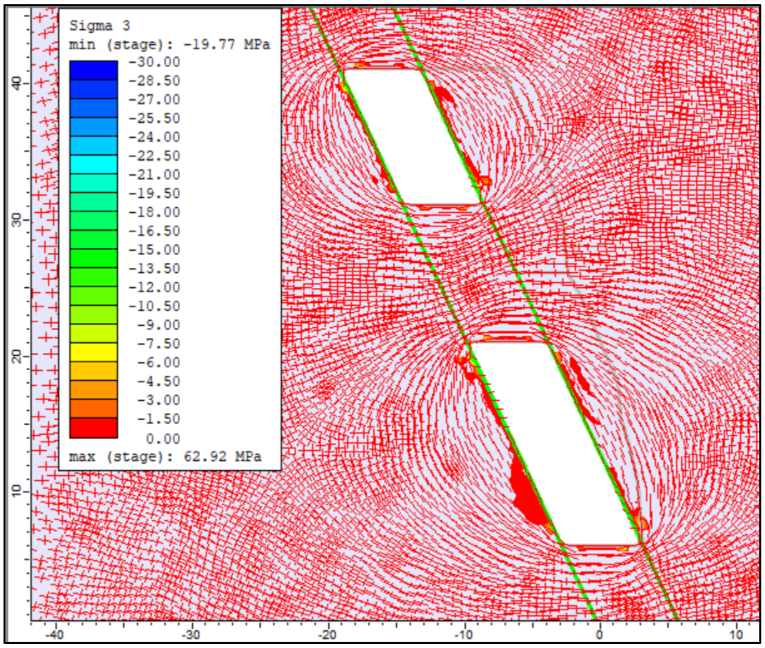

Figure 8: Tensile stress in stage 1

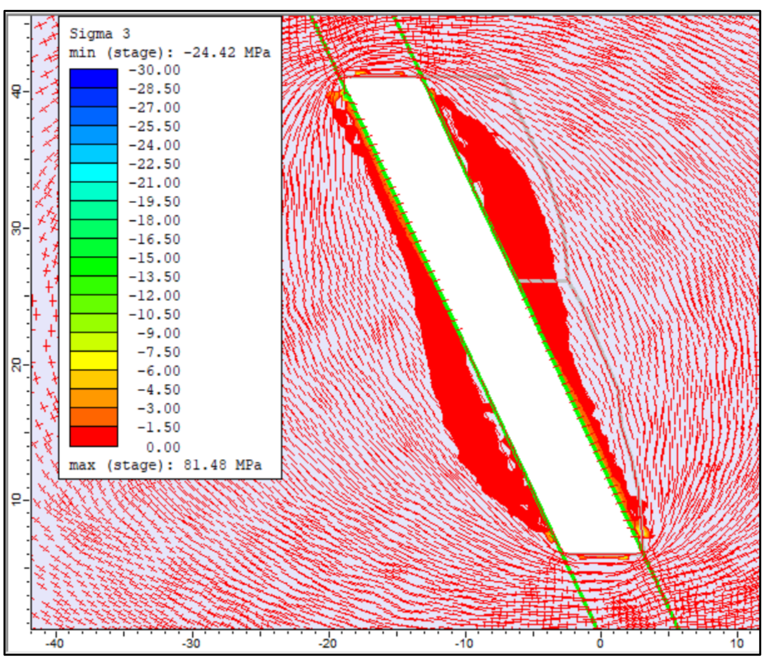

Figure 9: Tensile stress in stage 2

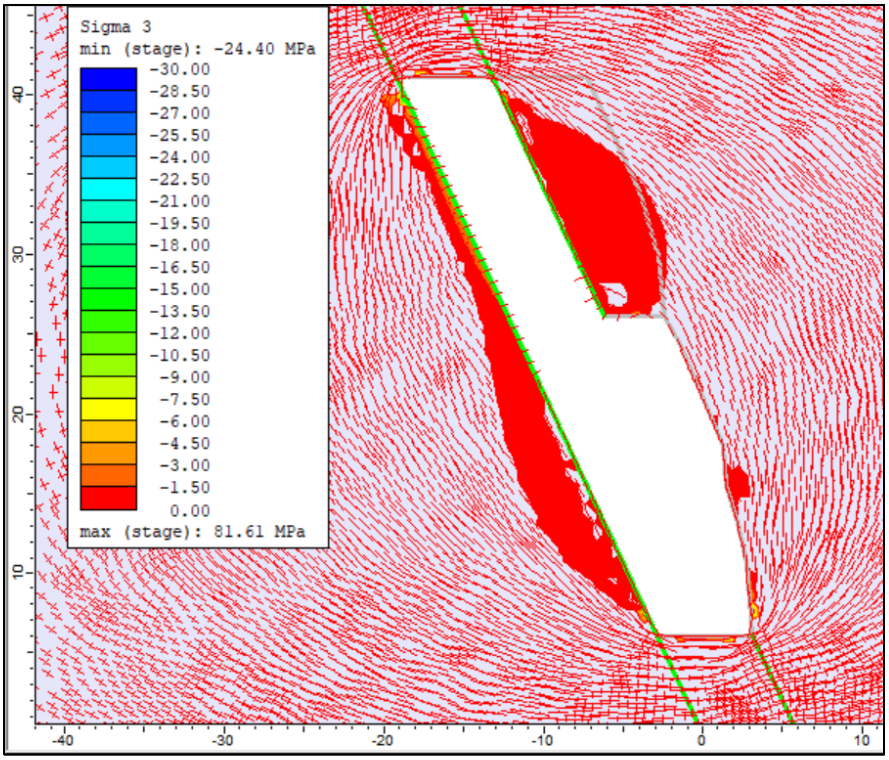

Figure 10: Tensile stress in stage 3

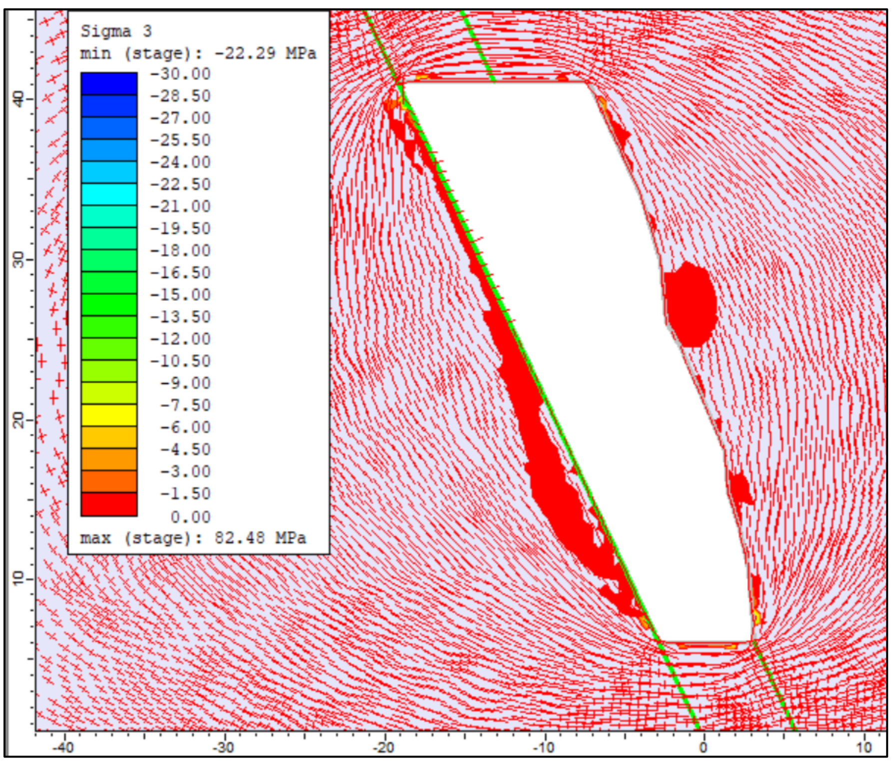

Figure 11: Tensile stress in stage 3

5 Discussion

5.1 Possible Cause of Failure

As the excavation goes on, the tangential stress around the excavation rock mass increases. At stage 1 in Figure 8 the failure zone is not so significant, and the stage seems stable. But when the ore is mined out between stope E1 and E2 by rill mining method, the failure zone extended extensively as shown by tensile stress in Figure 9. The development of tensile stress is believed to be a major cause of failure.

Where there is more open space, the tangential stress may be more than strength of rock mass, and this may be the cause of failure. Additionally, a change in blasting and filling plan might have contributed to the failure of headwall as the retreat without backfilling and suppor of 27 m is longer distance than preplanned approximately 9m of retreat. In hard rock the failure mode is extensional (Li, 2016) and where there is high in situ stress in competent rock, the possibility of rock burst is also high. However, in our case, there is less possibility of rock burst and higher possibility of tensile force induced instability problem.

The foliation is also parallel to the ore body and dipping 65° with the horizontal. When no other joint system is considered, the direction of drive of excavation is not favorable with respect to the foliation plane. If the direction of excavation were perpendicular to the foliation plane, it would have been regarded as favorable direction. In case when foliation strikes parallel to the excavation axis, shearing along the discontinuous becomes dominant phenomena (Schubert & Mendez, 2017). The largest displacement can be observed perpendicular to the foliation due to opening of joints and bending of the layers. In our case, the failure has occurred in the zones parallel to the foliation on the walls.

When bolt was applied and the minimum tensile stress was analyzed, there was no change in the minimum principal stress even after bolt of 1m spacing and 3m length was used.

5.2 Recovery of ore from E1 and E2

Over break and failure due to stability issue are not desired in the type of deep mining at large depth. The overbreak and failure due to stress issues has extra contribution to the amount of broken rock that has to be mucked out of the high-speed developing tunnel (Ortlepp, 2001). Nevertheless, all the failed rock should be removed from the tunnel, and the head wall should be protected with rock support to secure it in its place. The rill mining method should be revisited and the cut and fill method which seemed appropriate in this case should be opted. The length of retreat after blasting should be assessed carefully and adjustment should be made so that the ore can be excavated efficiently.

5.3 Recommended rock support method

Literature (Li, 2021; Ortlepp, 2001) suggest that support with high yield ability/ deformability perform good in high stress condition than stiff and strong support.Similarly, end anchored bolt performs well than fully grouted stiff bolts. (Li, 2017) suggests that the bolt length should extend at least 1 m beyond the failure zone. The deformation should be monitored and if the deformation velocity is higher than 2mm/day, caution should be adopted as it is an indicator of instability (Li, 2016).

Another method is to build drift tunnels nd install cable bolts on the head wall. However, this method is expensive and takes time.

(Li, 2017) suggests the rock support installed in 4 layers. (1) Rock bolts installed sporadically or systematically, (2) containment support or surface retaining like meshes or shotcrete, (3) Cable bolting which extend beyond failure zone and (4) External support like steel sets. In the present case, Layers 1,2 and 3 are relevant in our case which might have helped in making headwall stable.

6 Conclusion

The hanging wall collapses were observed in the deep mine operation primarily induced from the deviation in the mining plan. It was notably due to rill mining without immediate backfilling. The RS2 numerical modelling showed that there is pronounced tensile stress at the headwall of excavation, resulting in collapses. To rectify these issues, it is advisable to reassess the mining method and change it to cut and fill again if possible and provide appropriate support before excavating longer sections.

7 Limitations of work

In this work, the input parameters for numerical modelling are based on relevant research. The numerical model is analyzed without considering the actual joint of the rock mass. The elastic model is used which is simple however doesn’t replicate critical aspects of material behavior, such as dilation and stiffness stress interaction (Hammah, 2022).

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