Numerical Analyses of Field Monitoring in Stope J 10-3 at Kristineberg Mine

David Saiang and Erling Nordlund
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SUMMARY

This report presents the numerical analyses of the field monitoring work conducted at the Kristineberg underground mine. The work is a component of a larger Vinnova funded project that focused on the interaction between rock support and rock mass in response to large ground deformations.

The Kristineberg mine has a long history of large ground deformation which consequently incites ground control problems for the mine. Over the years the mine has developed various mining techniques and backfilling procedures to manage this problem. In general the ground control problems at the mine are highly influenced by the wall rock geology. The wall rock, that is the footwall and hanging wall, comprise of highly altered chlorite schist, which are internally referred to as talc-schist. They very often occur as seams with thickness barely ranging from 0.1 m to as wide as 3.0 m. Coupled with high ground stresses the talc squeezes and slides into the stope if undercut by the excavation or either bends or bulges inwards when exposed but not undercut depending on the loading direction. The deformation magnitudes have often been reported to be in the order of 0.2 to 0.5 m and seldom up to 1.0 m. Conventional rock support system, consisting of fibre re-enforced shotcrete and rebar rock bolts, has regularly failed under these conditions.

The conditions described above made Kristineberg mine an ideal environment to study the interaction between rock support and rock mass in response to large ground deformation. Hence a field investigation was conducted at the mine in J-orebody in stope number J10-3, which is located at a depth of 1200 m. The monitoring involved; convergence measurements, total station surveys, bore-hole photogrammetry, damage mapping and instrument bolt measurements. The monitoring was conducted in Cut #4 of Stope J10-3. Two systems of rock support were utilised in stope investigated with the objective to evaluate the performance of the rock support systems. The support systems were: (i) shotcrete + rebar and (ii) shotcrete + D-bolt. The two rock bolt types, rebar and D-bolt, were utilized in alternating 5 m long rounds as the face advanced. Bolting was regularly spaced in a 1.0 m by 1.0 m pattern.

To study the ground deformation behaviour the geology of the stope was extracted from drill-hole data and stope face mapping and utilized in numerical modeling. First, the numerical models were simulated without ground support in order to study the deformation and failure characteristics of the rock mass around the stope. Next, the rock supports were installed and the response of the rock support to the observed ground deformations were analysed. Apart from the test stope specific models (i.e., J10-3 models), additional models were developed and simulated for various geological scenarios as observed throughout the mine.

The numerical models revealed all the typical mechanisms of instability that have been conceptualized through observations and earlier studies. Talc obviously was the most
influential lithology that controlled the deformation characteristics of the stope and ultimately on the rock support system. Combinations of bending, bulging, shearing and tensile mechanisms induced a complex loading pattern on the rock support system. Often the rock bolts, for example, would experience all of these mechanisms at once or during different stages of the excavation rounds as the cut is developed.

It is difficult to make precise conclusions regarding the performance of the two rock bolt types utilized in the test stope. This is due to two reasons: (i) the instrumented bolt measurements were quite unsuccessful due to many of them failing prematurely and (ii) the deformation magnitudes experienced in the test stope were much smaller than expected and thus inducing less strain on the rock bolts. Nevertheless, the convergence measurements on the bolt heads does indicate to some extend that, the D-bolt, being ductile, appears to relax with increasing deformation, while the rebar bolt being stiff does not. This observation is made at least after the cut has advanced more than 30 m from the point of measurement, by which time the stope convergence has settled.
1 INTRODUCTION AND BACKGROUND

1.1 Definitions

Some terminologies related to the mining method in test stope have been consistently used in this report and therefore it is necessary to define them before proceeding. In the J-ore zone the cut-and-fill mining method is used. The main terminologies, used repeatedly in the report are; stope, cut and round (or slice). The definitions of these terminologies are illustrated with the help of Figure 1-1 and the following are the descriptions, including those of the geology.

**Stope** – A stope at Kristineberg is approximately 50 to 60 m high. The stope is mined using the cut-and-fill method, where each cut in the stope is backfilled soon after being mined out. A stope consists of up to 10 cuts.

**Cut** – The stope is divided into cuts; each cut is on average 6 m high. A cut is approximately 150 m long and is accessed through the middle via a ramp and cross-cut.

**Round** – A cut is developed horizontally along the strike of the orebody in 5 m rounds until the cut is completely mined out. In this report the term slice has also been used to mean the same.

**Chlorite schist or talc schist** – Weak layer of highly altered schist, it is clayey, greasy, soft and brittle. It can be crumbled with fingers when wet, but it is very stiff when dry. Other terms have also been used: talc seam, talc zone and talc layer. It is observed that the talc schist on the footwall side is often weaker than that in the hangingwall side.

**Chlorite quartz** – clay altered moderately weak chloritic quartz. This lithology frequently accompanies the talc schist layers, that is, they often found adjacent to the talc-schist.

**Cordierite quartz** – Competent host rock surrounding the orebody.

**Stope J10-3** – the stope number in which the measurements were conducted. In the mine block models the stope J10-3 is also identified with the number 091005.
1.2 Aims of the investigation and analyses

The Kristineberg mine investigation and analysis has two main objectives: (i) to increase the understanding of the interaction between the rock support and the rock mass in highly ground conditions or weak rock and (ii) to evaluate the performance of the ground support under large ground deformation conditions. These also the objectives of the main Vinnova project.

1.3 Constrains

A number of issues were encountered during the field investigation which ultimately constrained the data analysis and interpretation as well as numerical modelling and analysis of the monitoring data. First, the deformations observed in the test stope were much smaller than expected, thus making it difficult to evaluate the performance of the rock support under large deformation conditions. Second, majority of the strain gauges on the instrumented bolts failed to return reliable results for various reasons, including disruptions and damage. Third, an unexpected shift in cut placements within the test stope which tend follow the ore veins, had apparently inhibited ground movement.

For numerical analyses the constrains include; (i) difficulty in establishing the correct geology for the numerical models, (ii) complex geometrical shapes of the cut rounds, coupled with
unknown geology disabled any efforts for three-dimensional numerical models, (iii) the D-bolt, could not be modelled with UDEC and Phase\textsuperscript{2} using the usual approaches for ground support modelling with the two codes.

1.4 Kristineberg mine
1.4.1 Geology

The Kristineberg deposit is hosted in a zone of chloritized and sericitized quartz and feldspar-rich acidic volcanics. The ore zone strikes E-W and the dip varies between 45\textdegree{} and 70\textdegree{} S, and plunging approximately 45\textdegree{} SW. The host rock is a schistose seritic quartzite which ranges from a strong quartzitic rock to a weak highly altered material close to the proximity of the orebody. Immediately bounding the orebody are highly weathered chloritic talc-schist, which vary in thickness from 0 to 3 m or more. The material is very weak, friable and greasy with schistocity. The contacts between the chloritic talc-schist and the ore body or the neighbouring competent rock are often planar with talc coatings with low friction. The contact may also be a shear zone of thickness up to 1 m or greater, which is composed of altered material including schist and sericitic quartzites.

The orebody changes in characteristics from hangingwall to the footwall plunge. Typically the ore in footwall plunge is weaker than in the hangingwall side as a result of increased geologic disturbances. One the hangingwall side the ore is relatively competent, leading to reducing strength gradient from hangingwall to footwall.

From a geotechnical standpoint, the specific geologic conditions of interest are:

1. Very poor quality, thin bands of chlorite schist occur between the orebody and fairly competent host rock;
2. Planar, very weak contact planes or altered zones occur adjacent to the orebody; and
3. The orebody is disturbed in some locations with chloritic schist impurities in the ore.

Figure 1-2 shows the main orebodies at Kristineberg. There are eight main main ore lenses:

- **A-orebody** which is historically the main ore. It consists of massive dominated sulphide lenses, extending from the surface to a depth of about 1200 m. The A-orebody has essentially been mined out.
- **B-orebody**, which also consist massive pyrite dominated sulphide lenses.
- **Einarsson-orebodies**, which consists of mainly Au-Cu rich veins and lenses of sulphides in hard quartzitic rock.
• K-orebody, J-orebody, M-orebody which consist of mainly Zn-Cu rich veins and lenses of sulphides.

• L-orebody, which is a continuation of the A4 ore lens. This orebody also consists of Zn and Cu rich veins of massive sulphides.

Present mining is concentrated in E, J, M, L and B-orebodies. Field measurements study were carried out in in the J-orebody or “J-malm” as in Figure 1-2.

![Figure 1-2: The orebodies at Kristineberg. Measurements were performed in the J-orebody, that is ‘J-malm’ in this figure.](image)

1.5 Mining method

Two mining methods are applied at the Kristineberg mine. The standard cut-and-fill method is applied for narrow orebodies and drift-and-fill for wide orebodies (greater than 8 m), see Figure 1-3. Typically the stopes are 150 m long, along the strike and, are developed from the centre and mined in each direction by breasting. Access to the centre of the stope is via a crosscut developed from the footwall ramp. The cuts are approximately 5 to 6 m high and 6 m wide, and are excavated in 5 m rounds by the drill and blast method.
1.6 Weak ore contact zone

The ground control problems at Kristineberg mine are primarily related to the geological conditions. The presence of the weak talc-schist in both footwall and hangingwall and the weak interfaces between the rock units result in failures in the roof and sidewalls in a pattern which is fairly common in the mine as conceptually illustrated in Figure 1-4. First the failure initiates within the weak talc-schist, which is followed by a slip along the contact if the contact is undercut by the stopes. The sequence of instability development can be described as follows.

As a stope is excavated the relatively hard ore in the roof is subjected to a complex loading and unloading pattern. A high stress concentration occurs parallel to the roof and at the same time a stress relief occurs perpendicular also to the roof. Consequently the ore fails in extension and fractures into relatively thin plates parallel to the free surface of the roof. On the other hand, a form of foundation failure is also apparent as the ore roof punches into the weak side walls which results in squeezing the altered materials of the sidewalls into the stope. The downward shear of the sidewall rocks, in turn, drags roof ore downward and pulls it apart along its fractures. This ends in fallouts of blocks from the roof along its contact to the altered zone. When the sidewalls are relatively competent the shear failure mechanism is either less dominant or non-existent and thus the walls deform in bending mode.

Various authors have tried to conceptualize and explain the typical failure modes or patterns at Kristineberg. Figure 1-4 shows one examples of these, which shows, typically, shear failure in the weak footwall, bending of the hangingwall and, fracturing and separation of the roof slabs. Various mechanisms are responsible for these failure patterns as illustrated in Figure 1-5 (A). Some failure patterns are complicated due to complex geology as illustrated in Figure 1-5 (B). The problem with this kind of complexity is that the deformation is unpredictable and hence
the approach for support design becomes difficult.

Figure 1-4: Conceptualized failure characteristics and mechanisms at Kristineberg mine (Board et al., 1992)

Figure 1-5: (A) Failure patterns and mechanisms associated with weak contact zones (Board et al., 1992). (B) A complex geology, where a shear zone intersects the orebody leading to a large zone of chlorite schist (Board et al., 1992).
2 FIELD MEASUREMENTS

Reporting of the field monitoring and analysis component of project has been covered in separate report by Westblom, Perez and Nordlund (2012). In this report the summaries of the monitoring and analyses are presented to aid the numerical modelling and analysis component, which is the focus of this report. However, the authors’ version of the description of field monitoring and analysis is also presented as Appendix A to provide additional information that aided the interpretation of the numerical modelling results.

2.1 Test Area

Field measurements were carried out in Cut #4 of Stope J10-3 at level 1195 m in the J-ore zone. Figure 2-1 shows the J-ore zone (circled) among the different orebodies at Kristinerberg, while Figure 2-2 shows the test area. Stope J10-4, also shown in Figure 2-2, is above Stope J10-3. Additional measurements have been conducted in Stope J10-3 cut#5 and J10-4 cut #1 but as a separate project.

Figure 2-3 shows additional details of cut #4 of Stope J10-3 – the test area. The vertical profiles through Y825, Y850 and Y875 show the vertical stope profile from cut #1 to cut #4. Labelled R1 to R10 are the excavation rounds, each round is 5 m long hence the monitored section was 50 m long.

![Figure 2-1: The orebodies at Kristineberg. Measurements were performed in the J-ore zone.](image-url)
Figure 2-2: Stope J10-3 Cut #4 is the test area for the analysis presented in this report. Measurements in J10-3 cut #5 and J10-4 cut #1 were conducted for another project. Y825, Y850 and Y875 are sections through which drill-hole geology was obtained.

Figure 2-3: Stope profiles through sections Y825, Y875 and Y850. Monitoring was conducted in Cut #4 for the analysis presented herein.
2.2 Instrumentation and Monitoring

The 50 m long cut was monitored using extensometers, total station, strain-gauged rockbolts and borehole camera survey over a period of approximately 50 days. The extensometers and total station monitored the ground deformation, while the strain gauges in the rock bolts monitored the deformation along the length of the rock bolts. However, majority of the strain gauges failed during the monitoring and therefore the measurements of strains on the bolts were not successful. The layout of the instrumentations is as shown in Figure 2-4 with Table 2-1 showing the instrumentation density. Instrumentation sections are located in the middle of each round. Sections where the instrumentation is intensified are referred to high density profiles, which are specifically aimed at monitoring the interaction between the rock bolts and rock mass. Profiles S3, S4, S6 and S7 are the high density profiles. Profiles S3 and S7 are supported with rebar rock bolts while S4 and S6 with D-bolt. Profiles 3:4 and S6:7 are deep-hole extensometer sections. On either side are rounds supported by rebar and D-bolt. Appendices A to E show the details of the instrumentation for the low and high density profiles, deep hole extensometer placement profile and bolt instrumentation.

To avoid damaging and shocking the instruments the blast rounds advanced two rounds (or 10 m) ahead of the instrumentation. This means instrumentation began from section 1 (S1) after round 2 (R2) was excavated and monitoring began with round 3 (R3).

Each alternate round was supported with the rebar and D-bolt; odd numbered rounds with rebar and even numbered rounds with D-bolt. Shotcrete of varying thickness of 10 to 15 cm was applied following the rock bolt installations.
Figure 2-4: Monitoring plan. R1 to R10 represent the 5 m cut rounds, S1 to S10 represent the instrumented sections (after Perez and Nordlund, 2014).
Table 2-1: Bolting and instrumentation profiles for the excavation rounds. Sections 3-4 and 6-7 (shaded) are high density profiles by virtue of the amount of instrumentation.

<table>
<thead>
<tr>
<th>Measurement Section</th>
<th>Bolt type installed</th>
<th>Monitoring method</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Rebar</td>
<td>D-bolt</td>
</tr>
<tr>
<td>S1</td>
<td>X</td>
<td>X</td>
</tr>
<tr>
<td>S2</td>
<td></td>
<td>X</td>
</tr>
<tr>
<td>S3</td>
<td>X</td>
<td>X</td>
</tr>
<tr>
<td>S4</td>
<td>X</td>
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</tr>
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<td>S7</td>
<td>X</td>
<td>X</td>
</tr>
<tr>
<td>S6:7</td>
<td></td>
<td>X</td>
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<tr>
<td>S8</td>
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<td>X</td>
</tr>
<tr>
<td>S9</td>
<td>X</td>
<td>X</td>
</tr>
<tr>
<td>S10</td>
<td>X</td>
<td>X</td>
</tr>
</tbody>
</table>

2.3 Monitoring Results

Detailed analyses of the Kristineberg field measurement can be found in the report by Perez, Westblom and Nordlund (2012). The results presented herein are not intended to replicate their analysis. Therefore the aforementioned report is recommended for detailed understanding of the analyses. Nevertheless, abstracts from the above report are presented herein to make the current report complete, with additional inputs by these authors for further clarifications.

2.3.1 Convergence measurements

The convergences of cut rounds were monitored by tape extensometers and total station measurements. The tape extensometers measured the horizontal convergence of the footwall (FW) and hangingwall (HW), while the total station measurement provided a full convergence profile of the cut rounds. However, only the tape extensometer measurements covered the full span of the investigated length of Cut #4, while the total station measurements were only done in sections 2, 3, 6 and 7, which are considered as high density profiles (see Figure 2-4 and Table 2-1). Note that, all measurements, both total station and tape extensometers, were done on bolt heads.

2.3.1.1 Tape extensometers
The results of the tape extensometer measurements of the HW-FW convergence are shown in Figure 2-5. These convergences were achieved with ground support, i.e. shotcrete and rock bolts, installed. Furthermore, it is estimated that, approximately 50% of the convergence has occurred prior to support installation. This implies that, the convergences shown in Figure 2-5 are approximately 50% of the total convergence. The maximum horizontal convergence in the 50 m long cut was approximately 88 mm, which was measured in section 2 (S2), which is again approximately 50% of the total convergence prior to support installation. However, it must be stated that, this was a very odd and unexpected ground deformation magnitude for Kristineberg, since deformations of up to 0.5 m are common in Kristineberg.

Figure 2-6 shows the maximum convergences with respect to the two rock bolt types used. There is no clear distinction between the convergences observed in the rebar and D-bolt supported rounds. Although there are slightly higher convergences in the D-bolt supported rounds in the first 4 rounds, the next 6 rounds showed lower convergences compared to the rebar supported rounds. The D-bolt supported rounds were expected to deform more than the rebar supported rounds, because the D-bolt is more ductile than the rebar. The lack of distinction is probably attributed to two factors:

- that the ground deformation was too small to affect the deformation characteristics of the two rock bolts and,
- that there is a delay in ground deformation behind the advancing face.

In fact, it can be seen in Figure 2-4 that, the deformations appear to converge after more than 10 days, which corresponds to the 5th or the 6th cut rounds (R5 or R6). Since each round is 5 m long, this would imply that the maximum convergence is experienced approximately 25 to 30 m behind the advancing face. This would mean that, convergences measured in S1 to S5 (or R1 to R5) are indicative of the complete convergences, since they located more than 30 m behind the advancing face, while S6 to S10 (or R6 to R10) are located less than 30 m from it and therefore the convergences would be partial. Hence, to compare the performance of the D-bolt and the rebar, the measurements from S1 to S5 would be representative. From Figure 2-6 it can be seen that between sections S1 to S4 the D-bolt deformed more than the rebar, which in an ideal situation, would confirm the expectation that the D-bolt should deform more than rebar. As the face advances the load in the rock bolts should successively increase, causing the rock bolts to deform according to their designed characteristics, in this case the rebar is stiff while the D-bolt is ductile.
Figure 2-5: Tape extensometer profiles. Note: the odd number alternative sections are within rebar supported rounds, while even number alternative sections are within D-bolt supported rounds.

Figure 2-6: Maximum HW-FW convergence with alternate rebar and D-bolt supported rounds.

2.3.1.2 Total station

The total station convergence profiles of the sections S2, S3, S6 and S7 or high density profiles (see Table 2-1) are shown in Figure 2-7. Although three-dimensional movement of the bolt heads in X, Y, Z vectors are complex to represent in two-dimensional profile, the pattern of deformation is however obvious. The deformation vectors in Figure 2-7 are relative to initial
positions of the bolt heads. However, the resulting deformation profiles have been exaggerated with respect to the excavation profiles in order to visualize the deformation pattern.

In sections S2 and S3 large movements are observed in the hangingwall (HW). In the footwall (FW) side large movements are observed in the lower half of the FW. The deformations are consistent with the locations of the weak talc and moderately weak sericitic quartz zones. In the HW side the moderately weak sericitic quartz has been significantly under-cut, while on the FW side the talc is full exposed up to FW mid-height.

The deformation in the HW roof and shoulder is less than in the middle and lower half of HW. Logically it would be expected that large deformations should occur in roof and shoulder given the location of the talc and undercutting by the stope. However, this is not case. In fact the observations confirm the earlier conclusions by Board et al. (1991) that, failure initiates from lower and middle half of the HW and FW and propagates into roof. Therefore, the middle and lower half of the HW and FW should show larger deformations than the upper half of the FW and HW. In fact, this confirms the logic for performing the tape extensometer measurements at lower the half of the stope walls, since large deformations were expected there.

In sections S6 and S7 large deformations occur both in the FW and HW. In the FW, the lower half appears to swell more than the upper half. This is consistent with the fact that the lower half is closer to the inclined talc seam than the upper half. In the HW the talc seam has been partially under-cut, consequently resulting large deformations.
Figure 2-7: Accumulated displacement profiles of the sections from total station measurements. The bolt heads served as measurement points (after Perez et al., 2012).

2.3.2 Borehole camera

Boreholes were drilled in all cut rounds in the HW and FW to monitor ground movements as the cut face advanced. The boreholes were routinely inspected and photographed during the monitoring period. Some boreholes showed significant movement behind the stope walls, while some showed no closure or collapse. Those that showed significant movements and eventual collapse were those experiencing significant shearing of the talc seam.

Some holes barely showed any closure, although loose rock materials and sometimes slight narrowing of the boreholes were observed. It is believed that, in such cases movements may be normal to the stope wall, typically due to bending; in which case the differential movements would be too small to affect the bore holes, except in the case of excessive bending and swelling.

Figure 2-8 to Figure 2-10 shows the boreholes where significant movements have occurred. Figure 2-8 shows borehole closure in the HW in section S1. The closure is apparently normal
to the borehole, which is consistent to the excessive bending and bulging of the HW in section S1. Figure 2-9 shows shearing of the talc seam in FW in section S2. A 13 mm shear movement was observed on Day 9 of the monitoring, which corresponded to the excavation of round R4. The shear movement increased to a likely maximum of 31 mm on Day 28 of the monitoring, which corresponded days after excavation of round R10. After 42 days the shear movement remained at 31 mm indicating the movement has reached equilibrium. Figure 2-10 shows shear in the HW between sections S5 and S6. The shear movement in this hole has reached a maximum of 45 mm in Day 28 but the camera could reach to a depth of 0.66 m. On the Day 42 the camera could reach up to depth of 0.49 m, since further behind the hole is apparently closed. A face mapping done on Day 35 between sections S5 and S6 showed significant damage to the shotcrete lining in FW, consistent with the observations in the boreholes and total station profiles for section S6 shown in Figure 2-7.

Figure 2-8: Observations in section S1: (A) Borehole camera shots of the borehole closure principally due to bending and swelling of the HW, (B) HW-FW convergence in S1 resulting (after Perez et al., 2012)
Figure 2-9: Observations in S2/Profile: (A) Camera shots of the shear movement in the FW side observed during the duration of the monitoring (B) HW-FW convergence in S2 (after Perez et al., 2012)
Figure 2-10: Observations in between S5 and S6: (A) Camera shots of the shear movement in the FW side observed during the duration of the monitoring (B) Failure mapping showing damage to shotcrete liner between S5 and S6 due to shear movement in the FW.

2.3.3 Failure mapping

Surface failure mapping by visually assessing and recording damages to shotcrete and failure of rock bolts were carried out during the monitoring period. Although, majority of the rockbolts remained intact, the shotcrete showed cracking as the faced advanced. Cracks were immediately evident the face advanced 5 to 6 rounds ahead.
3 GEOLOGY OF THE TEST AREA

In order to model the rock mass behaviour and rock support response as accurately as possible capturing the geology of the test area was necessary. The geology data was captured mainly from the drill-holes that intersected the test stope and from face mapping campaign as the stope was developed.

3.1 Drillhole data

A large cluster of drillhole data was provided by Boliden, which however proved difficult to extract any useful geological information. The most useful drillhole information are those specifically extracted from the drillhole intersections that crossed the test area (i.e. Stope J10-3) at Y825, Y850 and Y875 (see Figure 3-1). Figure 3-2 shows the lithologies intersected by of the drillholes.

![Figure 3-1: Drillhole intersections were obtained for profiles Y825, Y875 and Y850.](image-url)
3.2 Face mapping data

Face mapping of Stope J10-3 has also been carried out by the Kristineberg mine geologist. These mapping were done for the roof and advancing faces. Figure 3-3 shows the geology resulting from the roof mapping in cuts #3 and #4, while (b)

Figure 3-4 shows the geology resulting from mapping of the advance faces from cut #1 to cut #6 through profiles Y850 and Y900.
Figure 3-3: Geology from roof mapping in Stope J10-3 at (a) Cut #3 and (b) Cut #4.
Figure 3-4: Geology from face mapping of the cuts through (a) Y850 which pass through the middle test area and (b) cross-section Y900 is in the opposite stope (see Figure 3-1).
3.3 Simplified geology for numerical models

A simplified geology of the test area (i.e. Stope J10-3) was constructed by combining the drillhole and stope face mapping data. These are shown in Figure 3-5 to Figure 3-7. The geology of the rock mass immediately around ore (i.e. the wall rock) is very important from rock support interaction perspective and therefore attempt was made to define the local geological as properly as possible. The far field geology is less important for impacting the interaction around excavation is therefore simply interpreted as the main host rock. The drillhole profiles, shown in Figure 3-1, clearly show significant variation in geology near the vicinity of the orebody, which is consistent with the history of the ore formation process. In the far field the rock mass is fresh and therefore geology is that of the fresh host rock.

The wall rock lithology mainly comprised three units; the cordierite-quartz (which is the host rock), the chlorite-quartz and the chlorite-schist (also referred to as talc-schist). The lithological units are highly sericitized with increasing intensity of sericitization from the hangingwall to the footwall. As a result the chlorite-schist on the footwall side appears to be highly altered into talc or clay and is therefore much weaker than when it appears in the hangingwall. Figure 3-8 shows as example of the talc exposure in the footwall. The contact between talc-schist and other units are typically defined by clay-filled, slicken-sided smooth surfaces, where slip can occur with very little resistance.
Figure 3-5: Projected geology through profile Y825.

Figure 3-6: Projected geology through profile Y850.
Figure 3-7: Projected geology through profile Y875.

Figure 3-8: Occurrence of talc in the footwall in Stope J10-3.
4 NUMERICAL ANALYSIS OF FIELD MONITORING

4.1 Unsupported models

4.1.1 Descriptions

Numerical analyses were performed for sections through profiles Y825, Y850 and Y875 (Figure 4-1) with their respective geology shown in Figure 3-5 to Figure 3-7. For the unsupported models no ground supports were applied. The objective was to study the ground deformation behaviour under unsupported conditions.

The inputs used in the numerical models are shown in Table 4-1 and Table 4-2, where those in Table 4-1 are based on previously calibrated values by Board et al (1991) and refined through parametric analysis by Elhami (2012). For the backfill the parametric values reported by Knutsson (1981) were used. The interface properties have been estimated using empirical relations and model calibrations.

The in-situ stresses are those from hydraulic fracturing measurements reported by Stephansson as:

Board et al (1992) used the in-situ stresses obtained by Leijon (1990) using over-coring methods. However, it is now accepted that the in-situ stresses obtained from hydraulic fracturing measurements are more representative of Kristineberg than those obtained using over-coring method. Hence, the in-situ stresses reported by Stephansson (1993) based on hydraulic fracturing measurements were used, where are given as:

\[ \sigma_v = 0.027Z \]
\[ \sigma_H = 2.8 + 0.04Z \]
\[ \sigma_h = 2.2 + 0.024Z \]

Where; \( \sigma_v \) is the vertical stress, \( z \) is the depth, \( \sigma_H \) is the maximum principal stress and is parallel to the orebody, and \( \sigma_h \) is the intermediate principal stress and is perpendicular to the orebody.
Figure 4-1: Modelled profiles with their respective model shown Figure 3-5 to Figure 3-7.

### Table 4-1: Material parameters for numerical analyses

<table>
<thead>
<tr>
<th>Material</th>
<th>Youngs modulus, $E$ (MPa)</th>
<th>Poissons ratio, $\nu$</th>
<th>Rockmass compressive strength, $\sigma_{cm}$ (MPa)</th>
<th>Cohesion, $c$ (MPa)</th>
<th>Friction, $\phi$ (°)</th>
<th>Tension, $\sigma_t$ (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cordierite-quartzite</td>
<td>39000</td>
<td>0.25</td>
<td>23.2</td>
<td>6.7</td>
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<td>0.40</td>
</tr>
<tr>
<td>Chlorite-quartzite</td>
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</tr>
<tr>
<td>Talc-schist</td>
<td>15000</td>
<td>0.30</td>
<td>5.2</td>
<td>2.0</td>
<td>15</td>
<td>0</td>
</tr>
<tr>
<td>Orebody</td>
<td>27000</td>
<td>0.25</td>
<td>15.1</td>
<td>5.0</td>
<td>23</td>
<td>0.40</td>
</tr>
<tr>
<td>Backfill</td>
<td>81</td>
<td>0.35</td>
<td>0.7</td>
<td>0</td>
<td>36</td>
<td>0.04</td>
</tr>
</tbody>
</table>

### Table 4-2: Talc-schist interface properties

<table>
<thead>
<tr>
<th>Normal stiffness, $k_n$ (GPa/m)</th>
<th>Shear stiffness, $k_s$ (GPa/m)</th>
<th>Cohesion, $c$ (MPa)</th>
<th>Friction, $\phi$ (°)</th>
<th>Tension, $\sigma_t$ (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>6000</td>
<td>60 GPa</td>
<td>0</td>
<td>5</td>
<td>0</td>
</tr>
</tbody>
</table>
4.1.2 Results

The results for the unsupported stopes through profiles Y825, Y850 and Y875 are shown in Figure 4-2, Figure 4-3 and Figure 4-4, respectively.

In profiles Y825 and Y850 the talc occurs both in the HW and FW. In the HW the talc seams have been undercut, leading to significant ground deformation in the HW. In the FW the deformation is small since the talc seams are located further behind the FW. Total station measurement in Section 6 (S6), located in Round 6 (R6), inserted in Figure 4-3, confirms the deformation characteristics observed in the numerical simulation through profile Y850. No total station measurements were available for profile Y825. Camera images through the borehole located in the FW, between S5 and S6 (i.e. between R5 and R6), show significant shearing of the FW talc (see Figure 4-3 (b)).

In the profile Y875 the talc occurs in the FW and also through the middle of the orebody. The talc seam in the middle of the orebody is fully undercut by the stope, while the seam in the FW is partially undercut. The deformation of the talc in profile Y875 occurs in the form of shearing, sliding and squeezing. The total station measurement of Section 3 (S3), located in Round 3 (R3), inserted in Figure 4-4, is the closet total station measurement to profile Y875, since none was available for Section 1 (S1) or Round 1 (R1). The inserted total station measurements for S3 shows similar deformation pattern as observed numerically in profile Y875. In fact, the talc seam that is located on the FW of Y875 appears to extend to S2 and S3. This is clearly evident from the fact that, the borehole camera images in the FW at S2 show significant shearing within the talc (see Figure 4-4 (b)).

The numerical simulations of the unsupported stopes show a number of mechanisms involved in deformation of the talc including; bending, shearing and dilation. These mechanisms are important to know as they will ultimately affect the response of the rock support elements.
Figure 4-2: Ground displacement characteristics around profile Y825, Cut #4. Large deformations occur in the hangingwall where the talc-schist seam has been undercut.
Figure 4-3: (a) Ground displacement characteristics around profile Y850, Cut #4. Large deformations occur in the hangingwall where the talc seam is closest to the stope. Inserted on the top left is the total station profile through Section 6, a section where Profile Y850 crosses. (b) Shearing of the talc observed in the FW in Section 6 (S6) via borehole camera.

Figure 4-4: (a) Ground displacement characteristics around profile Y875, Cut #4. Large deformations occur in the roof and the footwall where the talc-schist seams has been undercut. Inserted on the top left is the total station profile of Section 3 (S3), which is the nearest section to profile Y875 where the total station measurement was made available. (b) Shearing of the talc in FW in Section 2 (S2) observed through the borehole camera.
4.2 Supported models

4.2.1 Descriptions

Profiles Y825, Y850 and Y875 were also simulated with rock supports installed. Profile Y875 was supported with D-bolt and shotcrete, while profiles Y850 and Y875 were supported with shotcrete and rebar. The rock bolts are 2.7 m long and regularly spaced 1.0 m apart. Table 4-3 gives the properties of the rock bolts. The D-bolt has two components; the non-deformable anchors and deformable sanks. The D-bolt is considered a high strength-ductile rock bolt. Like the rebar the D-yields at 0.2% strain, however, while the rebar is considered to fail at yield, the D-bolt is reported to strain plastically up to 14% before failing (see Li, 2010).

In the numerical simulation the D-bolt is modelled in two parts; the non-deformable anchors are assigned very strength and modulus, while the sanks are assigned the reported strength and modulus of the D-bolt. However, to make the D-bolt strain plastically a residual strength equal to the yield strength is assigned.

| Table 4-3: Strength and stiffness parameters of the Rebar and D-bolt. |
|--------------------------|--------------------------|
|                          | Rebar                    | D-bolt                   |
|                          | Anchor                  | Sanks                    |
| Young’s Modulus          | 200 GPa                 | 200 000 GPa              | 200 GPa                  |
| Tensile Strength         | 200 kN                  | 100 000 kN               | 200 kN                   |
| Residual tensile strength | 0 kN                    | 100 000 kN               | 200 kN                   |

<table>
<thead>
<tr>
<th>Table 4-4: Fibre re-enforced shotcrete parameters used in the simulation (after Malmgren, 2005).</th>
</tr>
</thead>
<tbody>
<tr>
<td>Thickness (cm)</td>
</tr>
<tr>
<td>----------------</td>
</tr>
<tr>
<td>10 cm</td>
</tr>
</tbody>
</table>

4.2.2 Ground reaction curve

A ground reaction curve was created for the stope in order to assist in determining when the rock support should be installed in models. A simple horseshoe shaped excavation geometry was assumed, with simplified geology assumed to have the properties of the orebody. The excavation is located at the same depth as the real stopes being investigated.

The modulus of the stope is reduced in 50% increments until the stope is excavated and the resulting ground reaction curve is shown in Figure 4-5. The critical limit is reached when the rock mass modulus is reduced to about 25% of the original value, resulting in inward
displacement of about 60 mm. However, for the numerical modelling the 20% limit for modulus reduction is chosen. With this limit a convergence of 75 mm was obtained, which is when compared to the convergence measurement data, for example Figure 2-5, that approximately 50% of the convergence would have occurred before the ground support elements (shotcrete and rock bolts) were installed. Hence the excavation and support installation were executed in the following order:

Step 1: Reduce the modulus of material within the cut to be excavated to 20% of the original value and cycle the model to equilibrium.

Step 2: Excavate by cut, install rock bolts and shotcrete, and cycle to equilibrium

Step 3: Backfill and repeat Step 1 for next the cut and cycle to equilibrium

The above steps were began for Cut #1 and repeated all the way to Cut #4.

![Figure 4-5: Ground reaction curve for the stope. The critical limit is reached when the modulus is reduced to 25%. For modelling 20% limit for modulus reduction is chosen.](image)

4.2.3 Results of the supported models

Figure 4-6, Figure 4-8 and Figure 4-10 show the results from the supported stope models, while Figure 4-4, Figure 4-7 and Figure 4-9 show the numerically simulated horizontal convergences
for the profiles. The measurements points for numerical convergence are same position as the measurement points from the field tests. Note that the convergence time is with respect to computer time when equilibrium is reached.

The yielding of the rock bolts correspond to the areas where the talc layers are actively deforming. In profile Y825 segments of the rock bolts in the HW have yielded in tension, corresponding to the HW talc that has been undercut and is deforming into the stope. Similarly in profile Y850 the rock bolts also yielded in tension in the HW, where the talc is actively deforming by bending. In Profile Y875 the rock bolts that intersected the talc seams in the roof and FW have yielded in tension. The yielding is consistent with the mechanisms that drive the deformation of talc around the stope. For example, the rock bolts installed in the talc seam in the roof of Cut #4 in profile Y875 is yielding in tension since the sliding of the talc down the middle of the stope induces tensile stress due to the downward drag and high lateral stresses. This phenomenon is known to drive the back parallel cracks at the Kristineberg mine, leading to the church dome shaped roofs.

Profile Y875 particularly shows significant yielding of the shotcrete liner all around the profile. This profile is in fact located in the area where large shear movements were observed in the FW via the borehole camera (see Figure 4-4). In the HW borehole camera images showed closure of the borehole as the HW converges into the stope (Figure 4-10 (b)).

The numerically simulated horizontal convergence profiles are similar in magnitude and characteristics as observed in the field monitoring exercise.
Figure 4-6: Cut #4, Profile Y825 is supported with D-bolt. Segments of D-bolt have yield in tension mainly in HW side, where the talc has been undercut and deforming to the stope. Yielded shotcrete segments are shaded red.

Figure 4-7: Convergence around profile Y825, Cut #4. The measurement points are located approximately in the same position as in the field.

Figure 4-8: Cut #4, Profile Y850 is supported with rebar. Segments of rebar have yield in tension mainly in the HW side, where talc is deforming into the stope. Yielded shotcrete segments are shaded in red.
Figure 4-9: Convergence around profile Y850, Cut #4. The measurement points are located approximately in the same position as in the field.
Figure 4-10: (a) Cut #4, Profile Y875 is supported with rebar. Segments of rebar have yield in tension in the roof and the FW, where the talc seams are actively deforming into the stope. Yielded shotcrete segments are shaded in red. (b) Camera view of borehole closer due to convergence in the HW.

Figure 4-11: Convergence around profile Y875, Cut #4. The measurement points are located approximately in the same position as in the field.

4.3 Discussions and conclusions of field monitoring models

It was not possible to develop geological models that accurately represent the geology around the stopes because of incomplete and missing data. However, by comparing the drillhole geological data and the face mapping that was carried out in cuts #3 and #4, as well combining geological knowledge it was possible to develop geological models of the wall rock through profiles Y825, Y850 and Y875. The borehole camera snapshots and failure mapping of cut #4 also indicated that, the deformation mechanisms observed were consistent with the interpreted geology.
Numerical modelling based on the “real” geology was necessary as it would then be possible to compare field observations against results from numerical simulations, as well as identify geology and mechanism that control ground deformation around stopes. Furthermore, it provided the basis for model calibration.

The models with real geology clearly showed the talc to be the most influential lithology that controlled deformation around the stope. Three main mechanisms associate with talc were observed; shearing, dilation and bending. Bending typically occurs if the talc seam is thin. In this case it is forced to separate from the contact by compressive force from the ore in the roof and the lateral stresses. Dilation occurs with thick talc seams, where they squeeze and “belly” into the stope by dilating. Shearing is coupled with dilation.

Since the talc-schist is very weak it is punched by the ore in roof immediately above the stope. The ore then drags downwards under its own weight and in the process forces the talc-schist to slip along the clay filled contacts.

The segments rock bolts and shotcrete apparently yielded in areas where the talc was actively deforming. It was not possible to evaluate the difference between the performance of the D-bolt and rebar, since the deformations in the test stope were smaller than anticipated. Majority of the strain gauged rock bolts failed and data was unreliable, making it difficult to assess the performance of the rock bolts.

It is also possible that an entire length of a rock bolt can located inside the weak talc, if it is sufficiently thicker than the bolt length or parallel to the bolt application as in in profile Y875 in Figure 4-10 (a). In this instance the bolt will be straining with the talc, in which case the bolt may not be useful in anchoring the talc to the competent host rock.

The deformation of the stope is also a function of face advance. This can be seen from the fact that deformation continued to increase as the face advanced, with furthest rounds showing the largest deformation compared to rounds closest to face. Furthermore, shotcrete cracking was not seen in the early stages of the excavation until the final rounds were excavated. It is also estimated that at around 50% of the deformation has already occurred prior to installation of monitoring instruments. Displacements tracked from the unsupported numerical models show displacements to be almost twice that observed in the test stope, which of course were measured in supported ground conditions.
5 CONCEPTUAL NUMERICAL ANALYSES

5.1 Conceptual cases

The field measurements and numerical models based on the “real” geology revealed that, the talc is the most influential geology affecting the stope deformation. This observation is consistent with the generally observed behaviour at the mine and also with the previously conducted studies (e.g. Board et al., 1991, 1992). In consultation with Boliden engineers, the most likely or the typically observed scenarios were developed to be applied in the conceptual models. These are shown in Table 5-1 and Table 5-2. Table 5-3 shows a geometrical scenario where the cuts in a stope are offset from each other, due to faulting and folding of the orebody. The offsetting of the cuts is thought to have a significant effect on the deformation of the stope.

The geology for the conceptual models is simplified into four main units; (i) the orebody, (ii) host rock, (iii) talc-schist and its clay filled contact and (iv) the pyrite zone. The hangingwall talc-schist is slightly more competent than footwall talc-schist.

Table 5-1 shows the rock mass parameters for the conceptual models. The interface properties are same shown in Table 5-2.

Table 5-1: Conceptual scenarios for talc (red colour fill) in the FW with relatively thin layers in HW: (A1) thin layer with varying distance, (B) medium layer with varying distance, (C) thick layer with varying distance, (D) with medium layer through the middle of the orebody.
Table 5-2: Conceptual scenarios for talc (red colour fill) and pyrite seams (blue fill) in the FW, with thin seams of talc in the HW.
Table 5-3: Conceptual scenarios for faulting and folding: (G) Fault displaces the orebody and (H) Folding of the orebody

<table>
<thead>
<tr>
<th>G1</th>
<th>G2</th>
</tr>
</thead>
<tbody>
<tr>
<td><img src="image1.png" alt="Diagram G1" /></td>
<td><img src="image2.png" alt="Diagram G2" /></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>H1</th>
<th>H2</th>
</tr>
</thead>
<tbody>
<tr>
<td><img src="image3.png" alt="Diagram H1" /></td>
<td><img src="image4.png" alt="Diagram H2" /></td>
</tr>
</tbody>
</table>

Table 5-4: Rock mass inputs used in the conceptual analyses

<table>
<thead>
<tr>
<th>Rock type</th>
<th>$E$ (MPa)</th>
<th>$\nu$</th>
<th>$\sigma_{cm}$ (MPa)</th>
<th>$c$ (MPa)</th>
<th>$\phi$</th>
<th>$\sigma_t$ (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Host rock (cordierite-quartzite)</td>
<td>39000</td>
<td>0.25</td>
<td>46.4</td>
<td>6.7</td>
<td>30</td>
<td>0.40</td>
</tr>
<tr>
<td>Orebody</td>
<td>27000</td>
<td>0.25</td>
<td>12.6</td>
<td>5.0</td>
<td>23</td>
<td>0.40</td>
</tr>
<tr>
<td>Hangingwall talc</td>
<td>18000</td>
<td>0.30</td>
<td>7.0</td>
<td>2.5</td>
<td>19</td>
<td>0.20</td>
</tr>
<tr>
<td>Footwall talc</td>
<td>15000</td>
<td>0.30</td>
<td>3.5</td>
<td>1.5</td>
<td>9</td>
<td>0</td>
</tr>
</tbody>
</table>

5.2 Results - unsupported models

For the purpose of this report results from only a number of selected models are presented (see Table 5-5) to illustrate the deformation behaviour of rock mass around the stope with respect to the scenarios illustrated in Table 5-1 to Table 5-3. Descriptions of the behaviours are also described in Table 5-5. It is clear from these descriptions that the deformation behaviour of the stopes are highly dependent
on the thickness and location (i.e. distance away from HW or FW) of the talc. Pyrite, with high frictional resistance, assists in reducing deformation by resisting shear.

The conceptual analyses show that, the offsetting of the cuts apparently contributes to the reduction in the stope deformation. Faulting and folding, creates a scenario where the cuts have to be developed to conform to the shape of the orebody, thus resulting in cuts that are irregular and frequently offsetting each other, both along a the dip and strike.

Table 5-5: Conceptual models, numerical results and conceptual descriptions of the rock mass behaviour around the stope from the unsupported models

<table>
<thead>
<tr>
<th>Conceptual case</th>
<th>Numerical result</th>
<th>Conceptual description</th>
</tr>
</thead>
</table>
| Case 1: Thin talc seams (0.2 m) on the footwall FW and anhangingwall (HW) | ![Image](image1.png) | • The thin talc seams in the FW and HW are squeezed into the stope in the form of bending after being separated from the host rock. They appear too thin for volumetric dilation.  
  • In the roof the ore punches into the weak thin talc layers on either side, forcing them to break and separate along the contact.  
  • The downward drag of the ore, due to its weight and high lateral stresses, induces shear slip to occur along the contact. Meanwhile the broken section of the talc exposed in the stope FW and HW become free standing beams on which the compressive weight of the ore is applied. Thus forcing the talc seams to bend to stope like flexural beams and fail by crumbling.  
  • Back-parallel cracks develop in the roof due to forces resisting downward drag of ore and high lateral compressive stresses |
| Case 2: Medium size talc seam (1 m) in the HW       | ![Image](image2.png) | • The talc seam exposed in the HW separates at the contact and bends into the stope. Volumetric dilation occurs because the talc seam is relatively thick.  
  • In the roof the ore punches into the weak talc seam in the HW. Because the seam is relatively thick it is not broken as in Case 1. The ore is dragged downwards on the HW side, inducing sliding along the contacts of talc seam and forcing it to squeeze into the stope starting from at the shoulder.  
  • Movements are restricted on the FW side, leading to a situation similar to “fix-free” boundary conditions. As the ore is free to slide in the HW side, but restricted in the FW, a rotational moment is created, where the ore slides and rotates at the same time, while inducing sub-vertical tensile cracks that progress deep into the back of the orebody. Fallouts would leave behind a church roof back in the HW shoulder.  
  • If the talc seam breaks below the HW shoulder a wedge will be formed, permitting sliding of the talc seam in a circular fashion. |
| Case 3: Medium size talc seam (1 m) in the FW       | ![Image](image3.png) | • The behaviour is the same as in Case 2, except that they occur in FW side.  
  • Note: Significant deformations occur at the FW or HW shoulders, depending on the location of the talc seam. The ore punches into the talc seam and applies a downward compression on the seam, causing it to bulge at the stope. In all cases there is a significant difference in the modulus of the talc compared to the host rock and the ore. Therefore, there is distressing effect caused by the talc. For that reason the ore will likely to be sliding under its... |
Case 4: Thick talc seam (3 m) in the FW
- Same behavior as in Case 3, however deformation is much larger. There is significant volumetric dilation of the talc seam induced by the compression from the ore in the roof and the horizontal compressive stresses.
- At the base of the stope, i.e. on the FW side, rotational displacements occur, which could lead a circle failure of the footwall at the base of the stope.
- A similar behavior is expected in the HW if the seam occurs in the HW side, however, with slightly lower magnitude.
- The “bellying” of the footwall induces tension cracks to occur since the top and bottom end of the talc seam are restricted by the ore and the backfill.

Case 5: Talc seam in the middle of the ore
- The ore on either side of talc compress the talc in the middle and squeezes it into the stope.
- Progressive tensile cracks are formed in the roof as the ore drags downwards on either side of the talc seam, inducing sliding along the talc seam contact. Fallouts will leave behind a church above the middle of the stope.
- Maximum deformation occurs in the middle of stope back.

Case 6: Talc seam located 1 m from FW
- The behavior is similar to case 3 and 4. However, the presence of the 1 m thick host rock, separating the talc seam and the ore, limits the amount of squeezing deformation in the footwall.
- The host rock forced by the squeezing talc to bends in to the stope. The host rock will crack in tension as it “bellies” in to the stope.
- In the roof a behavior, similar to Case 1, is observed. Increasing strains cause sub-parallel cracks to develop in the back of the stope.

Case 7: Talc seam located 3 m from FW
- At 3 m from the footwall the influence of the talc seam on the footwall deformation is diminishing. This indicates that the further the talc seam is to the stope the less the effect is on the stope deformation. However, it is also relative to the stope height. If the stope height is greater, then the distance from the...
<table>
<thead>
<tr>
<th>Case 8: Pyrite between the FW talc seam and FW rock</th>
<th>• There is significant amount of resistance to sliding along the contact between the pyrite seam and talc seam. Clearly pyrite appears to resist shearing due to its high frictional resistance.</th>
</tr>
</thead>
</table>
| Case 9: Pyrite seam through the middle of the talc seam | • The unobstructed deformation of the 3 m talc seam, as seen in Case 4, is restricted by the presence of pyrite seam in the middle, thus reducing the overall FW deformation.  
• Shearing of the talc seam contact with footwall rock is diminished since the deformation of the outer segment of the talc seam is small compared to the inner segment that bounds the stope.  
• The presence of pyrite apparently obstructs and reduces both normal and shear deformation of the talc. |
| Case 10: Fault displaces the continuity of the ore half-way. | • The HW talc seam deforms inwards in the form bend of bending. Shearing minimal.  
• Horizontal deformation occurs in the footwall, but shearing is apparently non-existent.  
• In the roof the ore punches into the talc seams on the sides (HW and FW). The downward drag of the orebody noticeable in hangingwall but on the footwall.  
• This may also imply that the step-wise mining of the stopes may restrict large shear deformation on the footwall. In the HW, however, slightly largely deformation will be seen.  
• The disturbance to the continuity of the ore and talc seams, caused by faulting and folding, seem to affect the stope deformation behavior significantly. |
| Case 11: Fault displaces the continuity of the ore fully. | • The HW talc seam deforms by bending into the stope, with limited amount of shearing.  
• The FW talc seam also deforms inwards and normal to stope. Shearing is limited to the FW shoulder.  
• In the roof the ore punches into FW and HW talc seams. Downward drag of the ore cause shearing that is limited to the shoulders of the stope walls. |
• The stepped stoping significantly minimizes the deformation of HW and FW.

**Case 12:** Folding causes the ore to thin and swell in along its dip and strike.

- Both the HW and FW talc seams show small amounts of deformation. Shearing is limited.
- The offsetting of the cuts in a step-wise fashion results in limiting the deformation of the talc.
- The folding also allows the stopes to vary in orientation relative to the talc seams. This may a reason for the small displacements observed.

**Case 1:**

- The HW talc seam deforms by bending into the stope, with limited amount of shearing.
- The FW talc seam also deforms inwards and normal to stope. Shearing is limited to the FW shoulder.
- In the roof the ore punches into FW and HW talc seams. Downward drag of the ore cause shearing that is limited to the shoulders of the stope walls.
- The step-wise stoping significantly minimizes the deformation of HW and FW.

**Case 12:** Folding causes the ore to thin and swell in along its dip and strike.

- Both the HW and FW talc seams show small amounts of deformation. Shearing is limited.
- The offsetting of the cuts in a step-wise fashion results in limiting the deformation of the talc.
- The folding also allows the stopes to vary in orientation relative to the talc seams. This may a reason for the small displacements observed.
5.3 Results - Supported models

To assess the behaviour with rock support installed, the same cases used in the unsupported models were supported with rebar and shotcrete. The properties of the rebar and shotcrete and the same as those used in the real geology case models. Ground support installation procedures are those applied in the real geology case models. Table 5-6 show the results from these analyses. Yielding of the rock support elements are consistent with observations in the real cases analysed in Section 2.3, where the rock support elements are notably impacted by the properties of talc and spatial characteristics. Shotcrete experiences significant shear where sliding along the talc interface occurs, while the rock bolts show significant amount of the tensile failures. This observation on the rock bolt is consistent with field observations, where the rock bolts appear to get “sucked in” to the walls as a result of wall converge into the stope, resulting in the failure of anchor plates as the bolts stretch.

The thickness of the talc layers also seems to determine the impact length of the rock bolts, see Cases 2 to 4 for example. The bolts appear to yield along the length of the bolts that are exposed to the talc layers. The presence of thin seams of pyrite only limits the deformation, but does not prevent the straining on the rock bolts, see Case 9.

Table 5-6: Conceptual models, numerical results and conceptual descriptions of the rock mass behaviour around the stope from supported models.

<table>
<thead>
<tr>
<th>Conceptual case - supported</th>
<th>Numerical result – Cut #4</th>
<th>Conceptual description of support response</th>
</tr>
</thead>
<tbody>
<tr>
<td>Case 0: Plane talc altered interface (0 m) between ore and host rock on FW and HW.</td>
<td></td>
<td>• Rock bolts in the roof and FW shoulder yield in the tension (in response to tensile parallel back break in the ore) • Shotcrete yields in HW, the shoulders of FW and HW, and toe of FW.</td>
</tr>
<tr>
<td>Case 1: Thin talc seams (0.2 m) in FW and HW</td>
<td></td>
<td>• Rockbolts in roof yield (in response to tensile parallel back break in the ore and sliding along the interface) • Shotcrete yields in roof and shoulders of FW and HW.</td>
</tr>
</tbody>
</table>
**Case 2:** Medium size talc seam (1 m) in the HW

- Rockbolts in roof and HW yield (in response to tensile parallel back break in the ore, sliding along the interface and inward dilation of HW talc)
- Shotcrete yields in roof, shoulders FW and HW and toes of FW and HW.
- Competence of rock bolts have potential to protect easy yielding of shotcrete in HW

![Diagram](image1)

**Case 3:** Medium size talc seam (1 m) in the FW

- Rockbolts in roof yield (in response to tensile parallel back break in the ore, sliding along the interface and bulging and dilation of talc)
- Shotcrete yields in FW, middle of HW, shoulder of HW

![Diagram](image2)

**Case 4:** Thick talc seam (3 m) in FW

- Rockbolts in roof yield and FW (in response to tensile parallel back break in the ore, sliding along the interface and bulging and dilation of talc)
- Shotcrete yields in middle of HW, shoulder of HW and FW.
- Yield of bolts protect the shotcrete.

![Diagram](image3)
### Case 5: Talc seam (1 m thick) in the middle of the ore

- Rockbolts yield in the roof (in response to sliding along talc interface and tensile parallel back break in the ore).
- Yielding of shotcrete in hangingwall and footwall.

### Case 6: Thick talc seam (3 m) in FW

- Rockbolts yield in the roof (in response to tensile parallel back break in the ore) and in the FW (due to sliding along talc interface)
- Yielding of shotcrete in mid HW and shoulders of HW and FW.
- Competence in rock bolts protect shotcrete.

### Case 7: Talc seam located 3 m from FW

- Rockbolts yield in the roof (in response to tensile parallel back break in the ore)
- Yielding of shotcrete in mid HW, shoulders of HW and FW and toe of the FW.
- Competence in rock bolts protect shotcrete.
Case 8: Pyrite seam at boundary between 3 m thick talc and host rock in FW

- Rockbolts yield in the roof (in response to tensile parallel back break in the ore) and in the FW (due to inward bulging of talc).
- Yielding of shotcrete in FW, mid HW, shoulders of HW and FW

Case 9: Pyrite seam through the middle of the talc seam

- Rockbolts yield in the roof (in response to tensile parallel back break in the ore) and in the FW (due to inward bulging and dilation of talc).
- Yielding of shotcrete in mid HW and shoulders of HW and FW

Case 10: Fault displaces the continuity of the ore half-way.

- Rockbolts yield in the roof (in response to tensile parallel back break in the ore) and in the FW (due to inward bulging and dilation of talc).
- Yielding of shotcrete in mid HW and shoulders of HW and FW
5.4 Discussions and conclusions from conceptual models

The conceptual models are simple but represent some of the common characteristics of the wall rock at Kristineberg mine, particularly the talc. Although the occurrence of talc is generally common in the FW, it does however, occur in the HW and through the middle of the orebody. In the HW the talc seams are thinner than in the FW, indicating that the thickness of seams progressively grow from the HW to the FW.

The effects of talc on the deformation of the stope are affected by several factors:

i. Thickness of the talc seams. Thin bands of talc generally respond by bending, after being separate along the contact. Talc seams that immediately surround the ore are easily broken in the roof as the competent ore punches into the soft talc. Once broken the weight of the ore is applied directly on to the standing talc band causing it to flex and fail by crumbling. Thicker seams of talc, which are typically found in the footwall, generally squeeze and dilate into the stope. This
is caused a combination of the high lateral stresses and the downward compression caused by the weight of the ore. “Bellying” of the FW also seems to be associated with the thick talc seams. Wedges of cracks will show appear in the middle of the FW height as a result. At the base of the FW and under the backfill wedge shaped circular failure may occur. The backfill will heave if the shear failure occurs under the backfill.

ii. Location of the talc seams. The further the talc seams are to the stope, the effect diminishes. The competent rock mass separating the stope and talc seam behaves like a rock pillar. Nearer the talc is to the stope, the competent rock mass in between behaves like a slender pillar. If the talc seam is thick, the impact on the slender rock pillar is significant, as it is forced bend in to the stope.

iii. Occurrence of pyrite seams with talc. Pyrite seams appear to resist shearing of the talc, due to its high frictional resistance. This observation is consistent with the field observations made in Cut # 4 where lenses of pyrite were seen in the camera boreholes in some of the sections.

iv. Undercutting of talc seams. The mechanisms that drive deformation and failure are also affected by whether the seams are undercut or not. If the seams are not undercut by stope then bending is the primary mechanism. If the seams are undercut then shearing is the primary mechanism.

v. Talc through middle of the ore. If the talc seam is in the middle of the ore and is under-cut, it is forced to extrude by the high lateral stresses.

vi. The yielding of the ground support is associated with the deformation pattern of the talc. Since the talc often yields in a complex pattern of shearing, bending and bulging the rock support consequently appears to yield in tension, shear or combination of both.

vii. The shearing of the talc along the contact interface causes the rock bolt to bend and thus putting pressure on the anchor plates, which subsequently causes the anchor plate to break-off and separate from the rock bolt. This phenomenon is likely to be observed in the footwall and hangingwall.

viii. The downward sliding of the ore in the roof, coupled with the tensile back-parallel cracking of the ore exerts tensile stress on the rock bolts in the roof. Hence, tensile of the rock bolts are noticeable in roof. Anchor plates will suffer from the initial rock mass movements.

Other phenomena have also been observed:

i. Development of back parallel cracks in the roof. The parallel to sub parallel
cracks appear when the ore is dragged downwards, under its weight, with the
talc contact acting as the sliding (see Table 5-5). The downward drag of the ore
can be visualized as acting on “fix-free” boundary condition. The movement
is fixed on the boundary where talc is absent; while it is free to move downward
along boundary where the talc is present. Therefore movement is first initiated
from the wall where talc is present and because it is fixed on other wall a
rotational moment is created, leading to church dome shaped back parallel
cracks. A rock bolt would be subjected to shear, bending and rotational
moment under this scenario.

ii. If talc is present on both sides (HW and FW), the ore tend to slide into the stope
under its weight, as well as being forced by the high lateral stresses. Back
parallel cracks develop and propagate into the back until a state of equilibrium
is reached. A rock bolt would be in tension under this condition.

iii. Effect of faulting and folding. Faulting and folding results in irregular orebody
geometry. This results in cuts that offset each other, both up dip and along the
strike. The offsetting tends to cause less ground deformation for the subsequent
cuts. This scenario was clearly observed in the experimental stope, where cuts
are offset and is likely to be one of the main reasons for the low deformation
magnitudes observed in Cut #4. From the ground control point of view the
offsetting of cuts could be an advantage.

Since the maximum principal stress is known to be perpendicular to the orebody then bending
will be main the deformation mode if shearing does not occur. This is clearly seen in conceptual
models where the displacement vectors are perpendicular to stope walls where the talc is either
absent or thin.

Although the conceptual models show instabilities in FW, HW and the roof, the instabilities in
the roof are significant with respect to rock fall and immediate safety. Obviously the stability
of the walls is important for stabilizing the roof, since instability in the walls would provide a
pathway for the entire instability leading up to the roof.

Most of the fallouts would have occurred at the same time as blasting. This is evident from the
profiles in the test area where fallouts have occurred during blasting near the HW and FW
shoulders and other areas talc has been undercut.

6 GENERAL CONCLUSION

The unsupported models based on “real” geology showed ground deformation patterns that are
consistent with the deformation characteristics observed from the field monitoring data, that is,
convergence measurements and total station survey. Different mechanisms drive instability and ground deformation at Kristineberg and these include: shearing, dilation, bending and rotation. Depending on the characteristics of the talc, these mechanisms can be complexly coupled, thus subjecting a rock support element to a coupled mechanism.

The conceptual models revealed the complex mechanisms that drive the instability at Kristineberg. The understanding of these mechanisms is the key to understanding the response of rock support elements to these mechanisms. It is clear that, in Kristineberg these mechanisms are complexly coupled together and thus subjecting a single ground support element to different mechanisms at the same time.

The conceptual models confirm talc to be influential wall rock geology that controls the ground deformation around the stopes. The effect depends on the size of the talc seams and their location from the HW and FW. They bend if they are thin and they squeeze and dilate if they are thick. Separation and shearing occur along the contacts. The effect of the talc diminishes the further they are located from the HW and FW.

Pyrite lenses often found in the footwall with talc tends to reduce deformation by restricting shear due to its high frictional resistance.

Folding and faulting orebody appears to have an advantage from the ground control perspective. The offsetting of the cuts, to make them conform to the shape of the orebody, tends to restrict the deformation of the talc, probably due to creation of favourable stress scenarios due to destressing and that the cuts are probably acting as if they are independent of each other.

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8 REFERENCES


