Mining sequence for Lappberget 1250

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Abstract

Hereby presented thesis is a final project for joint master degree in European Geotechnical and Environmental Course at Delft University of Technology, Wroclaw University of Technology and University of Miskolc. The subject was granted by Boliden and was executed in cooperation with INERIS.

In longhole open stoping a choice of mining sequence is an important aspect, which can have a great impact on mine’s safety and profitability. With a various extraction patterns, stress distribution within the orebody can be manipulated. In general, the most desired scenario is to operate in areas, where rock mass had failed before any development started, resulting in distressed and safe working environment.

In this paper influence of mining sequence in Garpenberg mine on stress distribution was invested by means of numerical modelling. The model was built using FLAC 3D software treating the rock mass firstly as an elastic material and introducing plastic behavior in later stages. Different materials and geological variability was also progressively included, as the complexity of models was increasing. Material parameters were obtained from INERIS or assessed on the basis of other studies. Initial stress state calculations and model calibration was performed beforehand by INERIS as well. Three sequencing scenarios were modelled and compared. With the aim of evaluating, which one is the best from the rock mechanical point of view, a failure criterion was based on extreme principal stresses. In order to specify what are the stresses at failure, some past events and failures were recreated in the models. In the end the results were interpreted, which led to conclusions and recommendations on how to progress with mining sequence.

Keywords:

Mining sequence, longhole open stoping, traverse open stoping, numerical modelling, FLAC 3D, Boliden, Garpenberg, Lappberget.
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Boliden, August 2016

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1. Introduction

1.1. Background

Mining large orebodies requires proper extraction sequencing in order to operate safely and economically. Main factors that influence the choice of a sequence are: mine development advance, ore grade and stress distribution within the orebody (Villaescusa, 2003).

Mining sequence for Lappberget 1250 had been chosen based on a generic approach. Although the stress state itself was analysed through numerical modelling, it has not been used to examine the relation between the sequencing pattern and the outcoming stresses. Moreover the current sequence is designed only till 2017. There are no further detailed plans on how to proceed with mining afterwards. Therefore a development of the existing sequence has to be done. Having in mind also some of the minor failures that happened in the mining faces, which were a result of either the complex geology with extensive weak zones or stress concentration, which was an effect of the extraction pattern, it would be wise to investigate a future sequence more in details.

The project used to be a part of INERIS research program with collaboration of Boliden on the subject of ‘Global Stress Monitoring: approach to better assess ground instabilities, seismic and rock burst hazard in deep mining’. The objective of that ongoing study is to integrate mine development, in situ stress changes, seismic activity, and numerical modelling to better understand and control mine stability, as well as improve early-warning capabilities (Bigarre, 2016). Nevertheless the project timeline is too extensive and uncertainties about mining sequence of Lappberget 1250 need to be answered now. Hence presented here Master Thesis was derived from that project.

1.2. Objective and scope

The aim of this thesis is to perform an evaluation of the mining sequence for Lappberget 1250 block and related rock mechanical issues by means of numerical modelling. Three patterns will be considered: continuation of the pyramid shape, hourglass shape, and acceleration with the production in the eastern part of the orebody (Mozaffari & Ekstrand, 2016). Induced stress redistribution will be assessed and related rock mechanical issues, for which a way how to eliminate or mitigate them will be proposed. The influence of the sequencing pattern on the sill pillar above the 1250 block will be taken into account as well. Modelling will be done first for simplified geology, introducing zones of weaknesses later on, in order to determine and compare the effects of geology and material properties on stress redistribution. The model will be run for both elastic and elastoplastic analysis, which results will be interpreted and compared with each other. Conclusions will be drawn and recommendation on how to progress with mining sequence will be done.
1.3. Methodology

Modelling will be done using three-dimensional finite difference software FLAC 3D. Model will be built from scratch, except for the block model of the geological zones, which will be provided by INERIS. Rock parameters and initial stress state values will be obtained also from INERIS, who conducted their own stress measurement campaign in Lappberget 1250 in December 2014. During the period between April and October 2015 additional strain measurements were collected and then converted into incremental stresses. Obtained total stresses were compared with the predicted ones, which helped calibrating the model more accurately (Bouffier, 2015). Since the model built for the sake of this dissertation is a simplification of INERIS model the same virgin stresses can be used and no further calibration is needed.

1.4. References


Bigarre, P., 2016, Global Stress Monitoring: approach to better assess ground stabilities, seismic and rockburst hazard in deep mining, Internal presentation #895309

Bouffier, C., 2015, Stress measurement campaign and stress monitoring experiment in Lappberget mining area, at Garpenberg mine, Boliden, Internal presentation #856715

Mozaffari, S., Personal communication

Ekstrand, J., Personal communication
2. Overview of Garpenberg mine

Garpenberg mine is located in Hedemora municipality in Sweden (see Figure 1) and is the oldest Swedish mine still in operation, yet one of the most modern in the world. The first traces of mining activity in Garpenberg date back to the 13th century. It was acquired by Boliden in 1957 (Boliden Garpenberg brochure, 2016).

![Figure 1: Location of Garpenberg mine marked as a red pin (Google Maps, 2016)](image)

Garpenberg is a polymetallic underground mine operating from roughly 500 to 1250 meters deep using various mining methods, mainly longhole open stoping, but also cut-and-fill and longitudinal stoping. Production in 2015 reached 108 kt of zinc, 42 kt of lead, 288 t of silver, 0.8 kt of copper and 559 kg of gold (Boliden Mineral AB, 2016). Mine consists of couple orebodies presented in Figures 2 and 3 with overall mineral reserves and resources shown in the Table 1.
Figure 2: Garpenberg area with limestone-marble primary horizon and ore bodies (Boliden Mineral AB, 2015, Garpenberg Mineral Reserves and Resources Report, Internal report)

Figure 3: Cross section through Garpenberg’s orebodies (Boliden Mineral AB, 2015, Welcome to Boliden Garpenberg, Internal presentation)
Mineral reserves, 31st December 2015

<table>
<thead>
<tr>
<th>Quantity, ktonnes</th>
<th>Au, g/t</th>
<th>Ag, g/t</th>
<th>Cu, %</th>
<th>Zn, %</th>
<th>Pb, %</th>
</tr>
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<tr>
<td>Proven</td>
<td>12,500</td>
<td>0,3</td>
<td>109</td>
<td>0,06</td>
<td>5,3</td>
</tr>
<tr>
<td>Probable</td>
<td>27,300</td>
<td>0,3</td>
<td>115</td>
<td>0,04</td>
<td>3,2</td>
</tr>
</tbody>
</table>

Mineral resources, 31st December 2015

<table>
<thead>
<tr>
<th>Quantity, ktonnes</th>
<th>Au, g/t</th>
<th>Ag, g/t</th>
<th>Cu, %</th>
<th>Zn, %</th>
<th>Pb, %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Measured</td>
<td>16,800</td>
<td>0,3</td>
<td>110</td>
<td>0,05</td>
<td>3,2</td>
</tr>
<tr>
<td>Indicated</td>
<td>54,200</td>
<td>0,3</td>
<td>100</td>
<td>0,05</td>
<td>2,4</td>
</tr>
<tr>
<td>Inferred</td>
<td>11,700</td>
<td>0,5</td>
<td>65</td>
<td>0,08</td>
<td>3,7</td>
</tr>
</tbody>
</table>


2.1. References

Boliden Garpenberg brochure, 2016,

Boliden Mineral AB, 2016,
http://www.boliden.com/Operations/Mines/Garpenberg/

Boliden Mineral AB, 2015, Garpenberg Mineral Reserves and Resources Report, Internal report

Boliden Mineral AB, 2015, Welcome to Boliden Garpenberg, Internal presentation

Boliden Annual Report 2015,
3. Lappberget

3.1. Overview

Lappberget orebody constitute of over 60% of all the Garpenberg’s mineral reserves and its mined ore tonnage is equivalent to 80% of total ore mined in Garpenberg (Garpenberg Mineral Reserves and Resources Report, Internal report, 2015). It was discovered during an exploration drilling in 1997 and was put into production in 2003 (Koppen, 2008).

3.2. Geology

Lappberget is a steeply dipping volcanogenic hydrothermal deposit with width up to 250 meters. It has been formed below the sea floor in a caldera vent of a larger shallow marine rhyolite-dacite volcano in a stratabound replacement process. In Figure 4 the Lappberget anticline can be seen, its immense height is unique to the Garpenberg syncline and its origin is still not fully known. The general stratigraphic layers counting from the top are (Ghasemi 2012 & Allen 2003):

- Young volcanogenic rocks and volcanic/elastic sediments
- Limestone, which is in contact with ore
- Ore, mainly sulphides, skarn and impregnated quartzite
- Old volcanogenic rock types, often fine-grained.
Lappberget ore body consists of massive ore as well as impregnated ore, where some lower ore grades can be found. Part of the orebody, which is the subject of this thesis, consists primarily of flogopite quartzite and mica quartzite. Within the orebody there are some weak zones which influence considerably the stability and mining progression. They are either zones of heavily crushed rocks or schists, mainly talc schists (which are the weakest type of rock within the orebody boundaries), but also flogopite, sericite, and mica schist (Olson, 2016). Their presence causes also water problems, since production water can easily filtrate these fractured layers leading to leakages and flooding in the mine. Figure 5 shows the weak zones prediction regions
which are based on exploration drillings interpretation and are being updated together with drifts development.

![Figure 5: Weak zones in colours on each mining level for Lappberget 1250 (Boliden internal files, 2016)](image)

An example of a big rock failure that was caused by hitting a weak zone in the face of a crosscut drift can be seen in the Figure 6.

![Figure 6: Failure in a weak zone on level 1132, stope 15 (Olson, 2016).](image)

Beside the ore body, areas with an unfavourable direction of foliation in limestone can be also considered as troublesome, as it can be seen in Figure 7. They are however not taken into account as weak zones. On the other hand there can also be
found some areas of very hard rocks, which can make drilling difficult, such as skarn (diopside, garnet, actinolite, tremolite skarn).

Main metals bearing minerals in Lappberget are sphalerite, galena, various silver vein minerals and some smaller amount of chalcopyrite. It can be said that gold grade increases with depth, while silver grade decreases (Olson, 2016). Apart from them a huge amount of pyrite can be identified.

3.3. Mining method

Currently the whole Lappberget orebody is mined using the longhole open stoping method (called also traverse open stoping). In the past though, post-pillar cut-and-fill mining was used for levels 881-822 – which stopped in 2012. Nowadays production is carried out from level 530 to 1257. It is divided in four blocks (see Figure 8): 700, 800, 1100, and 1250, where the last one is the object of this thesis (Garpenberg Mineral Reserves and Resources Report, Internal report, 2015). Nevertheless, there are already plans to mine the next block below 1250, hence the entire underground infrastructure is being expanded with a ramp, located at 1314 meters down to date.
Mining in Lappberget begun in 2003 on level 881 with the post-pillar cut-and-fill method, which can be seen in Figure 8 as the levels between blocks 1100 and 800. At that time the choice of the method was justified by the access to the required equipment and knowledge as well as lack of paste backfill (Koppen, 2008). Ore was extracted in horizontal slices, just like in standard cut-and-fill, advancing from the bottom to top leaving pillars in between as a primary support throughout all the
levels. Waste rock was used as a backfilling material in the mined out cuts. Figure 9 illustrates an example of such method.

![Figure 9: Post-pillar cut-and-fill mining method (Buchan, R., M., Hard-rock room and pillar, Queen’s University, http://minewiki.engineering.queensu.ca/mediawiki/index.php/Hard-rock_room_and_pillar#cite_note-euler-2, March 2012)](image)

Later on this method was replaced by longhole open stoping (see Figure 10), which is more suitable for large steeply dipping orebodies. The orebody is split into separate stopes, which are mined in a predefined order, called sequence. They are approached by crosscut drifts that are developed from the perpendicular footwall drift. In Lappberget, in the wider parts of the orebody, the additional drifts in the hanging wall are developed, that allow the access to the stopes from both sides – front and back.

![Figure 10: Transverse longhole stoping method (Escobal Guatemala Project, NI 43-101 Preliminary Economic Assessment Southeastern Guatemala, Tahoe Resources Inc, 2013)](image)

First the opening drifts (crosscuts) are done on the top of a stope for charging and blasting purposes and at the bottom for mucking. In order to obtain a free face for blasting the stopes, firstly a drop raise is drilled, followed by drilling rings, charging
them and finally blasting (see Figure 11). Blasted material is then mucked from the bottom with a remotely controlled LHD (Load, Haul, Dump machine), which prevents workers from operating under unsupported roof. The stope is backfilled and ore is then transported by trucks in the ramp to the underground crushers situated at levels 700 and 1087, from where it is hoisted to the surface through a hoisting shaft.

![Figure 11: Blasting of a stope in Lappberget 1250: drop raise together with drilling rings (Boliden internal files, 2016)](image)

The opening drifts are scaled and shotcreted down to two meters below the roof and supported with resin grouted rock bolts (see Figure 12). In some parts of the Lappberget orebody dynamic rock support is used in form of mesh with D-bolts, where seismicity can be expected.

![Figure 12: Opening drift on level 1108, stope 15 (Mozaffari, 2016).](image)

Mined out stopes are backfilled with paste or waste rock depending on what kind of stopes they are (for more information see Chapter 3.4). Pastefill is produced in the pastefill plant at the site, being a mixture of tailings and a binder, which is then transported underground as slurry by system of pipes. Pastefill pumped into a stope require 7 to 10 days of curing time, before it can be safely accessed on top and 28 days before the adjacent stopes can be mined (Mozaffari, 2016).
3.4. Mining sequence in 1250 block

Mining in Lappberget 1250 follows so far a variation of primary-secondary pyramid shape sequence. There are six levels of stopes, each being 25 meters high with stope numbers from 5 to 23. Primary stopes are 10 meters wide, while secondaries 15. Above the block there is a sill pillar 22 meters thick, which separates it from the block above. The stability of that pillar is an important issue that will be considered in the later stages of this thesis. The sequence has to abide by the recovery times of the paste fill, as well as by the rule that mining of secondary stopes cannot start before two levels of adjacent primary stopes are finished. Figure 13 illustrates the layout of the stopes in the top view with stope numbers.

The operation started with simultaneous mining of pillar 13 from bottom and top and was followed by a pyramid shape mining upwards. Primary stopes are backfilled with pastefill, while secondary ones are backfilled with waste rock. The western part of the orebody, this is stopes 15 to 23, are divided into sectors A, B and C, since they are too long to be blasted at once. Sectors B in the primary stopes are 40 m long, while 20 meters in the secondary stopes (blocks B in secondary stopes have to be filled with pastefill, thus they are desired to be the shortest possible in order to decrease the amount of used pastefill). Blocks A and B have various length with

Figure 13: Top view of stopes layout together with footwall and hanging wall drift in Lappberget 1250 (Boliden internal files, 2016).
respect to the orebody shape. Sectors B are the first ones to be mined in this part of the ore, following by A and C in accordance with the pyramid pattern of the sequence. Blocks A are approached from the back drifts (Mozaffari & Ekstrand, 2016). Production plan was already designed until the end of 2017, therefore the sequence will not be modified for that period. Figure 14 presents the past and planned production until December 2017. The numbers stand for the simplified order in which the stopes were or will be blasted.

Figure 14: Sequence up to December 2017 (Świtała, 2016).

3.5. References

Boliden Mineral AB, 2015, Garpenberg Mineral Reserves and Resources Report, Internal report


Olson, J., Personal communication

Ghasemi, Y., 2012, Numerical studies of mining geometry and extraction sequencing in Lappberget, Garpenberg, Master thesis, Luleå University of Technology, Luleå, Sweden
Mozaffari, S., *Personal communication*


Ekstrand, J., *Personal communication*
4. Numerical modelling

4.1. FLAC 3D

Numerical modelling performed for the sake of this project was done in Itasca’s software FLAC 3D (Fast Lagrangian Analysis of Continua) version 5.01, which is a three-dimensional explicit finite difference program. The model is built out of polyhedral elements with assigned material properties, forming a grid. With applied boundary conditions the model has ability to yield and flow, although simulating a collapse is not possible, since it is a continuous media code. Discretization is based on substituting partial derivatives with differences described at the adjacent grid points (Jing&Hudson, 2002). The explicitness of the software means that the solution necessitates calculation steps, which number can be dictated manually by user or run automatically. During each step forces, velocities and accelerations are computed at the grid points using Newton’s laws of motion, from where the zones strains and stresses are derived. A static solution is obtained when the model is in a state of equilibrium or steady state flow, meaning that the resultant force at each grid point is close enough to zero or has reached a constant nonzero value (Itasca, User’s Guide, 2013). FLAC uses a general sign convention, meaning negative sign denotes compression and positive tension. By default FLAC is operating in a small strain mode, meaning that the grid points positions are not changing even when the displacements are significant. For mechanical calculations in mining field considerable rock mass movements can be expected, therefore a large strain mode should be activated, which enables mesh deformation. All discontinuities have to be described by applying an interface, on which sliding or separation phenomenon can develop. Nevertheless the software does not handle well complex and numerous interfaces. Not introducing any interfaces to the model, may cause geometry errors (when displacement of a given grid point is bigger than the mesh size element it belongs to) at the contact surfaces between varying materials. Those errors though, are purely numerical problems resulting from non-continuous character of the material and significant changes in their properties – an immense decrease in Young’s modulus from a strong to weak body is triggering a huge amount of strain at the contact walls. To avoid this issue manipulating properties of adjoining materials might be applied, with the aim of decreasing the difference in strength between them. Another solution might be neglecting the weaker body and model it as a void – no material, no strains computed.

4.2. Model geometry

To obtain satisfactory accuracy of the results and keep the model size and computing time reasonable, mesh size within the 1250 block was built out of cubes with edge size $u_0 = 2 \text{ m}$ and with gradient $r = 1.08$ going outside of the block. Size of the gradient mesh $n$ was calculated with means of geometric series: $a = \frac{u_0(1-r^n)}{1-r}$, where $a$ is a distance between the 1250 block and boarder of the model in given direction.
Boundaries were set in a distance of five times the span of the excavation from the block, in order to avoid the influence of excavations on the model boundaries. The geometry of the ore body and the weak zones was provided by INERIS with the lowest and highest level of the orebody extended to the model borders (see Figure 15). Total number of zones within the model is 5,542,008 with 5,637,663 grid points, which takes 13.2 GB of memory.

Within block 1250, groups for all the stopes with numbers from 5 to 23 were created, in contrast to INERIS model where stopes only from 9 to 19 were examined. The six meters high development drifts were derived with dimensions simplified to the same width as the stopes. Ramp, footwall and hanging wall drifts were not included (see Figure 16).
4.3. Material parameters and model calibration

A measurement campaign was run in December 2014 by INERIS with the aim of recognising the virgin stress state and elastic parameters of the rock mass, which allowed a proper calibration of the model. It consisted of three boreholes with CSIRO cells, two for stress measurements and one for permanent monitoring in each borehole. Boreholes were drilled from stope 15 towards stope 13 at level -1155, as it is shown in Figure 17. However, only three horizontal cells and one permanent descending cell were set successfully - subscribed in the figure below in bold letters (Bouffier, 2015).
To obtain the elastic parameters triaxial and biaxial tests were conducted on the overcored H1 and H2 samples, which results are presented in Table 2. The discrepancy between the Young modulus for the two samples can be explained by high level of heterogeneity in the orebody, unlike the triaxial test results, which were unrealistic and also highly dependent on the sample size, therefore they were not taken into account (see Table 2). Additionally dynamic elastic modulus was determined by means of microseismic surveying at the level of 80 GPa. Nevertheless it cannot be directly compared with the static moduli values. Density of the rock mass was verified by laboratory tests, which given an average of 3030 kg/m$^3$ (Bouffier, 2015).

<table>
<thead>
<tr>
<th></th>
<th>Young Modulus (GPa)</th>
<th>Poisson coefficient</th>
</tr>
</thead>
<tbody>
<tr>
<td>Triaxial tests on little samples (76 mm)</td>
<td>95</td>
<td>0.16</td>
</tr>
<tr>
<td>Biaxial tests on H1</td>
<td>44.9</td>
<td>0.23</td>
</tr>
<tr>
<td>Biaxial tests on H2</td>
<td>64.1</td>
<td>0.23</td>
</tr>
</tbody>
</table>

Table 2: Measured elastic parameters of the overcored samples (Bouffier, C., 2015, Stress measurement campaign and stress monitoring experiment in Lappberget mining area, at Garpenberg mine, Boliden, Internal presentation #856715).

From the overcoring measurements in the horizontal boreholes principal stresses were calculated by means of inversion, which are shown in the Table 3 with orientations relevant to the drifts orientation.
Table 3: Principal stresses calculated from the overcoring measurements (Al Heib, M., 2015, Deliverable D3.3, Prototype of hybrid numerical rockburst prediction tool, including stress and seismic monitoring, I2Mine Innovative Technologies and Concepts for the Intelligent Deep Mine of the Future).

From those measured data, initial stress state was back-calculated by modelling the state of excavation as it was at the time of the campaign and comparing computed stress tensor with the measured values. The rates of virgin stresses were assumed as follows: \( \sigma_{xx} = 44.35 \text{ MPa} \), \( \sigma_{yy} = 47.33 \text{ MPa} \), \( \sigma_{zz} = 34.33 \text{ MPa} \) (Al Heib, 2015). Table 4 presents principal stresses values, both measured and computed for all the models that were run. Taking into account the distance between the H1 and H2 boreholes of only 1.8 meters, the differences in the collected data can be explained by local high variety in rock properties or presence of fault or fractures. The measurement uncertainties are an important factor as well. Consequently it is not possible to match perfectly the surveyed stress state with the modelled one. Especially for the simplified models created for the purpose of this thesis.

<table>
<thead>
<tr>
<th>Measured [MPa]</th>
<th>Modelled (INERIS)</th>
<th>Modelled (Switala) [MPa]</th>
</tr>
</thead>
<tbody>
<tr>
<td>H1 Elastoplastic with weak zones [MPa]</td>
<td>55</td>
<td>53</td>
</tr>
<tr>
<td>H2 Elastoplastic model</td>
<td>40</td>
<td>48</td>
</tr>
<tr>
<td>Elastoplastic model with weak zones</td>
<td>24</td>
<td>32</td>
</tr>
</tbody>
</table>

Table 4: Collective table of principal stresses measured and retrieved from the models (Switala, 2016)

For the elastoplastic analysis Hoek & Brown failure criterion was chosen, which is a good approximation of hard rock behaviour in the underground excavations. It is a non-linear empirical formulation which can be described by the equation:

\[
\sigma_1 = \sigma_3 + \sigma_{eli} \sqrt{\frac{m\sigma_3}{\sigma_{eli}}} + s
\]

(1)

where \( \sigma_1 \) is a major principal stress, \( \sigma_3 \) is a minor principal stress, \( \sigma_{eli} \) is a uniaxial compressive strength of an intact material, \( m \) and \( s \) are material constants. Hoek & Brown parameters for ore, waste rock and weak zones were assessed by INERIS based on previous studies in addition to their own evaluations. They considered only one type of backfill for primary and secondary stopes, treating it as an elastic material, whereas in this dissertation the distinction for two types of backfill is made. For the secondary stopes waste backfill was adopted with elastic properties taken from the previous research *Strength of Backfilling* in Lappberget done by Rafi Keshvary. Primary stopes were backfilled with pastefill, which features were estimated based on the mentioned earlier study. Pastefill was assumed to be an intact
rock since it is a freshly formed material, thus \( s = 1 \) and \( \sigma_{cl} \) corresponds to the uniaxial compressive strength of the rock mass (UCS). Knowing the critical values of tensile strength \( T_o = 140 \) kPa and UCS = 1000 kPa from the mentioned earlier paper, calculated for the most representative cross section, \( m \) constant can be derived by simulating biaxial tension tests. It can be done in two ways (Hoek 2002; Amadei):

1) Setting in equation (1) \( \sigma_1 = 0 \) and \( \sigma_3 = -T_o \), which gives \( T_o = \frac{\sigma_{cl}}{2} (\sqrt{m^2 + 4} - m) \)

2) Setting in equation (1) \( \sigma_1 = \sigma_3 = T_o \), which gives \( T_o = -\frac{\sigma_{cl}}{m} \)

Solving the two equations for \( m \) two possible values of the parameter were calculated: 7,00 and 7,14 respectively. Mean value of these two were taken into further consideration. Table 5 presents the collective set of parameters used for numerical modelling.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Symbol</th>
<th>Unit</th>
<th>Ore (~quartzite)</th>
<th>Waste rock (~limestone)</th>
<th>Weak zones (~talc schist)</th>
<th>Pastefill</th>
<th>Waste backfill</th>
</tr>
</thead>
<tbody>
<tr>
<td>Density</td>
<td>( \rho )</td>
<td>kg/m(^3)</td>
<td>3030</td>
<td>2000</td>
<td>2500</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Young's modulus</td>
<td>( E )</td>
<td>GPa</td>
<td>66</td>
<td>57</td>
<td>20, 0,5</td>
<td>22,5</td>
<td></td>
</tr>
<tr>
<td>Poisson ratio</td>
<td>( \nu )</td>
<td>-</td>
<td>0,2</td>
<td>0,18</td>
<td>0,3, 0,2</td>
<td>0,125</td>
<td></td>
</tr>
<tr>
<td>Compressive strength</td>
<td>( \sigma_c )</td>
<td>MPa</td>
<td>188</td>
<td>110</td>
<td>30, 1</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hoek &amp; Brown parameters</td>
<td>( m )</td>
<td>-</td>
<td>10</td>
<td>1</td>
<td>7,07</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>( s )</td>
<td>-</td>
<td>0,112</td>
<td>0,001</td>
<td>1</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 5: Summary of material parameters used in the models built for the sake of this thesis (Renaud, 2016; Świtala, 2016).

### 4.4. Modelling blocks above block 1250

After the models had been built and consolidation state had been solved, mined out blocks, above the 1250 block, were modelled with a waste rock backfill for the elastoplastic models. Their sequence was not relevant to this study, however their presence influences significantly the stress state in the area. At first, contour of the blocks up to -900 meters was provided, which was the first case solved in FLAC. Later on the contour of the upper blocks was extended till -530 meters, which was corresponding to the real mining activity footprint in Lappberget (see Figure 18). Nevertheless, after the computations had been run, no major differences in stress state were noticed in the 1250 block in comparison to the previous scenario. Figures 19 and 20 present major principal stress in cross section, where the biggest changes between the -900 blocks and -530 blocks were found in elastic model. It can be seen that at the top of the sill pillar, locally stress variation is up to 16 MPa, which decreases with the distance from the blocks (see Figure 21).
Figure 18: Geometry of the mined out blocks above block 1250 from -1080 to -530 meters (Świtała, 2016).

Figure 19: Elastic model, major principal stress [MPa], Y455 cross section through the mined out blocks from -1080 to -900 meters (Świtała, 2016).
Figure 20: Elastic model, major principal stress [MPa], Y455 cross section through the mined out blocks from -1080 to -530 meters (Świtała, 2016).

Figure 21: Major principal stress at the depths below the mined out blocks in the elastic model along X3780 Y455 (Świtała 2016).
For the elastoplastic model excavating blocks till -900 or -530 meters affected the sill pillar and block 1250 even less, as it can be seen on the Figure 22.

![Elastoplastic model (X3780, Y455)](image)

**Figure 22: Major principal stress at the depths below the mined out blocks in the elastoplastic model along X3780 Y455 (Świtała 2016).**

With that in mind, as well as the fact that the computation time in FLAC depends strongly on the size of volumes excavated at once – the greater the volume, the longer it takes for the software to bring the model back to the equilibrium state, it was decided to model only blocks between -1080 and -900 meters for the further calculations.

Besides omitting the mining sequence for these blocks, the issue of how to simulate the backfilling process was still a question for the elastoplastic model. Solving the model till its equilibrium after the extraction and before backfilling, would suggest that the whole area was mined out before any backfilling started, which generated huge amount of stresses. On the other hand, solving the model after the extraction phase only in few computational steps, not allowing for a complete stress redistribution, would imitate a scenario where backfilling started immediately after the blocks were mined out. Both situations are exaggerations, which results are shown in Figure 23 and 24, though the second one seems like a better approximation, since in reality those blocks were not excavated at the same time therefore the consequent stresses were certainly of lower values than in the first scenario. This second case was selected for the succeeding computations.
Figure 23: Major principal stress [MPa] in the elastoplastic model solved fully after excavating blocks between -900 to -1080 meters, cross section Y455 (Świtała, 2016).

Figure 24: Major principal stress [MPa] in the elastoplastic model solved in 10 computational steps after excavating blocks between -900 to -1080 meters, cross section Y455 (Świtała, 2016).
In later stages of the project it was noticed that the secondary stopes backfilled with waste rock start to carry load as the sequence progresses, with the major principal stress up to 40 MPa by the end of the block extraction. Also the stresses in the mined out blocks above, which are backfilled with the same material were up to 35 MPa. This behaviour is not expected from the rock fill. It can bear some load after it consolidates, however not this much. Thus, it can be concluded that the strength parameters assumed for the waste rock were too high. This raises a concern especially about the situation in the sill pillar, which in reality should be the one taking all that excess load instead of the waste rock itself. In order to quantify the influence of strength properties of waste rock on the model, a simulation of backfilling blocks 900-1080 with a weaker material was run. The new parameters were assessed based on the literature study with values of bulk modulus of 1 GPa and shear modulus of 0,5 GPa (Vargas, 2014). In Figure 25 a cross section through the sill pillar and block 1250, showing the maximum principal stress, is presented. It can be seen that the decrease in backfill strength affects the sill pillar significantly up to 75 MPa difference in $\sigma_1$. Also the level of stresses in the sill pillar with the corrected waste rock characteristics, are much closer to the ones calculated in the elastic model. Nevertheless in the block 1250, only in the stopes on level 1132-1108 increase in $\sigma_1$ can be noticed as an outcome of using weaker backfill waste rock. Further consequences of using the overestimated waste rockfill in the elastoplastic models will be discussed in the next chapters.

![Elastoplastic model (X3780, Y455)](image)

Figure 25: Major principal stress at the depths below the mined out blocks in the elastoplastic model along X3780 Y455 for different waste rock backfill parameters (Świtała 2016).
4.5. Failures and factor of safety

To be able to predict a failure, level of stresses at the state of failure has to be recognised. Having in mind Hoek & Brown criterion, where failure is governed by the two extreme principal stresses, it was assumed that failure occurs when the difference between the major and minor principal stress is higher than some critical value. Therefore a factor of safety (FOS) was determined as:

\[
FOS = \frac{\sigma_1 - \sigma_3}{\sigma_{1,\text{critical}} - \sigma_{3,\text{critical}}}
\]  \hspace{1cm} (2)

The definition of FOS here was reversed in order to avoid having a zero value in the denominator, which is not allowed in FLAC. Thus failure zones will be identified where FOS ≥ 1.

For this purpose some past events from block 1250, which were not connected directly to the weak zones, but more likely were caused by stresses, were reproduced in the models. The mine has also seismic monitoring network installed, nevertheless recorded seismic events in Lappberget 1250 are too scattered within and around the orebody to use them as an implication of failure.

4.5.1. Failure 1108 R15

First of them took place back in 2014, after the first stope 1257 R13 (stope 13 between -1257 and -1232 meters – named always after the lower level) was produced and stope 1132 R13 was being blasted. The failure occurred in drift 1108 R15 in form of cracks and major roof spalling together with some water leakages (see Figure 26). It might be worth mentioning that at that time in the mine full bolt grouting was not practiced, therefore the support system was weaker than it is nowadays.

Figure 26: Failure at 1108 R15 drift with miniature showing the location of the drift (Nyström, 2014).
After recreating this state of mining in the elastic model, the stress concentration in drift 1108 R15 were clearly much higher than in the other drifts in the block, which is compatible with the location of the failure. Figure 27 and 28 shows the difference between maximum and minimum principal stresses, which at the 1108 R15 drift reached 83 MPa. The same state calculated in the elastoplastic model does not show that distinctly the stresses accumulation in the discussed drift (see Figure 29). On the contrary, the $\sigma_1 - \sigma_3$ in 1108 R15 is highly similar to the rest of the drifts and equals 69 MPa. However such low stresses at 1108 level can be explained by the waste rock used as a backfill in blocks 1080-900, which underestimate the stresses in that area. Same thing can be said about the elastoplastic model with weak zones, where the only additions are the local areas of very low stresses due to those weak zones.

**Figure 27:** Maximum and minimum principal stress difference [MPa] at ‘failure 1108 R15’ in elastic model (Świtała, 2016).
Figure 28: Maximum and minimum principal stress difference [MPa] at ‘failure 1108 R15’ in elastic model, Y455 cross section (Świtała, 2016).

Figure 29: Maximum and minimum principal stress difference [MPa] at ‘failure 1108 R15’ in elastoplastic model, Y455 cross section (Świtała, 2016).
4.5.2. Extensometer shift

The second event occurred on 25.10.2015 when extensometer located in stope 1207 R14 recorded a sudden displacement between its anchors at 11 and 14 meters (see Figure 31) equalled to 14 mm. Some small fractures were also observed in that drift. These readings were linked with blasting of the adjacent stope 1207 R13 (marked in grey in figure), which happened at the same day. When this situation was modelled, the displacements computed in the place of mentioned earlier anchors, were around 14 mm in all the models (see Figure 30 and Appendix A for more figures), which was in accordance with the real life measurements. The corresponding major principal stresses in the models were around 50 MPa. Although this event cannot be treated as a failure, it proved models to be coherent with the real life observations. In particular the elastoplastic models, which despite the incompatibilities in the sill pillar, are still valid inside the block 1250.

Figure 30: Maximum and minimum principal stress difference [MPa] at 'extensometer shift' in elastic model, Y469 cross section; extensometer marked in black (Świtała, 2016).
4.5.3. **Failure 1157 R15**

More incidents started to appear in March 2016 in drift 1157 R15 (from Y450 to Y463) and are continuing to develop up to date (being the end of July). Initially cracks, walls damage and bolt failures were observed (see Figure 32), followed by floor heave and more floor cracks after the first blast of stope 1182 R13 was taken, which is the last part of the stope 13 to be extracted.
Looking at the elastic model in this stage, the highest value of $\sigma_1 - \sigma_3$, in the place where the failure happened, is around 65 MPa (see Figure 33 and 34). Although the model is indicating some other drifts as well, with similar or even higher stresses, there were no extra problems reported beside a few fractures in the shotcrete in the roof of 1182 R11 drift. Disregarding levels from -1132 meters and above, elastoplastic model shows akin results with $\sigma_1 - \sigma_3$ equals to 69 MPa in the area of interest. In the last model, the difference in the drift can be perceived where the weak zones appear – within them the stresses are dropping close to zero, nevertheless on their abutments stresses in ore are much higher than in previous models. That resonates in the level of $\sigma_1 - \sigma_3$ equal to 83 MPa in the failure region, which is close to a talc schist unit. Considering the model as whole, the failure area in this case is more outstanding, as the stresses are of slightly higher values than in the other openings (see Figure 35).
Figure 34: Maximum and minimum principal stress difference [MPa] at ‘failure 1157 R15’ in elastic model, Y450 cross section (Świtała, 2016).

Figure 35: Maximum and minimum principal stress difference [MPa] at ‘failure 1157 R15’ in elastoplastic model with weak zones, Y455 cross section (Świtała, 2016).
4.5.4. Pastefill failure

The most recent failure was noticed at the beginning of June 2016 in pastefill in drift 1157 R13 (when mining in underhand fashion, some drifts have to be developed twice – firstly through the ore and secondly through the pastefill, which is the issue here). Roof spalling was noticed when blasting started in stope 1182 R13 (see Figure 37). In elastoplastic model in the pastefill at failure $\sigma_1 - \sigma_3$ were up to 0.8 MPa and 1.6 MPa including the weak zones, whereas displacements observed in the models were up to 2 cm (see Figure 36).

![Figure 36: Displacement [m] at ‘pastefill failure’ in elastoplastic model, Y425 cross section (Świtała, 2016).](image)
4.5.5. Choice of failure criterion

Decision of which values of $\sigma_1 - \sigma_3$ should be used as critical ones for the safety factor calculations was based on couple of reasons. Mainly, it was more crucial to have a criterion connected to the orebody rather than pastefill, since failures in the rockmass are more common and can be more rapture and severe in consequences. Comparing the two events that happened in stope 15 in elastic model, stresses at 1157 R15 were much lower, yet already caused a failure. Thus these lower stresses might be considered closer to the actual threshold value after which a failure begins. Values of $\sigma_1 - \sigma_3$ in 1157 R15 in elastic and elastoplastic models are akin, which is another argument in favour of choosing this failure. Taking into account also the fact that in the elastoplastic models stresses in the upper levels of block 1250 are underestimated, it would be wrong to base a failure criterion on an event located at those depths. To sum up, event 1157 R15 was chosen for future failure predictions (see Table 6).

<table>
<thead>
<tr>
<th>Model</th>
<th>Failure 1108 R15</th>
<th>1157 R15</th>
</tr>
</thead>
<tbody>
<tr>
<td>Elastic</td>
<td>82</td>
<td>65</td>
</tr>
<tr>
<td>Elastoplastic</td>
<td>69</td>
<td>69</td>
</tr>
<tr>
<td>Elastoplastic with weak zones</td>
<td>76</td>
<td>83</td>
</tr>
</tbody>
</table>

Table 6: Values of $\sigma_1 - \sigma_3$ at failures in MPa obtained from different models; in bold letters values chosen as critical for safety factor calculations (Świtła).
4.6. **Sequencing**

Three possible continuations of the planned 2017 sequence were modelled (see Figure 38). However some simplifications had to be made with the view on time frames for the project as well as the software requirements. First of all, one stope is modelled to be excavated at once, while in reality it requires few separate blasting steps. Secondly, the drifts are modelled as if they were developed full stope length straight and directly before the appurtenant stope is produced, whereas in fact drifts are being built in phases with some time advance to the stopes. Moreover, in the elastoplastic models, where plastic zones develop, it is crucial to maintain an appropriate distance between regions that are excavated in one computational step, in order to avoid the interinfluence between them. It was assumed that the safe distance equals to \( \frac{3}{\sqrt{\pi}} \left( \frac{6V}{V} \right) \), where \( V \) is the volume of the mined out zone, which gives 70 up to 100 meters required span, depending on the stope volume. As a consequence the stopes which were blasted in the same time period, but were closer than the specified distance, had to be modelled as if they were taken out separately following the assigned order. Finally, backfilling in the models takes place immediately after a stope is mined out.

First proposed pattern is a continuation of the bottom to top pyramid shape in primary-secondary fashion. Second pattern assumes an hourglass shape sequence with mining proceeding simultaneously from top and bottom, except the secondary stopes, which are backfilled with waste rock, therefore cannot be mined underhand. The third pattern preserves the bottom to top pyramid shape although production is accelerated in the eastern stopes, where the ore grades were higher (Mozaffari, 2016).
4.7. References


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Amadei B., Strength properties of rocks and rock masses, Lecture notes, University of Colorado at Boulder, Boulder, USA

Nyström, A., Personal communication

Bouffier, C., 2015, Stress measurement campaign and stress monitoring experiment in Lappberget mining area, at Garpenberg mine, Boliden, Internal presentation #856715

Mozaffari, S., *Personal communication*


5. Results interpretation

While interpreting the results, a special attention was paid to the condition of secondary stopes. Undesirable are scenarios where a stope is close to failure prior to its development. It poses a threat of a failure happening while production, which is dangerous to people but also can lead to delays in operation and additional costs when rehabilitation is needed. Stopes which fail before they are approached are preferable – working space is distressed and therefore safe. Another area of focus was electrician station situated on -1080 meters approximately above stope 8 from the footwall drift side. The station has already experienced a significant amount of fracturing both on the roof and walls, where locally additional meshes were installed. Thus it was important to inspect how big of an impact are the different sequences on the station. Safety of drifts, where certain activities are performed directly by people, such as charging, play a significant role as well. However, as it was specified in chapter 4.6, sequence of drifts was not modelled precisely enough to use this criterion as a determinative factor for choosing the best sequence.

5.1. Elastic model

Elastic model was built only with ore as a material, in order to understand the influence of sequencing on the behaviour of rock mass and be able to distinguish which phenomena are provoked by geological factors and which are a consequence of geometry. The sequences were simulated in 20 steps, excavating several stopes at the same time, in order to save computation time and disc space.

By the end of 2017, when the three patterns were modelled, all the secondary short stopes (5 to 14), between already excavated primary stopes, appear as failed (black colour in FOS figures). The same can be said about the secondary B stopes in the western part of the orebody (15 to 23), which length is similar to the eastern stopes (see Figure 39). Primary stopes B there, were mined mainly exclusively, before production of blocks A and C started, therefore they were acting in a similar manner as the short stopes (see Figure 40).
Figure 39: Factor of safety at the end of 2017 in elastic model – cross section through Y 440 (Świtała, 2016).

Figure 40: Factor of safety at the end of 2017 in elastic model – cross section through Z -1195 (Świtała, 2016).
From that point forward regardless to the pattern, disparities in stope performances between the short stopes and long ones can be observed. As the sequences are progressing, all the short secondary stopes are failing immediately after the adjoining stopes are blasted. In the western side however, the secondary stopes are failing only at their extremities in blocks A and C. A symmetrical arch shape failure footprint is visible with safety factor in the middle parts often close to 1.0 (see Figure 41). An exception from that rule is the top level of stopes 1108 – 1132 and partially stopes 1132 – 1157, which behave differently depending on a sequence pattern. The biggest failure extent presents pattern 3, for it causes the sill pillar to fail the soonest out of the three patterns, what affects the upper levels of block 1250 (see Appendix D). Pattern 1 (see Appendix B) exhibits a smaller amount of failed area within the two top levels of stopes, than pattern 3. Whereas pattern 2 (see Appendix C) can be characterised as the one with the least failed upper stopes as well as the lowest pace of damaging the sill pillar. There are much more zones with safety factor between 0.8 and 1.0 than in pattern 1 or 3, which can be considered as close to failure and therefore dangerous.

Patterns impact on the sill pillar was assessed by comparing a stage of mining for every pattern, at which the volume of ore extracted was approximately the same. Various cross sections through the sill pillar and block 1250 (see Appendix E) showed the vastest size of sill pillar in failure in pattern 3 as it was stated before. Additionally, influence of the mined out blocks 1080 – 900 on levels 1108 – 1157 can be clearly identified, emanating in non-symmetrical allocation of failure zones. Concentration of failure areas can be recognised in blocks C (see Figure 42), straight above which are situated blocks from -1060 to -900 meters. At level -1080 the area where the electrician station is located, evince as failed from the very beginning of

Figure 41: Factor of safety in elastic model, pattern 1 – cross section Z.-1182 (Switala, 2016).
the modelled sequences (this is after year 2017). It can be seen that the footprint of the failed zone is consistent with the contour of the mined out blocks above, which suggest that the stress state there is governed mainly by the blocks 1080 – 900 and changes in sequencing in block 1250 are not of great relevance (see Figure 43). In all the patterns biggest deformations within the orebody were observed in the western stopes, especially in the secondary stopes B up to 6 cm. Slightly bigger displacements of around 7 cm could be noticed in pattern 2 (see Figure 44).

Figure 42: Factor of safety in elastic model, pattern 1– cross section Z.-1108 with miniature showing in 3D the location of mined out blocks above block 1250 (Świtala, 2016).

Figure 43: Safety factor in elastic model on the left with top view showing the location of the stopes on the right; cross section Z.-1080; approximated location of electrician station marked in red ellipse (Świtala, 2016).
5.2. Elastoplastic model

Elastoplastic model was built with distinction for ore and surrounding the block 1250 waste rock being primarily limestone. As backfilling material pastefill was used for primary stopes and waste rockfill for secondary stopes. The sequences were simulated in average 140 steps. Interpreting the outcomes from this model, levels above -1132 meters were disregarded due to underestimated stress state discussed in chapters 4.4 and 4.5.1. Moreover, comparison with the elastic model can be done only in early stages of sequencing when not many secondary stopes are produced and waste rock backfill does not carry a load yet, disrupting the stress state.

Keeping those restrictions, first thing to notice in the elastoplastic model is that the range of failure zones is considerably smaller for each of the patterns than in the elastic model (see Figure 45). The reasoning behind it is that stresses computed are lower in elastoplastic model up to 20 MPa in compare to previous model, but also the value of stresses at failure taken for the safety factor calculations was slightly higher. This kind of response is to be expected, since plasticity is a softer kind of behavior where rock can withstand additional portion of stresses while deforming plastically until it reaches failure. Therefore the only secondary stopes that are fully failing are stopes 6, 8, 10 and in pattern 3 additionally stope 14. The rest of the stopes are failing only at their ends, while long secondary stopes in the later stages are not even close to failure with FOS around 0.5 to 0.6 throughout most of their length. Deformations though are similar to the ones in elastic model, reaching up to maximum 8 cm (see Figure 46).
Figure 45: Factor of safety at the end of 2017 in elastoplastic model – cross section through Y 440 (Switala, 2016).

Figure 46: Displacements [m] in elastoplastic model, pattern 2 – cross section Y 440 (note: displacements marked with the cross are just an outcome of a misspelling error in the code, please disregard them).
5.3. Elastoplastic model with weak zones

Discussed in this chapter model is an upgrade from the last one by including the weak zones (being mainly talc schist) within the orebody. It was built with the aim of understanding the influence of remarkably weaker rock type lenses on total stress distribution.

Stress level in this model is alike to the elastoplastic model, beside the local changes in places of weak zones. Value of $\sigma_1 - \sigma_3$ inside the weak zones can be found between 10 and 0 MPa. This causes stress redistributions in the surrounding areas. Since weak zones are situated mainly in the eastern part of the orebody, short stopes are the ones affected the most. It can be observed that stresses are very often pushed out to the crowns of the pillars (see Figure 47). Consequently, short secondary stopes, where the talc schist is present are not even close to failure. Because of the much higher value of critical stresses set for the model, all the secondary long stopes are indicating failure only at some ends. Displacements within the weak zones are up to 14 cm, however in the ore they are akin to the ones obtained in the elastoplastic model (see Figure 48).

![Figure 47: Factor of safety, pattern 3, elastoplastic model with weak zones – cross section through Y 440 (Świtala, 2016).](image-url)
5.4. Models considerations

The idea of the modelling was to start with the simplest elastic model, with no variations in material properties and gradually advancing to more complicated models introducing plasticity and diverse materials. It would be anticipated that the elastoplastic model will produce much more deformations than an elastic model. Still the displacements obtained in those models were very much alike. Moreover, some previous tests and research performed on quartzite, proved it to be a brittle rock with nearly elastic response ending in a brittle failure, as it can be seen in Figure 49 (Jaeger&Cook&Zimmerman, 2007). Therefore it was assumed that the elastic model is a better approximation of the orebody.

5.5. References

6. Conclusions and recommendations

Uniformly in all the patterns and models a tendency can be noticed for the eastern stopes to fail more than the western stopes, which is clearly a feature connected to the stopes dimensions. As long as the primary stopes B were being extracted together with the eastern short stopes, all the adjacent secondary stopes were failing (not including the areas, where weak zones were emerging). As soon as the Primary B stopes were followed directly by primary A and C stopes, they were distressing the area, not allowing for full failure and causing the middle parts of secondary stopes (which are mainly B sectors) to be very often in a state close to failure. Thus, from the rock mechanical point of view, it would be advisable to mine stopes B well in advance, before extracting lower levels of stopes A and C.

Taking into account elastic results, it can be concluded that pattern 3 is the best solution out of the examined scenarios. It contributes to the fastest failure of sill pillar, which affects the upper secondary western stopes, causing them to fail throughout their whole length in contrary to the other patterns. Apart from that, accelerating with the production of the short stopes, which have a higher ore grade, has also financial advantages. Although failure of sill pillar in terms of the condition of electrician station is not that favourable, it is not considered to be a deciding factor. The station appears as failed throughout all the patterns, hence shape of block’s 1250 stopes is more crucial.

Weak zones proved to be of a significant influence on the stress distribution, causing their relocation and mitigation, which given the complexity of their geometry and occurrence is difficult to predict without numerical modelling. Therefore having that in mind as well as the type of the orebody, which is mainly a brittle rock, it is recommended to build an elastic model with weak zones treated as a plastic material for a better assessment. Backfilling could be taken into account as well if an investigation of paste fill stability is desired. Having such model, displacements could be verified and localised more precisely and deformation criterion can be introduced as a serviceability limit state for drifts on top of the stress criterion presented in this report, which can be treated as an ultimate limit state criterion.
7. References


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Appendix A – extensometer shift

Displacements [m] in elastoplastic model, Y469:

Displacements [m] in elastoplastic model with weak zones, Y469:
Appendix B – evolution of FOS in pattern 1, elastic model, Y440

(In black failed areas)
Appendix C – evolution of FOS in pattern 2, elastic model, Y440

(In black failed areas)
Appendix D – evolution of FOS in pattern 3, elastic model, Y440

(In black failed areas)
Appendix E – FOS, sill pillar, elastic model, Z -1095

(In black failed areas)