Dybdedokument for gruvedrift

1.0 MINING

1.1 Approach and Method

The principal objective for the underground mine design and scheduling for both the Nussir and Ulveryggen deposits was to maximise resource extraction of the economically viable material to demonstrate the technical and financial feasibility of mining the deposits. Secondary to this objective was ensuring that the proposed mining sequences prioritised mining zones and stopes containing Indicated Resources as assigned in the mineral resource estimation exercise for Nussir in 2016 and the last update to the Ulveryggen deposit in 2010. Compromises were made in ensuring that Inferred Resources were excluded or deferred in the mining plan to Indicated Resources. The practicality of the mining superseded using the resource categories as guidance for prioritising and sequencing the mine schedules. This is contrary to common practice requiring publicly listed companies submitting Technical Reports (such as JORC or NI 43-101 studies) to exclude or defer Inferred Mineral Resources from mining plans and cashflows.

The topography, strike length and depth of the Nussir deposit makes exploration drilling of sufficient density (to ensure that all of the proposed mining areas are to at least an Indicated level of confidence) infeasible for Nussir to have completed between the Preliminary Economic Assessment (PEA), completed in 2014, and the requirement to complete this PFS in 2016. The reader is therefore cautioned that although the Mineral Resources have been generated to a standard consistent with international reporting standards, the mine plan and schedule has included Inferred material which is also included in the Base Case project cashflow. The distribution of the diluted resources within each panel is tabulated in Table 5-1.

Table 1-1: Summary of Mineral Resources by Category Included in Nussir Mine Plan

Panel	Category	Diluted Tonnes	Cu (%)	Ag (g/t)	Au (g/t)
Panel 0	Inferred	1 162 422	1,11	9,92	0,10
	Indicated	-	-	-	-
	Sub-Total	**1 162 422**	**1,11**	**9,92**	**0,10**
Panel 1	Inferred	-	-	-	-
	Indicated	3 328 852	1,09	9,21	0,11
	Sub-Total	**3 328 852**	**1,09**	**9,21**	**0,11**
Panel 2	Inferred	2 665 691	0,67	11,12	0,14
	Indicated	923 999	0,93	14,21	0,10
	Sub-Total	**3 589 690**	**0,74**	**11,92**	**0,13**
Panel 3	Inferred	5 658 129	1,11	8,51	0,13
	Indicated	4 643 965	1,01	9,24	0,09
	Sub-Total	**10 302 094**	**1,06**	**8,84**	**0,11**
Total	Inferred	9 486 242	0,99	9,42	0,13
	Indicated	8 896 816	1,08	9,75	0,10
	Total	**18 383 058**	**1,03**	**9,58**	**0,11**

Inferred Resources comprise approximately 52 % of the Nussir mine plan.

Once a mining system was selected, the Nussir deposit was sub-divided into 4 panels numbered 0 to 3 from east to west. These panels were then further divided into levels and stopes based on the sub-level interval and stope size defined by preliminary elastic and linear-elastic 2D rock mechanics modelling and empirical design. The primary mine access, a tunnel, was designed from the Øyen processing plant site to the mid-point of Panel 1. Panel 1 was then designed to a level of detail consistent with a PFS. The remaining panels were not designed, but factors and development lengths were scaled from the stope layouts and using Panel 1 designs as a basis.

The stopes for each of the panels were scheduled in turn. Panel 1 was first, then Panel 3 followed by panels 2 and 0. The panel sequence was driven by 3 factors:

a) Ready access for mining, particularly Panel 1;

b) Contained grade and tonnes; and

c) Mineral Resource Category.

The Ulveryggen deposit was included in the mine plan with production between years 1 and 7 of the schedule. The Ulveryggen deposit currently comprises only Inferred Mineral resources but is accessible almost right from the start of site development. The parts of the Ulveryggen deposit that were included in the mine plan amounted to 2,5 Mt with an average grade of 0,94 % Cu.

The following sections describe the methods, analyses and findings of the mining study that inform the cashflow model.

1.1 Basis of Design

The length, depth and width of the Nussir orebody is amenable to mining by underground methods. Furthermore, local governmental regulations and permits have imposed limitations on the amount of surface disturbance which makes open pit mining of the Nussir orebody not feasible at this time. As a result of these constraints, open pit mining of the Nussir deposit was not considered in this study.

The Ulveryggen deposit consists of several smaller sized orebodies, and a common feature in these orebodies is that they become smaller with increasing depth. Small abandoned open pits already exist, and potential surface extraction using crown pillar recovery methods at Ulveryggen will be discussed in this report together with underground mining methods.

1.1.1 Mine Design and Scheduling

All mine planning and mine design discussed in this report is based on data received from Nussir. The diamond drillhole database has been compiled in Microsoft Excel by a technical resource geologist with Promin. The database has been checked for consistency prior to use and imported into the Rana Gruber planning software (GEMS) by Rana Gruber geologists. Any inconsistencies and errors were corrected and subsequently reported back to Promin. The database contains collar, geology, assay and geotechnical data (RQD and Q-value). While compiling the drillhole database, all drillhole ID's from the Nussir area were changed to a uniform format by Nussir (please note that this PFS report only refers to the new drill hole ID's).

The geological models for Nussir and Ulveryggen (solid and wire frame) and the block model were compiled by Adam Wheeler (Competent Person (CP) under JORC) and supplied to Rana Gruber via Nussir and Promin.

1.1.1.1 Coordinate System

All maps and sections presented in this study are plotted based on the UTM coordinate system. All coordinates and coordinate grids refer to UTM WGS84 zone 35N.

1.1.1.2 Mine Planning Software

All data is imported into the mine planning software GEMS (version 6.7) built upon a SQL database. Projects have been built in GEMS for both Nussir and Ulveryggen.

For the Nussir area a set of vertical sections (with 125 m steps between sections), starting with the reference '*0NUS, 125NUS'* in the east have been established (see Figure 5-1, where the ore model is shown in red). These sections are orientated normal to the average strike of the eastern part of the Nussir orebody.

For the Ulveryggen area a 50 m grid, comprising vertical sections references '0ULV, 50ULV, 100ULV etc. has been established roughly normal to the orientation of the orebodies (Figure 5-2). It is these section numbers which will be referred to throughout this report. If not otherwise stated, these section show a view corridor of ±25 m.

All horizontal sections (plan views) presented in this study refer to their elevation above or below mean sea level, for example plan view 'L-150' indicates a plan view 150 m below mean sea level.

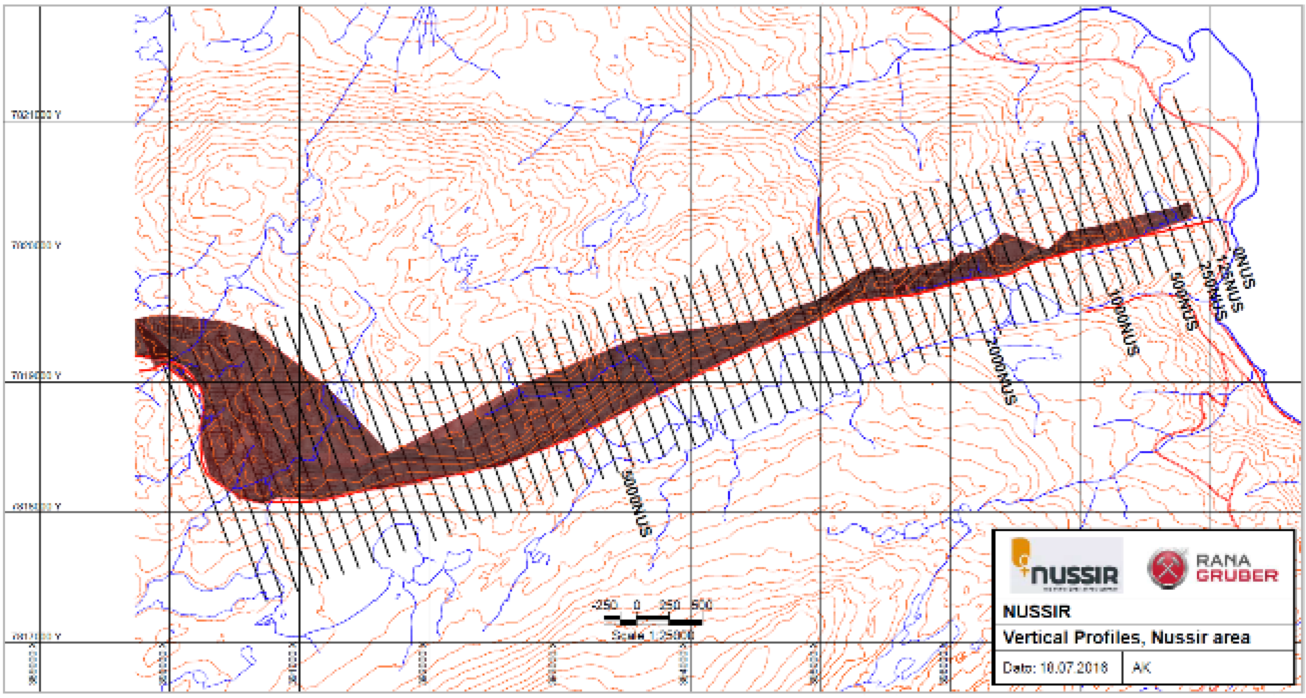

Figure 1-1: Map of the Nussir Area Showing the Sets of Vertical Sections (125m Steps).

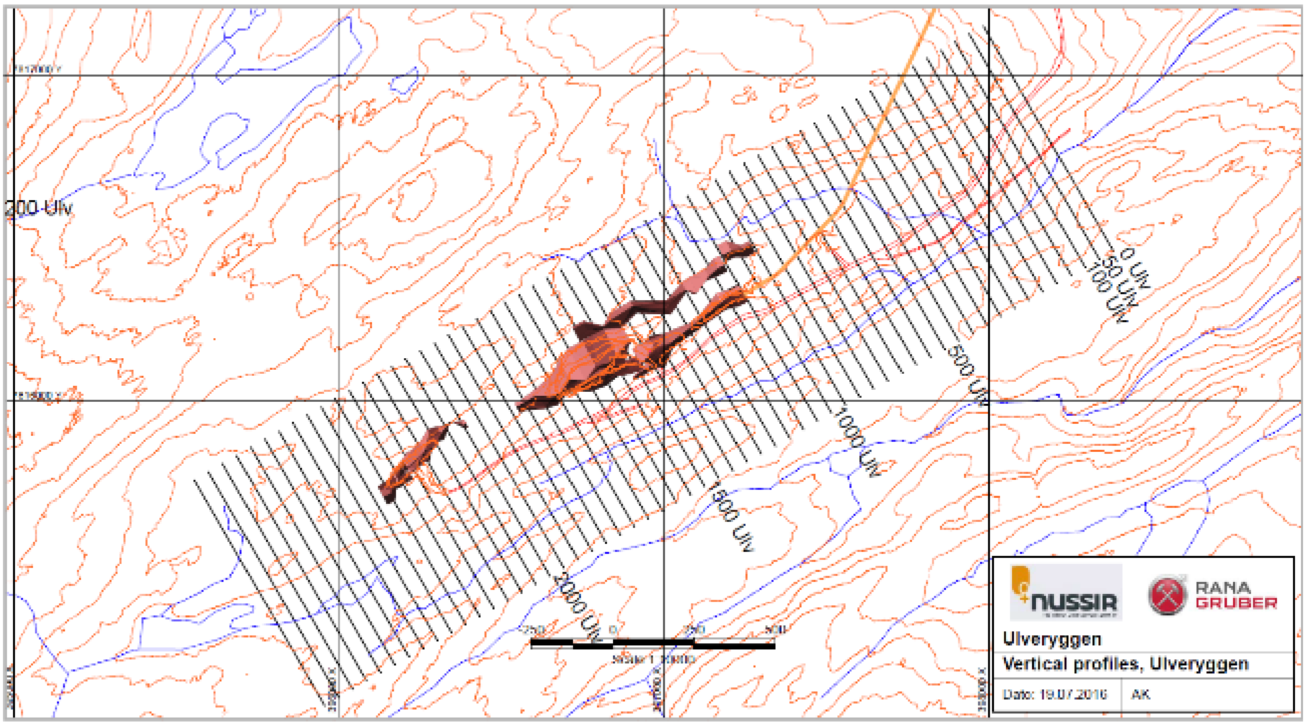

Figure 1-2: Map of the Ulveryggen Area Showing the Sets of Vertical Sections (50m Steps).

1.1.2 Geotechnical Analysis

1.1.2.1 Stress Measurements

In-situ rock stress measurements were completed for both the Nussir and Ulveryggen areas in spring 2016 as described by SINTEF (SINTEF, 2016). The stress measurements at Nussir were taken from a close to vertical drill hole (NUS-DD-15-003) at a depth of approximately 0 metres above sea level (mASL) with the first measurement obtained at a depth of 287,3 m. The Ulveryggen measurement was taken in the roof of the existing transportation tunnel at 'pel 800'. The test results for both locations indicate relatively high horizontal stresses (σH) in the area. In the Nussir area the horizontal stress measured was 19,1 MPa, while the Ulveryggen result was higher at 27,3 MPa. A summary of the test results can be found in Table 5-2.

Table 1-2: Stress Measurements in the Nussir/Ulveryggen Area (Larsen (2016))

Location	Borhole /Name	Method	Year	σH, MPa	σh, MPa	σv, MPa	σH from N Degree	Comment
Nussir	NUS-DD-15-003	Hydraulic fracturing	2016	19,1	9,2	5,7	69	Vertical drillhole, test depth = 0m a.s.l.
Ulveryggen	T1; pel 800	2D	2016	27,3	15		220	Test depth = 5m, in the roof

These results correlate with other stress measurements collected in the Finnmark region. Horizontal stresses in the order of 20 MPa have been measured by over-coring in the Sydvaranger iron ore mine, the Stjernøy nepheline syenite mine, and the Biddiovagge Cu-Au mine (Ref. 5-1: (Myrvang 2009)). The Sydvaranger measurements were taken at less than 50 m below surface in the main access ramp to a planned underground mine. There horizontal stresses lead to severe spalling in the roof of the tunnel. Myrvang (2009) reports further indications for high horizontal stresses such as off-set of vertical boreholes in road cuts in the Laksefjord and Porsanger region and exfoliation (surface parallel, semi horizontal fractures) in the Øyan aggregate pit.

1.1.2.2 Rock Mechanical Properties

In 2009, 2012 and 2016 core samples from different drillholes were tested for their rock mechanical properties including Young's Modulus E, Poisson's ratio and uniaxial compressive strength (UCS) (Ref. 5-1: (Myrvang (2009) and Ref. 5-2: (Hagen 2012)). All test samples from the Nussir site are taken from holes located in the eastern part of the deposit, with hole NUS-DD-07-001 just east of the large scale fold in the west, as shown in Figure 5-3, where the thick red line outlines the contact between the ore and the footwall.

The Ulveryggen sample tested in 2016 is from cores extracted during stress measurements in the transportation tunnel that runs beneath the abandoned open pits. The test results are summarized in Table 5-3.

In general, the UCS test results indicate strong rock units in the hangingwall, mineralization zone and the footwall. However, the 2012 UCS test results should be considered with some caution. Ref. 5-2 Hagen (2012) reports that the ISRM standard requirements for the UCS test (5 samples with a width to length ration of 2.5 to 3) could not be achieved using the received core samples. Most of the tests performed rely on 4 to 5 samples with somewhat lower width to length ratios. The sample showing the lowest strength recorded during the test (39.2 MPa, mineralized section from core NUS-DD-11-005) is solely based on one sample/core and as such does not meet the ISRM standards. Therefore, this sample was excluded from any further calculation.

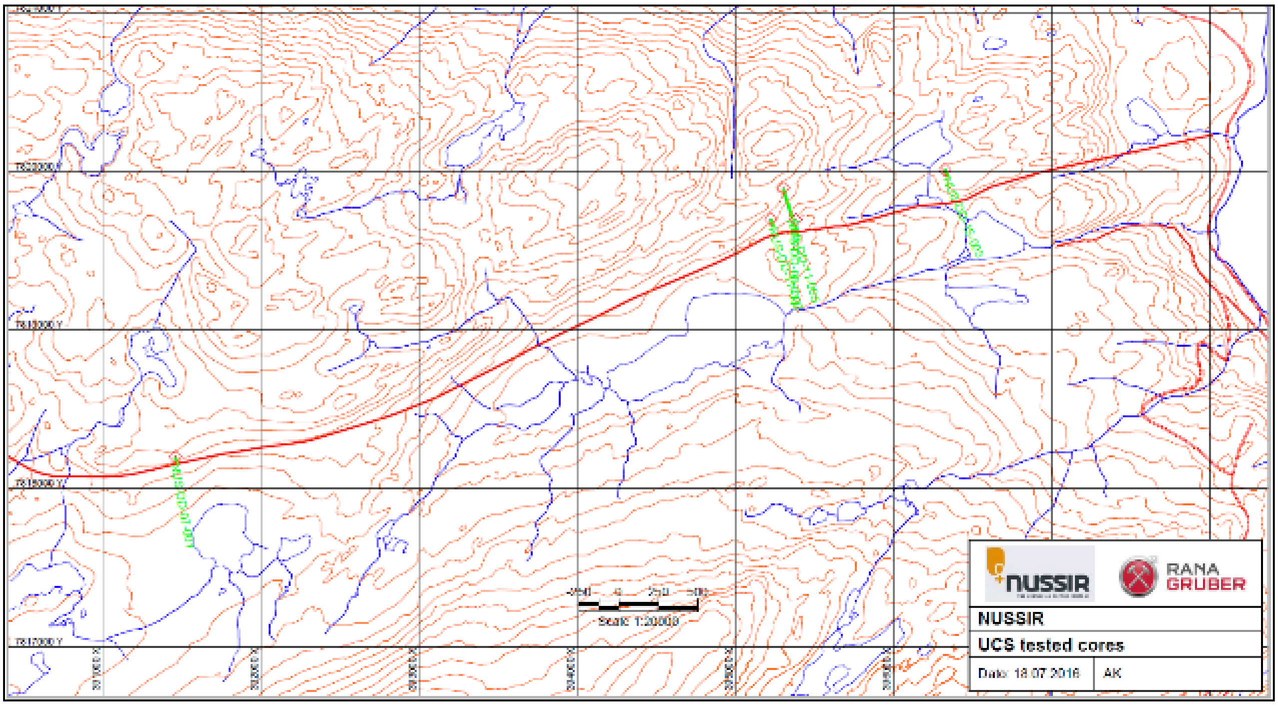

Figure 1-3: Map View of the Area Covering the Eastern Part of the Nussir Cu-Deposit Indicating the Location of the UCS Tested Drill Cores.

Table 1-3: Rock Properties (Average Values) From Selected Samples and Locations (Nussir and Ulveryggen).

Location	Year	Rock type	Rock unit	E-modul, Gpa	Poissons ratio	UCS, MPa	Unit weight kg/m3
NUS-DD-07-001, 47,0-48,0m	2009	Sandstone	Hangingwall			122	2 660
NUS-DD-08-011, 93,6-95,0m	2009	Sandstone	Hangingwall			125	
NUS-DD-08-011, 135,4-142,1m	2009	Sandstone	Hangingwall			92	2 740
NUS-DD-11-002, 142,0-143,0m	2012	Claystone	Hangingwall	68,7	0,2	69,1	2 734
NUS-DD-11-005, 309,0-310,0m	2012	Claystone	Hangingwall	69,5	0,18	67,7	2 715
NUS-DD-15-027, 309,0-309,7m	2016	Dolomite	Hangingwall	70,7	0,28	97,5	2 787
NUS-DD-15-BH1, 397,75-398,3 m	2016	Sandstone	Hangingwall	75,1	0,19	123,3	2 724
NUS-DD-15-BH18, 41,66-42,00 m	2016	Claystone	Hangingwall	82	0,25	110	2 763
NUS-DD-07-001, 51,5-53,5m	2009	Dolomite	Ore			102	2 730
NUS-DD-11-002, 147,0-148,0m	2012	Dolomite	Ore	69	0,24	120,8	2 698
NUS-DD-11-005, 316,0-317,0m	2012	Sandstone	Ore	45,7	0,27	39,2 *	2 667
NUS-DD-15-027, 312,0-312,7m	2016	Sandstone/ Dolomite	Ore	44,1	0,16	52,8	2 745

Location	Year	Rock type	Rock unit	E-modul, Gpa	Poissons ratio	UCS, MPa	Unit weight kg/m3
NUS-DD-16-BH18, 50,52-54,12	2016	Dolomite/ Sandstone	Ore	72,4	0,23	88,7	2 772
NUS-DD-11-002, 152,0-153,0m	2012	Claystone	Footwall	74,8	0,19	99	2 675
NUS-DD-11-005, 322,0-323,0m	2012	Sandstone	Footwall	77,3	0,19	175,8	2 657
NUS-DD-15-027, 314,1-315m	2016	Claystone	Footwall	64	0,23	82,8	2 712
NUS-DD-15-BH1, 405,3-408,5 m	2016	Sand-/Silt-/Claystone	Footwall	70,9	0,2	132	2 685
NUS-DD-16-BH18, 56,45-56,7 m	2016	Micaschist	Footwall	73	0,23	85	2 611
Ulveryggen, pel 800 in tunnel	2016	Unknown	Ulveryggen	102,2	0,26	221,7	2 690

* Unreliable test result, based on only one core.

1.1.2.3 Rock Quality Designation (RQD)

The Rock Quality Designation (RQD) index is described Ref. 5-3: (Deere 1964). The RQD is a measurement of the percentage of intact rock with a length of over 10 cm recovered from a given length of a drillhole (typically 1,5 m). RQD is often used as a measure of rock mass integrity during drill core logging because of its simplicity and because it is a quick way of obtaining rock quality information. It is used to provide quick identification of any weak or highly fractured rock units (whether it is fractured, sheared, jointed or weathered rock). This is described further in Ref. 5-4 Deere and Deere (1988). For the Nussir project, two datasets with RQD data exist. Unfortunately, these datasets cannot be easily combined because the same drillhole IDs are found in both datasets and different consistently overlapping sections down the holes are reported. There is no RQD data available for the Ulveryggen project. The RQD values for Nussir are the basis for further investigation using the rock mass quality system (Q-value) which will be discussed in the following section.

1.1.2.4 Rock Mass Quality (Q-value)

The rock mass quality system (Q-value) was developed by the Norwegian Geotechnical Institute (NGI) and introduced in 1974 (Ref. 5-5 Barton, et al. (1974)) to assess the rock mass stability of underground openings in hard rock jointed rock masses. Since its introduction, it has been revised twice (1993 and 2002) with many new examples from underground excavations in Norway, Switzerland and India. High Q-values indicate higher strength rock masses, whilst low values indicate the likelihood of poor stability, as shown in Table 5-4. The Q value is based on 6 parameters and is calculated using the following equation:

$$Q = \frac{RQD}{J_n} x \frac{J_r}{J_a} x \frac{J_W}{SRF}$$

The six parameters used to calculate the Q-value are:

RQD = Degree of jointing (Rock Quality Designation)
J_n = Joint set number
J_r = Joint roughness number
J_a = Joint alteration number
J_w = Joint water reduction factor
SRF = Stress reduction factor

These parameters are determined during geological mapping and core logging using tables that give numerical values for the different parameters.

When determining Q-values from drillholes it is important to remember the following:

1) Evaluation of the roughness coefficient J_F may be difficult because only a small section of each joint surface will be available for investigation, some may not be seen at all.

2) The drilling direction may influence the number of joints that are intersected by the drillhole so that the joint set number J_n might be under or overestimated.

3) It may be difficult to estimate the stress reduction factor (SRF) in massive rock. However, the SRF can be estimated based on overburden, depth below surface or if stress measurements are collected. Note that there were no stress measurements available at the time some of the older Q-values were determined. However, the assumed horizontal stress of 20 MPa used in those analyses is very close to the value of 19,1 MPa measured by hydraulic splitting in 2016 by Larsen, therefore it has been concluded that re-calculation of the previously obtained Q-values is not necessary. Since 2009 selected core material from 13 drillholes from the eastern part of Nussir were logged to calculate the Q-values, and all results are presented in Table 5-5. Q-values presented are average values where several consecutive sampled sections in the drillholes exist. Q-values were determined for the hanging wall, mineralized zone and the footwall.

In general, the reported Q-values fall into the categories 'fair' to 'good'. However, there are defined zones especially in and adjacent to the mineralized zone that fall into the 'poor' category. Figure 5-4 to Figure 5-7 show cross sections illustrating the orebody and the zones investigated for their Q-values, the colours correspond to the GEMS coding in Table 5-4. No Q-values are reported for the Ulveryggen mineral deposit.

Table 1-4: Classification of Rock Mass Quality Based on Q-values (Barton, et al. (1974)) and Colour Coding Used in GEMS, see also Vertical Sections Presented.

Q-Value	Rock quality	Colour coding (GEMS)
<0,1	Exceptionally Poor	
0,1 - 1	Very Poor	
1 - 4	Poor	
4 - 10	Fair	
10 - 40	Good	
40 - 100	Very Good	
100 - 400	Extremely Good	
400 - 1000	Exceptionally Good	

Table 1-5: Q-values from 13 Diamond Drillholes.

Borehole-ID	From	To	Q-Value Average	Geology	Comments
NUS-DD-06-007	40,7		15,60	Hangingwall	Western area
NUS-DD-06-007	79,4		6,59	Ore	Western area
NUS-DD-06-007	92,4		26,42	Footwall	Western area
NUS-DD-07-001	51		17,71	Hangingwall	Western area
NUS-DD-07-001	51,5		50,00	Ore	Western area
NUS-DD-07-001	75		20,79	Footwall	Western area
NUS-DD-08-011	113,1		24,20	Hangingwall	Eastern area
NUS-DD-08-011	142,1		34,73	Ore	Eastern area
NUS-DD-08-011	144,4		18,60	Footwall	Eastern area
NUS-DD-08-016	401,9		31,57	Hangingwall	Eastern area
NUS-DD-08-016	405,2		147,60	Ore	Eastern area

Borehole-ID	From	To	Q-Value Average	Geology	Comments
NUS-DD-08-016	407,3		0,00	Footwall	Eastern area
NUS-DD-08-023	127,4		25,63	Hangingwall	Eastern area
NUS-DD-08-023	128		166,70	Ore	Eastern area
NUS-DD-08-023	142		26,33	Footwall	Eastern area
NUS-DD-08-027	100,8		56,23	Hangingwall	Eastern area
NUS-DD-08-027	102,5		71,37	Ore	Eastern area
NUS-DD-08-027	103,3		54,60	Footwall	Eastern area
NUS-DD-15-001	365	370	55,53	Hangingwall	New drillholes
NUS-DD-15-001	370	375	128,00	Ore	New drillholes
NUS-DD-15-001	395	400	68,00	Hangingwall	New drillholes
NUS-DD-15-001	400	405	64,00	Ore	New drillholes
NUS-DD-15-001	405	410	100,00	Footwall	New drillholes
NUS-DD-15-002	335	340	33,00	Hangingwall	New drillholes
NUS-DD-15-002	340	345	47,00	Ore	New drillholes
NUS-DD-15-002	345	350	97,00	Footwall	New drillholes
NUS-DD-15-003	289,7	295	38,00	Hangingwall	New drillholes
NUS-DD-15-003	295	300	50,00	Ore	New drillholes
NUS-DD-15-006	230	234	7,00	Hangingwall	New drillholes
NUS-DD-15-006	244	247	90,33	Ore	New drillholes
NUS-DD-15-006	247	253	300,00	Footwall	New drillholes
NUS-DD-15-007	190	195	100,00	Hangingwall	New drillholes
NUS-DD-15-007	205	210	40,00	Ore	New drillholes
NUS-DD-15-008	76,6	84,6	17,00	Hangingwall	New drillholes
NUS-DD-15-008*	91	94,6	5,00	Ore	New drillholes
NUS-DD-15-008	94,6	97,7	75,00	Ore	New drillholes
NUS-DD-15-025	430	440	33,00	Hangingwall	New drillholes
NUS-DD-15-025	440	448,2	17,00	Ore	New drillholes
NUS-DD-15-025	449,5	451	0,50	Footwall	New drillholes

Eastern area is east of X-coordinate: 392000

* Alteration zone / Shear zone is crossing the ore

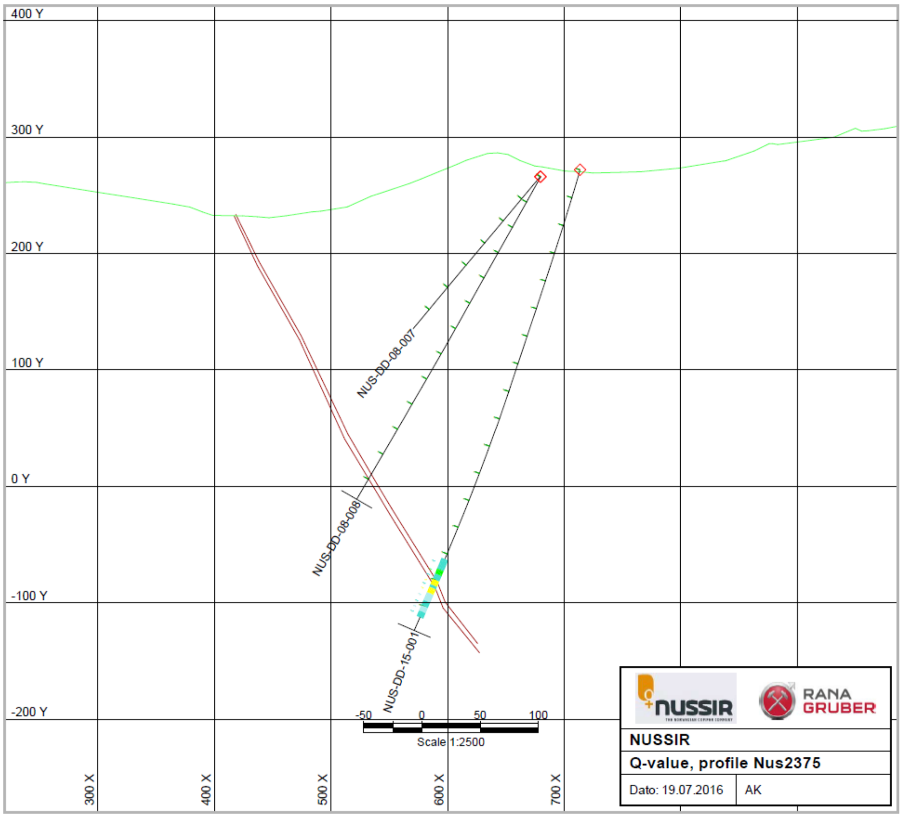

Figure 1-4: Q-values Determined in Drill Hole NUS-DD-15-001, Vertical Section 2375NUS.

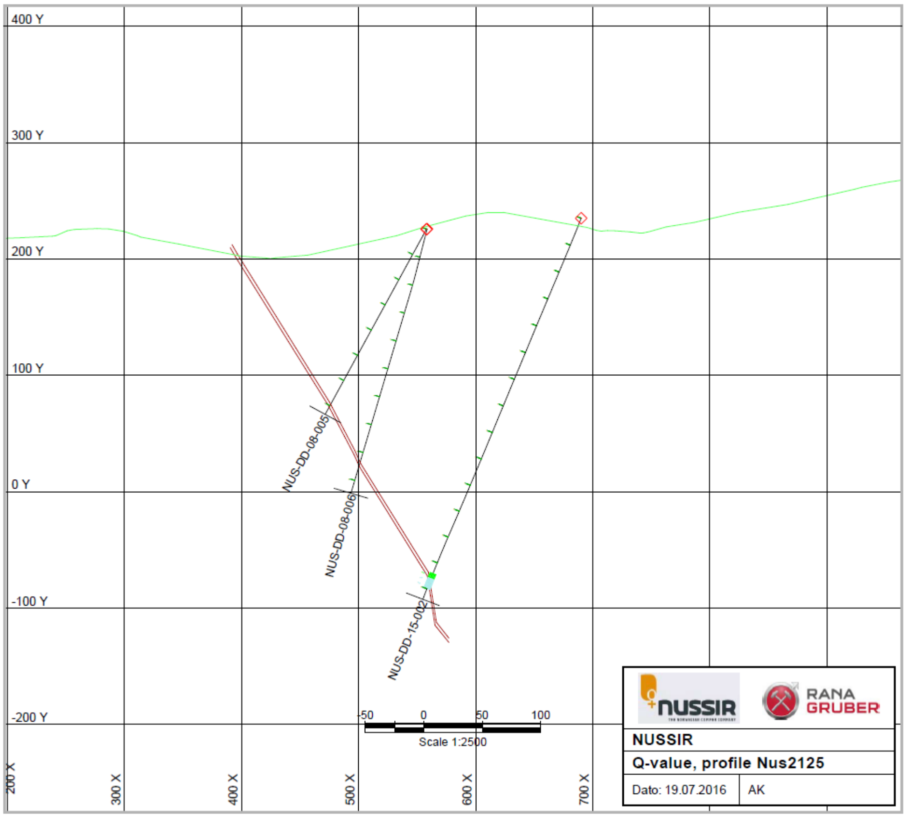

Figure 1-5: Q-values Determined in Drill Hole NUS-DD-15-002, Vertical Section 2125NUS.

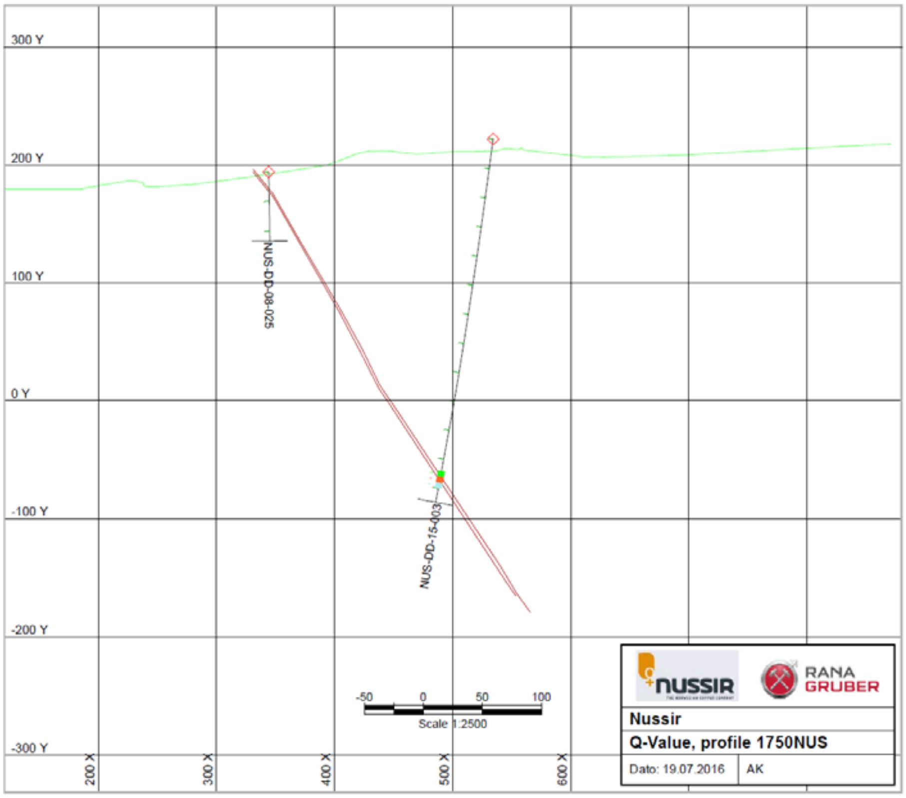

Figure 1-6: Q-values Determined in Drill Hole NUS-DD-15-003, Vertical Section 1750NUS.

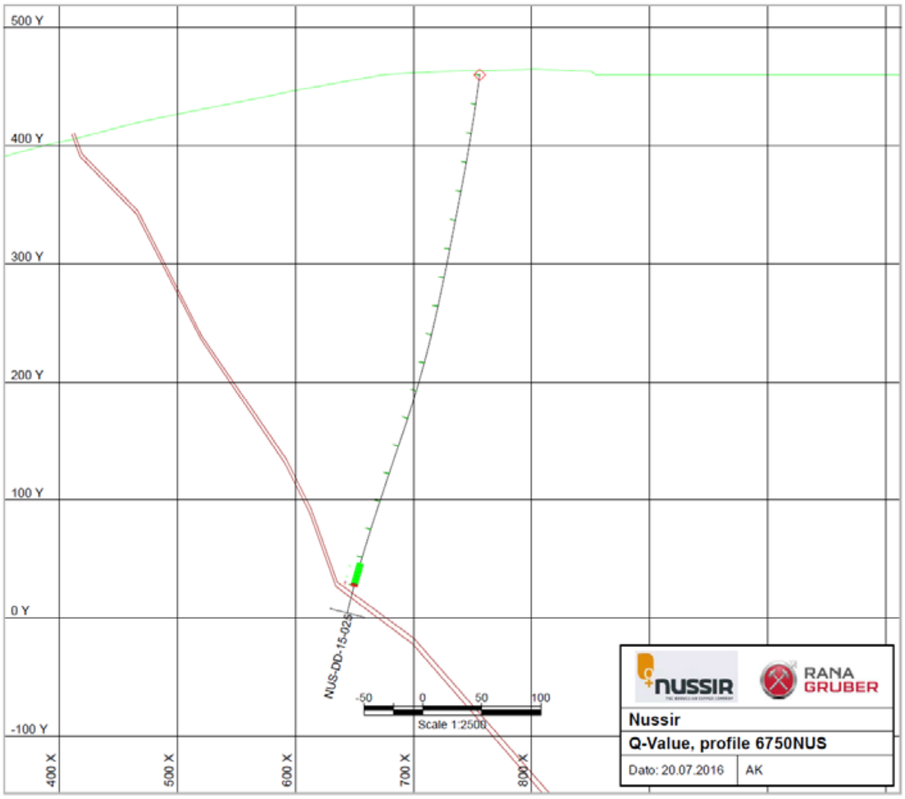

Figure 1-7: Q-values Determined in Drill Hole NUS-DD-15-025, Vertical Section 6750NUS.

1.1.2.5 Interpretation of Results of Geotechnical Testing

The rock property testing and rock mass classification analysis of the Nussir and Ulveryggen indicates that the hangingwall, footwall and orebody are relatively strong and intact rock masses. Regional stress measurements completed both at the site and within the Finnmark region have determined that the principal stress is about 20 MPa and is aligned roughly parallel to the strike of the Nussir deposit. The principal stress is less than about one third of the UCS of the Nussir rocks and only about 10 % of the Ulveryggen test. Further work should be completed to assess the rock mass, particularly testing that satisfies the ISRM standard, however the initial indications that the intact rock strength is much higher than the in-situ stresses.

Mining induced stresses in the vicinity of the proposed mining areas should be modelled to assess the risk to mining and mine stability based on the mining geometry, stope sequence and extraction rates. A 2D modelling exercise completed by Sintef (Sintef, 2016c) is summarised in the following sections, however more extensive work based on continued data collection and rock mass classification should serve as the basis for additional evaluations and stress modelling.

1.1.3 Mining Method Selection

The initial approach to mining method selection was undertaken using the Hartman chart for selecting mining methods (Ref. 5-6: (Hartman, 1992)). The Hartman method differentiates between deep and shallow deposits which separates out the main underground and surface mining methods. Mining method selection was carried out using the Nussir block model and other geological data. The size and shape of the mineralized zones at the Nussir deposit suggest mining by underground methods, and several methods could be applied. These are narrow vein mining techniques, sublevel stoping and vertical crater retreat, listed here in no particular order.

1) In *sublevel open stoping* the orebody is divided into separate, often large stopes. Use of this method assumes good stability of the openings is created. Vertical pillars may be left between stopes to support the hangingwall, and crown pillars (horizontal sections within the mineralized zone) may also be left to support mine workings on levels further above. The stopes are often quite large and most mines aim to maximise the size, this however depends on the stability of the rock mass, which will be the limiting factor during the stope/pillar design. Mining may either be overhand (lower drilling blocks are extracted first) or underhand (upper drilling blocks are extracted first). Drillability and type of equipment selected dictate the vertical distance between the sublevels. A system of draw points is excavated below the stopes for safe mucking with Laud Haul Dumpers (LHD).

2) *Vertical crater retreat* (VCR) mining is based on the crater blasting technique in which explosives are placed in large diameter holes and fired. The ore is drilled by specialised drill rigs (large diameter) from an overcut (drill and charging drift) downwards into the undercut. Holes are charged from the drill drift and sections of a specified thickness are charged and blasted. Holes must be stemmed well prior to blasting. Blasted ore is mucked out in a similar way as with the open stope method.

3) *Room and pillar* mining was originally designed for flat bedded deposits of limited thickness. Ore is recovered in open stopes and pillars are left behind to support the hangingwall. Numerous variations of room and pillar mining exist to adapt to slightly inclined orebodies.

4) *Narrow Vein mining* is carried out using specialised, small scale machines to minimise dilution in narrow vein-deposits. Veins with a thickness of about 2 m and wider can be mined using specialist jumbos drills for narrow drifts, small longhole rigs and small LHDs with a 2 m^3 to 3 m^3 bucket capacity. The unit operations are the same for most other mining systems only the scale of the equipment and the productivities are not as high.

Historic mining method strategies and analyses were also considered and used to inform the mining method selection process. Because of concerns over the mining operating costs and mobile fleet capacity, systems involving fill were not included in the mining method analysis. Principally, however, the current business plan for Nussir includes selling unmineralized waste rock to local aggregate suppliers as feedstock for their plants and the local construction aggregates market, thus making the availability of rock fill materials challenging.

1.1.3.1 Selection Criteria

The following deposit attributes and mining considerations are used to identify suitable mining methods for any given deposit:

- Deposit depth, strike, dip and other geometric properties;
- Productivity and efficiency;
- Recovery and dilution factors;
- Workforce and safety;
- Socio-economic considerations; and
- Operating and capital costs and revenue.

Deposit Geometry

The Nussir deposit under consideration in this study has a strike length of approximately 6 km, a nominal vertical extent of 300 m, dips at 60° from the horizontal towards the northwest and has an average thickness of 4 m. A longitudinal view looking south east (Bearing of 160°) towards Nussir is presented in Figure 5-8.

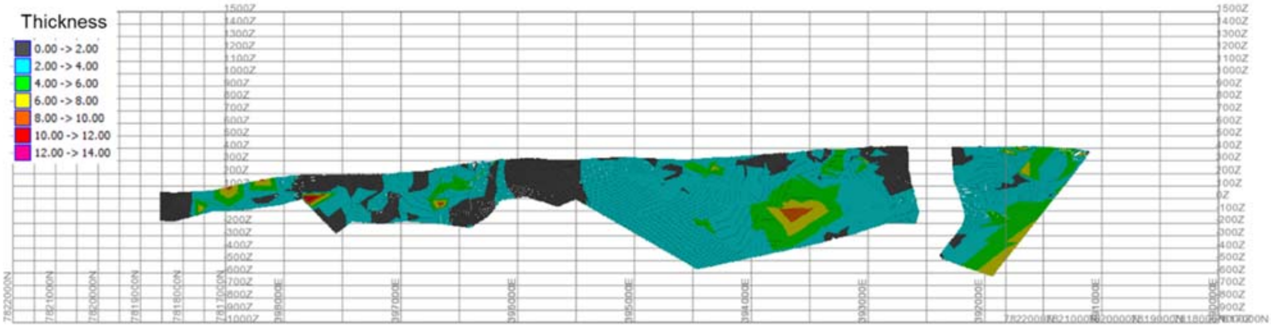

Figure 1-8: Longitudinal View of the Nussir Deposit

Productivity and Efficiency

The relative productivity for each mining method is another key aspect to consider and further reference information on a range of mining method applications is presented in Table 5-6.

Table 1-6: Relative Productivities of Underground Mining Methods

Mining Method	Relative Productivity	Average Tonnes Per Employee Shift
Block caving	Very high	-
Room and pillar mining	High	-
Sub-level stoping	High	100
Sub-level caving	High	85*
Sub-level stoping with fill	High	95*
Vertical crater retreat	Moderate	-
Mechanised cut and fill	Moderate	40*
Conventional cut and fill	Low	20
Shrinkage stoping	Very low	6

* Not from Hartman. This reference is from Golder

Sub-level stoping, VCR or a variant of room and pillar are appropriate methods to evaluate, considering the physical characteristics of the deposit. All of these systems are easily mechanised and automated.

Mining Recovery and Dilution

The relatively narrow widths (ca. 4m) of the Nussir deposit will make managing dilution and mining recovery areas of focus during mine operation. Un-planned dilution is uneconomic rock that enters the plant feed and process stream, reduces the overall grade, and increases the tonnage being handled and processed. It effectively increases the unit cost per metal unit.

Mining recovery is the ability for the ore handling system to deliver mineralized material to the process plant. Equipment specifications, blasting efficiency, stope shape and a number of other factors affect mining recovery. The relative dilution and recovery factors for a number of underground mining methods are listed in Table 5-7.

Table 1-7: Relative Dilution and Recovery Factors of Underground Mining Methods

Mining Method	Relative Dilution Factors*	Relative Mining Recovery Factors*
Block caving	3,0	0,75
Room and pillar mining	1,5	0,70**
Sub-level stoping	2,0	0,90
Sub-level caving	2,5	0,85
Sub-level stoping with fill	1,7	0,95
Vertical crater retreat (VCR)	2,0	0,90
Mechanised cut and fill	1,2	0,95
Conventional cut and fill	1,0	1,0
Shrinkage stoping	1,0	1,0

* Not from Hartman. This reference is derived by Golder **Accounts for the extraction ratio

The mining method should account for dilution and mining recovery. The more selective the system the less dilution and higher mining recovery. The trade-off is with both the productivity of the method and the relative unit operating cost.

Workforce and Safety

Mining methods that minimise workforce exposure time to hazards underground are deemed most suitable. Historic data indicates that rock falls and human-energised equipment (mobile plant) are the principal risks to workers. A mining system that minimises the number of excavations and stope entry points will be most effective at minimising the ground related issues, and remote mining or fully automated and/or mechanised equipment to reduce the workforce underground are mitigations to the risks with more traditional mining methods.

Nussir has indicated that the organisation will be seeking to automate as much of the mining process is as practical based on the availability of suitable equipment from the numerous mining equipment suppliers based in the Nordic countries. Norwegian, Swedish and Finnish operations have demonstrated a willingness to adopt remote, tele-remote and autonomous mining equipment, which reduces workforce exposure to the underground environment.

Socio-economic Considerations

The direct socio-economic impact of a mining method is a measure of the size of workforce required to sustain production. It is somewhat contrary to workforce safety in that the social benefits generally require a larger workforce, however there needs to be a balance between local benefits and the commerciality of a mining project.

Other benefits to a local community come in forms other than direct employment such as suppliers, service workers and other benefits derived from increased industrial activity in a region.

Operating and Capital Costs and Revenue

The cost of production is a key aspect to consider and further reference information on a range of mining method applications, as reported by Hartman (Hartman, 1992), is shown in Table 5-8.

Table 1-8: Relative Mining Costs of Underground Mining Methods

Mining Method	Relative Cost
Block caving	1,0
Room and pillar mining	1,2
Sub-level stoping	1,3
Sub-level caving	1,5
Sub-level stoping with fill	1,7
Vertical crater retreat (VCR)	4,3
Mechanised cut and fill	4,5
Conventional cut and fill	9,7
Shrinkage stoping	6,7

Mining Methods Considered

The proposed production target of 2 Mtpa (peak annual production) results in a daily production rate of almost 5 500 t. This rate calls for an efficient and mechanised mining method, which rules out narrow vein mining techniques.

The three mining methods presented below are discussed in the terms of their application, advantages and disadvantages in Table 5-9.

Sub-level Stoping

Based on the above discussion it is recommended that a slightly modified open stope mining method is employed for both the Nussir and Ulveryggen orebodies, as described in the sections below.

Vertical Crater Retreat

Vertical crater retreat (VCR) also has its disadvantages for narrow mineralization such as Nussir. In VCR, mining the orebody has to be completely drilled over large sections before extraction can commence. The disadvantage here is that a large development and production drilling program must be completed before production can start. Additionally, drilling and blasting using the VCR method requires greater drilling and charging precision and expertise relative to conventional open stope blasting.

Room and Pillar

The deposit dips at a difficult operating angle of 60° (towards the northwest), which is too steep to operate mechanised equipment and too shallow to ensure continuous ore flow on the footwall of the stopes. There are design techniques and mining methods adapted to steeper dipping orebodies, inclined room and pillar in particular.

Table 1-9: Mining Method Criteria Application

Mining Method	Suitable Application	Project Advantage	Project Disadvantage
Sub-level stoping	■ Minimum mineralized zone width of 3 m - 6 m. Can be as little as 1.5 m and as much as 30 m. ■ Regular and continuous zone shape. ■ Steeply dipping mineralized zone (>50°). ■ Competent hangingwall and footwall. ■ Ore is competent to work on and under.	■ Easily mechanised. ■ Moderate to high productivity. ■ Moderate to high production is achievable. ■ Economic material can be drawn off immediately. ■ Low labour intensity. ■ Fair – high recovery (75 – 90%). ■ Large equipment can be utilised. ■ Low to moderate mining/production cost. ■ Low risk mining method as personnel do not enter stope. ■ Repetitive procedure offering training and safety advantages. ■ Country rocks and mineralization are relatively strong.	■ Large amount of development required resulting in relatively high and early capital costs. ■ Non selective leading to potential for fair dilution (typically 20% for narrow stopes). This will increase with increasing the distance between sublevels. ■ Strong engineering and technical support required, particularly with longhole drill planning. ■ Not flexible if mineralized zone is not continuous or undulates and its thickness is quite variable over short distances. ■ Long blind-up holes may reduce mining recovery and increase dilutoin due to drilling accuracy for up-holes >25m
Vertical Crater Retreat	■ Minimum mineralized zone width of 3 m - 6 m. Can be as little as 1.5 m and as much as 30 m. ■ Tabular or lenticular or massive mineralized zone shape. ■ Steeply dipping mineralized zone (>50°). ■ Competent hangingwall and footwall ■ Ore is competent to work on and under. ■ Ore and surrounding rocks are strong enough to sustain repeated blasts	■ Same as for longhole open stoping	■ Same as for longhole open stoping aside from up-hole drilling issues. ■ Requires specialist drilling, charging and blast planning skills. ■ Is labour intensive relative to longhole due to need for repeatedly loading small quantities of explosives for blasting ■ Proposed stope height may compromise necessary drilling accuracy.

Mining Method	Suitable Application	Project Advantage	Project Disadvantage
			■ Repeated blasting may damage the hangingwall and footwall, increasing dilution.
Room and Pillar	■ Mineralized zone width of >3m - 10m; should be fairly large in lateral extent. ■ Ideal for tabular or continuous mineralized zone shape. Can be applied to irregular shaped mineralized zones. ■ Mineralized zone dip from 0° to 45°. ■ Requires strong ore to serve as pillars. ■ Requires reasonably high grade ore to support relatively higher mining costs. ■ Suitable where there is a requirement for selective mining with poor mineralization continuity.	■ Suitable for mechanisation. ■ Moderate productivity and production rate. ■ Moderately safe as exposed and unsupported areas are minimised. ■ Large scale ground movement is minimised. ■ Selective method, poor to fair recovery (40 – 80% if some pillars extracted) with minimal dilution (5 - 10%).	■ Labour intensive; requires lots of workplaces to be productive. ■ Low resource recovery ratio as ore is tied up in pillars. ■ Requires consistently strong ore rocks for local roof/wall support. ■ Minimal open volume available to serve as alternative mining areas for production flexibility. ■ Moderate cost for mechanisation due to requirement for many points of access due to minimal open volume. ■ Fairly high mining cost.

Discussion

Room and pillar at a 60° dip would require a fleet of specialised equipment to push ore down the footwall as there would be no gravity induced flow using this method. It is readily mechanised but automation would be difficult and developing enough production faces would require a lot of time and a lot of equipment. This method was discounted for both Nussir and Ulveryggen.

VCR would be difficult to adapt to a narrow orebody. VCR requires roughly parallel drill holes, which is the planned blast design for Nussir, however the potential for damage to the footwall and hangingwall due to repeated blasting on the same rings may increase dilution and decrease mining recovery. It also requires specialist level blast design, charging and timing skills. Whilst possessing virtually identical attributes to longhole stoping, the need for larger diameter drill holes to ensure that the spherical charge unit weight is adequate would make this non-ideal from a stability and operability view for both Nussir and Ulveryggen due to mining widths and productivity.

Sub-level open stoping (SLOS) is a widely recognised and easily adapted productive and efficient mining system. The mining system requires sustained drilling, charging, blasting and loading which is where the productivity and efficiency are most realised. Numerous blasts of a number of rings daily will ensure that there is adequate tonnage to maintain daily production.

SLOS is also easily adapted to varying widths and dips of stopes and orebodies, albeit between stopes and not within the stopes. The equipment is proven and amenable to automation and remote operation. Hence an open stoping mining method was selected as the basis for mine productivity and design for Nussir and Ulveryggen.

1.1.4 Nussir Mine Design

Finding a stable but efficient stope size is important when implementing SLOS. The stope width is determined by the thickness of the orebody, which varies between 3 to 5 m. Where the ore is narrower than 3 m the stopes will not be economically viable to mine. Therefore the main parameters for dimensioning the stopes are their horizontal length along strike and vertical heights. The vertical height is also constrained due to a maximum drilling length and the wish to minimise the number and overall length of the drill drives. Vertical and horizontal sill pillars are introduced between the stopes to improve ground stability. Introduction of point pillars might be considered when necessary. This is to avoid the need to cable bolt the hangingwall, which is a lengthy process.

1.1.4.1 2D Modelling for Stope Design

This section of the report has been composed by Golder based on data collection and analyses completed by SINTEF in 2016 which were presented in a draft report submitted to Rana Gruber (SINTEF, 2016). Some of the content has been extracted directly from the work completed by SINTEF, particularly figures and tables, and is noted in the text. The following is a synthesis and summary of that work and draft report.

2014 and 2015 Geotechnical Data Collection and Analyses

SINTEF completed a site visit and core logging exercise in collaboration with Nussir staff in August and September of 2016, (SINTEF, 2016). This was subsequent to stress measurements collected in winter 2016, (SINTEF, 2016). The data collection work completed through 2015 and 2016 was used to inform a 2D stress modelling exercise to confirm the dimensions of the stopes, and rib (vertical) and sill (horizontal) pillars between the stopes.

The SINTEF work included an analysis of rock properties for footwall mineralization and the hangingwall of the Nussir deposit in the vicinity of Panel 1 on geological section 2250, as illustrated in Figure 5-9.

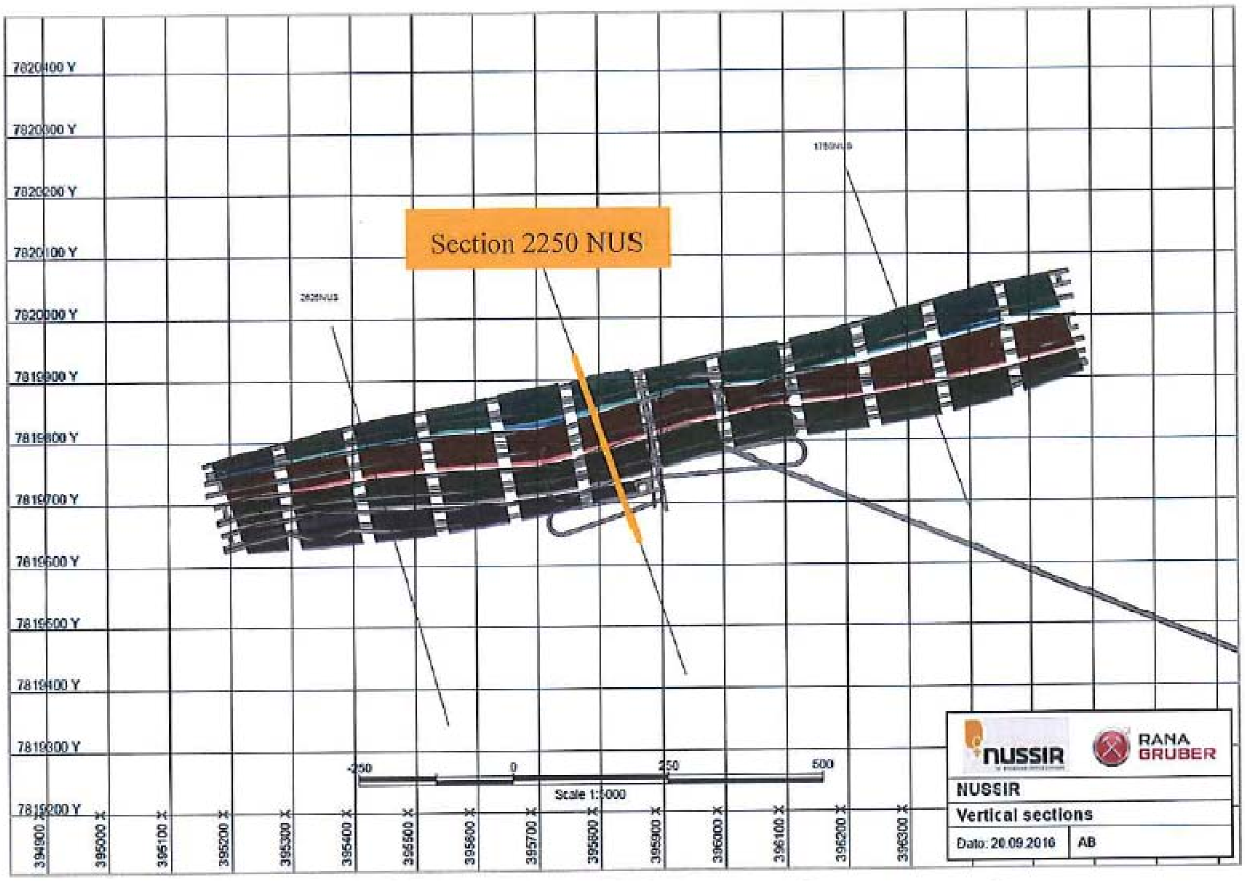

Figure 1-9: Nussir Layout of the Panel 1 Stopes and Location of Section Analysed (SINTEF, 2016c)

Using data collected as part of the 2015 and 2016 exploration programme completed by Nussir, SINTEF completed a numerical analysis of the stopes and pillars based on the mine design for Panel 1 produced by Rana Gruber. The 2D modelling analyses the post-mining stresses around the pillars and stopes. The modelling was used as an input to the empirical stope design method referred to as the Mathew's Stability Graph Method. The modelling was performed assuming that longhole open stoping (LHOS) was the mining method.

Rock Mass Properties and In-Situ Stress Conditions

Laboratory test results, core logging data and stress measurements were completed by SINTEF as part of the PFS work program. The cross-section modelled and the relative locations of the exploration drill holes used for laboratory testing are indicated on Figure 5-10, along with the location of the in-situ stress measurements. Drill holes NUS-DD-08-11 and NUS-DD-15-BH1 were the source of the rock samples for testing the geotechnical properties of the rock masses.

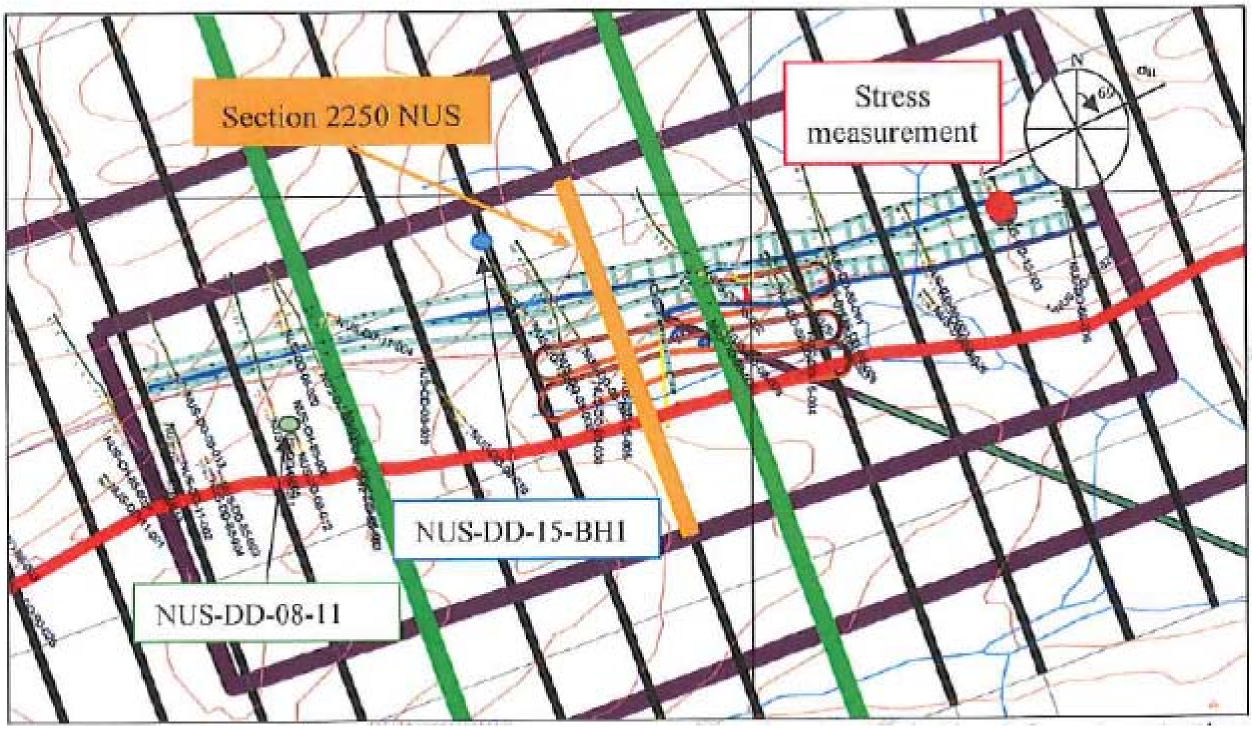

Figure 1-10: Location of the stress measurements, core logging and laboratory test sample drill holes (SINTEF, 2016c)

The results of the rock property testwork is summarised in Table 5-10.

Table 1-10: Laboratory test results on rock properties (SINTEF, 2016c)

Borehole	Length (m)	Lithology	E (GPa)	Poisson	UCS (MPa)	Density	PLT (MPa)
NUS-DD-15-BH1	397,75-398,3	Sandstone	75,1	0,19	123,3	2 724	19,6
NUS-DD-15-BH1	405,3-408,5	Sand/silt/claystone	70,9	0,2	132	2 685	-
NUS-DD-08-011	93,6-95,0	Sandstone	-	-	125	-	-
NUS-DD-08-011	135,4-142,1	Sandstone	-	-	92	2 740	-
		Average	*73*	*0,2*	*118,1*	*2 716*	*19,6*
		Standard Deviation	2,1	0,01	15,4	23	6,3
		% Standard Dev.	3 %	3 %	13 %	1 %	32 %

A rock mass quality evaluation was completed using the core logging information collected from borehole NUS-DD-15-BH1, 002 and 003 (SINTEF, 2016b). The data was collected from fresh core and logged in collaboration with Nussir field geologists to ensure that rock types were consistent with Nussir lithology and other geological logging methods. The respective inputs to the Q data was collated and tabulated then the Q-value for each logging interval was calculated by SINTEF.

Table 5-11 presents the results of the core logging and rock mass quality parameters using the Q-method collected by SINTEF and Nussir.

Table 1-11: Rock mass assessment using the Q-method (Ref. 5-7 (NGI, 2013), SINTEF.2016c)

Borehole	Depth		Length	Joint freq. (/m)	RQD (%)	JN	Jr	Ja	Jw	SRF	Q
	from	to	(m)								
NUS-DD-15-001	365	366	1	3	95	2	1	1	1	1	48
	366	367	1	5	90	2	1	1	1	1	45
	367	368	1	5	90	2	1	1	1	1	45
	368	369	1	2	95	2	1	1	1	1	48
	369	370	1	2	90	1	4	0,75	1	1	480
	370	371	1	0	100	2	1	1	1	1	50
	371	372	1	4	95	2	1	1	1	1	48
	372	373	1	4	95	2	1	1	1	1	48
	373	374	1	4	95	2	1	1	1	1	48
	374	375	1	2	95	2	1	1	1	1	48
	394	395	1	4	90	2	1	1	1	1	45
	395	396	1	2	100	2	1	1	1	1	50
	396	397	1	1	100	2	1	1	1	1	50
	397	398	1	1	100	2	1	1	1	1	50
	398	399	1	1	100	2	1	1	1	1	50
	404	405	1	1	100	2	1	1	1	1	50
	405	406	1	5	95	2	1	1	1	1	48
	406	407	1	4	95	3	1	1	1	1	32
	407	408	1	3	95	3	1	1	1	1	32
	408	409	1	4	95	3	1	1	1	1	32
NUS-DD15-002	335	340	5								33
	340	345	5								47
	345	350	5								9
NUS-DD15-003	285	289.7	4.7								16
	289.7	295	5.3								1
	295	300	5								50

The average values of the various domains are compiled in Table 5-12.

Table 1-12: Average Q-Values by Rock mass Domain (NGI, 2013)) by SINTEF

Domain	Q-avg	Std	% Std
Hangingwall	74	118	160 %
Ore	48	1	3 %
Footwall	48	48	100 %

SINTEF performed stress measurements in two different locations (SINTEF, 2016a). The first measurement was obtained by hydraulic fracturing within the proposed PFS start-of-mining area using borehole NUS-DD-15-003 (refer to Figure 5-10). The second measurement was obtained in the existing tunnel at Ulveryggen, below the mining area, using a 2-D over-coring method.

For the purposes of the PFS, only the test performed within the proposed Nussir mining area in Panel 1 was used to inform the mine design. The results of the stress analyses completed from borehole NUS-DD-15-003 were:

- Maximum horizontal stress, σ_H = 19,1 MPa
- Secondary horizontal stress, σ_h = 9,2 MPa

Vertical stress, σ_V is the theoretical vertical stress as a function of depth, there the load is given by rock density [t]/[m^3] x depth below surface [m].

The orientation of σ_H is N69°, which is nearly perpendicular to the section to be analysed and roughly parallel to the strike of the Nussir deposit. The stress tensor σ_h is perpendicular to σ_H and therefore parallel to the model plane.

Numerical Modelling

Numerical modelling of the proposed stope designs and pillar arrangements was completed by SINTEF based on the orebody geometry and stopes provided to SINTEF by Rana Gruber. The typical arrangement of the stopes were labelled A, B, C and D, as shown in Figure 5-11. The stopes were planned to be mined from the bottom-up, from Stope D up to Stope A.

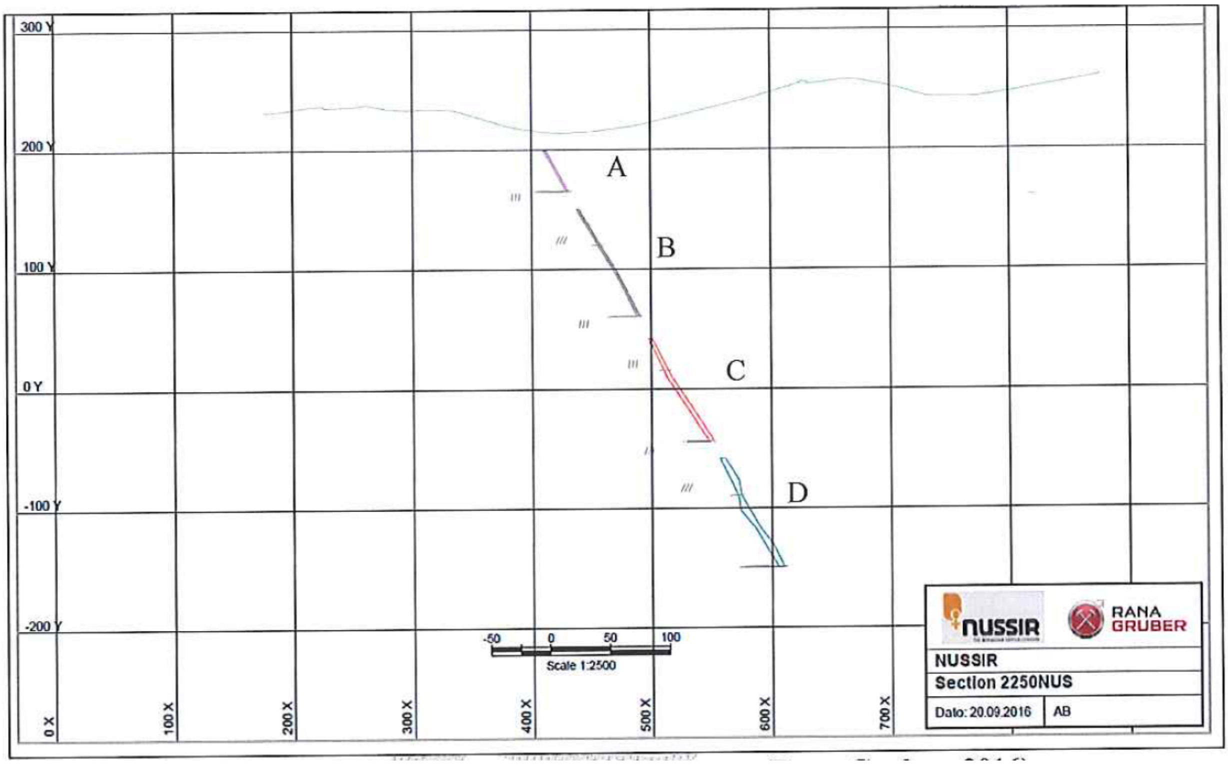

Figure 1-11: Orebody in section 2250NUS shown defined stopes (SINTEF, 2016c))

SINTEF set the stope width to 6 m for the purpose of modelling and the inclination of the stopes was set to 60°, which was based on average orebody geometry in the vicinity of the cross-section and Panel 1. Three scenarios were modelled using varying sill pillar thicknesses of 15 m, 20 m and 30 m. The dimensions of the stopes are presented in Table 5-13. Note that the numbers have been rounded.

Table 1-13: Scenarios modelled for varying sill pillar dimensions (SINTEF, 2016c)

Stope	Stope Phases	Sill Pillar 15m		Sill Pillar 20m		Sill Pillar 30m	
		Total height (m)	Excavation stope (m)	Total height (m)	Excavation stope (m)	Total height (m)	Excavation stope (m)
A	2	39	20	37	18	32	16
B	3	89	30	84	28	74	25
C	3	89	30	84	28	74	25
D	3	87	29	84	28	79	26

The numerical model assumed the ground surface to be horizontal at an elevation of 230mASL. Figure 5-12 presents the numerical model configuration for a 20 m thick sill pillar. Table 5-14 summarises the criteria used for the model simulations.

Table 1-14: Criteria used in model simulations (SINTEF, 2016c)

Model ID	Sill pillar (m)	Failure criterion
Model#1	15	Mohr-Coulomb - Elastic
Model#2	20	Mohr-Coulomb - Elastic
Model#3	30	Mohr-Coulomb - Elastic
Model#4	20	Mohr-Coulomb - Elastic-Plastic

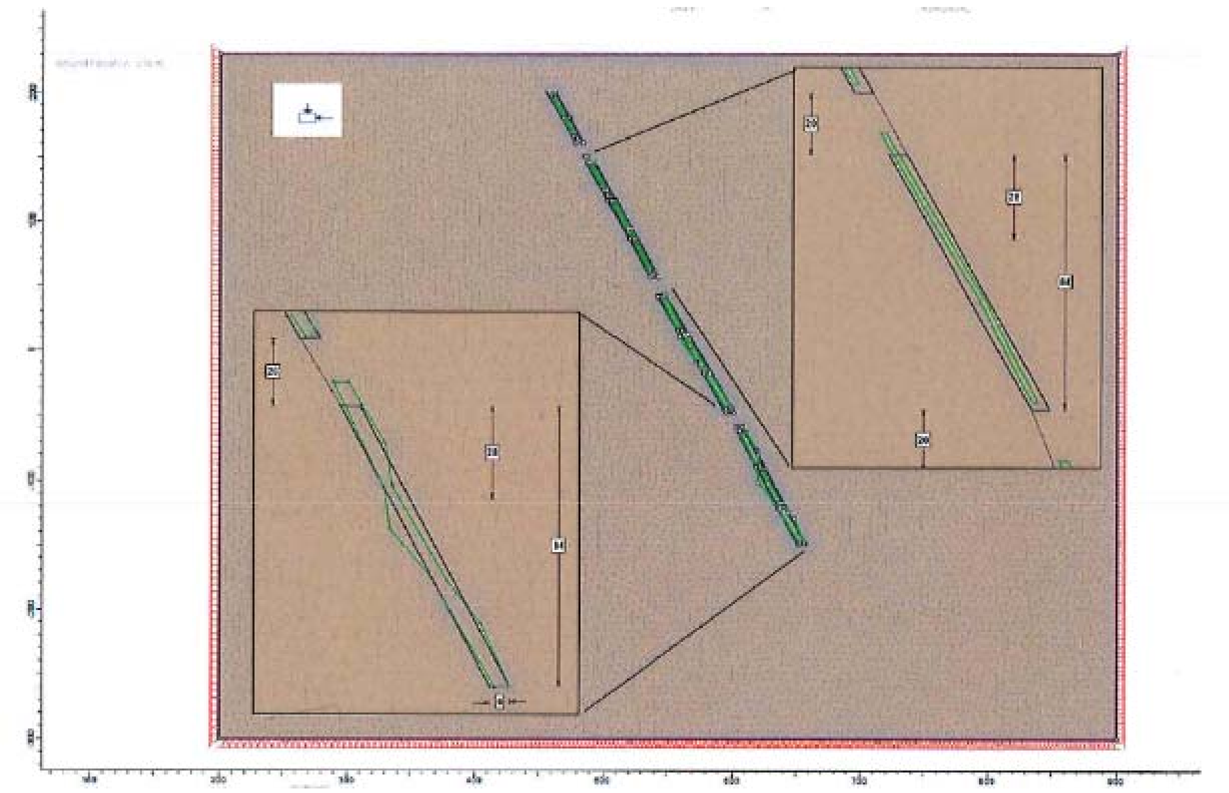

Figure 1-12: Numerical model configuration for model#2 (20m sill pillar) (SINTEF 2016c)

Core logging data and laboratory test results were used to estimate the rock mass properties. SINTEF used recognised methods to estimate the rock mass parameters for input into the numerical model (Aydan et al, 1993; Aydan and Kawamoto, 2001; Hoek et al., 2002; Hoek and Diederischs, 2006; Barton, 2006; Palmstrom, 2009). Table 5-15 summarises the rock mass qualities used as inputs to the numerical modelling runs.

Table 1-15: Rock mass properties used for numerical modelling (SINTEF, 2016c)

Rock mass property	Units	Value	
		Peak	Residual
Density	Kg/m^3	2 700	
Deformation modulus (E_m)	GPa	26	
Poisson's ratio	-	0,2	
Tensile strength	MPa	9	3
Friction angle	°	60	18
Cohesion	MPa	11	3
Dilation angle	°	-	0

The in-situ stress conditions used for the numerical model were based on the testing completed in 2016 (SINTEF 2016). Table 5-16 presents the principal in-situ stress applied to the model using 230 mASL as the surface elevation. The principal stress is perpendicular to the model plane.

Table 1-16: In-site stress conditions used for numerical modelling (SINTEF, 2016c)

Stress	Value	Orientation
σ_2	$1.6 \cdot \sigma_v$	Horizontal
$\sigma_3 = \sigma_v$	Gravity	Vertical
σ_1	$3.4 \cdot \sigma_v$	Horizontal/perpendicular to the model plane

2D Modelling Results and Discussion

The 2D modelling results indicate the potential for high stress concentration in the sill pillars in Panel 1. The deepest sill pillar between stopes C and D was determined to be the most critical, and was further analysed to assess the pillar and stope stability. Figure 5-13 shows the main stress distribution in Model#2 (20 m thick sill pillars) and the vertical stress, σ_v. Figure 5-14 shows the main stress distribution in the opening of the first excavation stage for Stope D, which would be the first lift mined in the Panel.

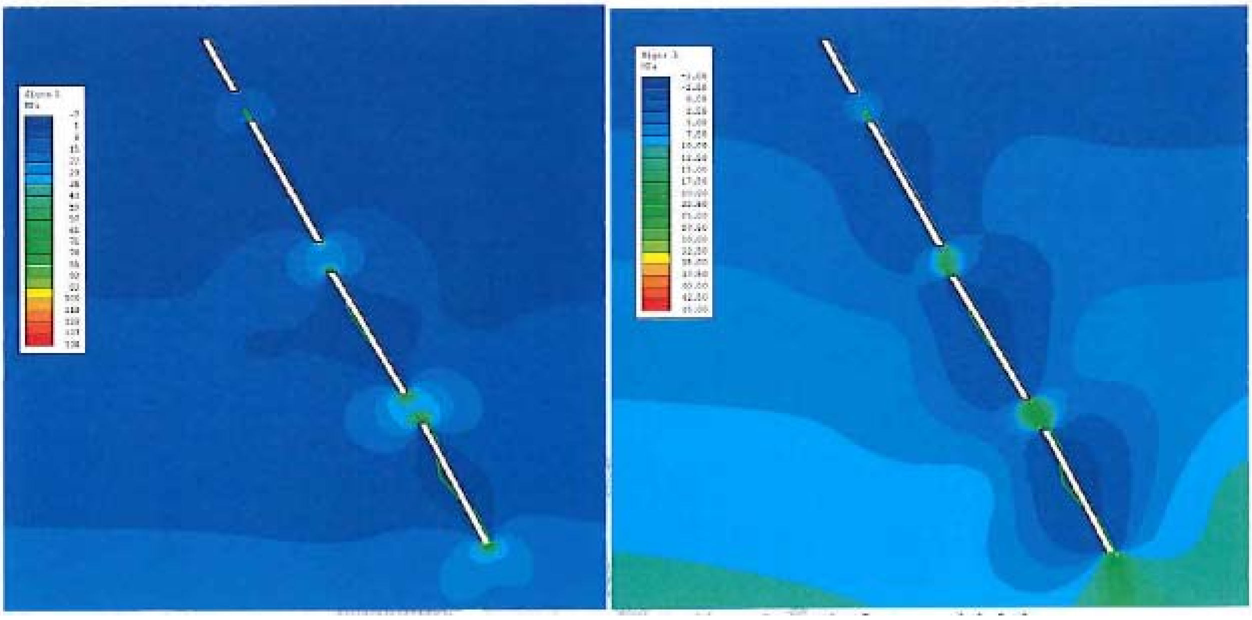

Figure 1-13: Model#2 (20 m sill pillars); Principal stress (left) and vertical stress (right) (SINTEF, 2016c)

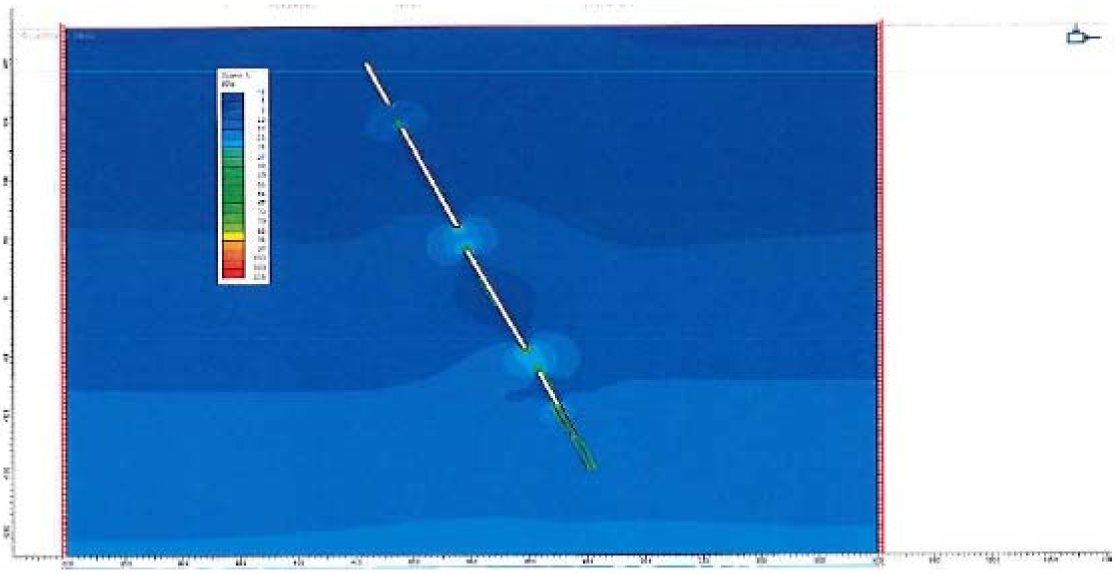

Figure 1-14: Model#2 (20 m sill pillars); Principal stress in the opening of the first excavation stage for Stope D (SINTEF, 2016c)

Figure 5-15 illustrates the principal stress distribution in the sill pillar between stopes C and D for Model#2.

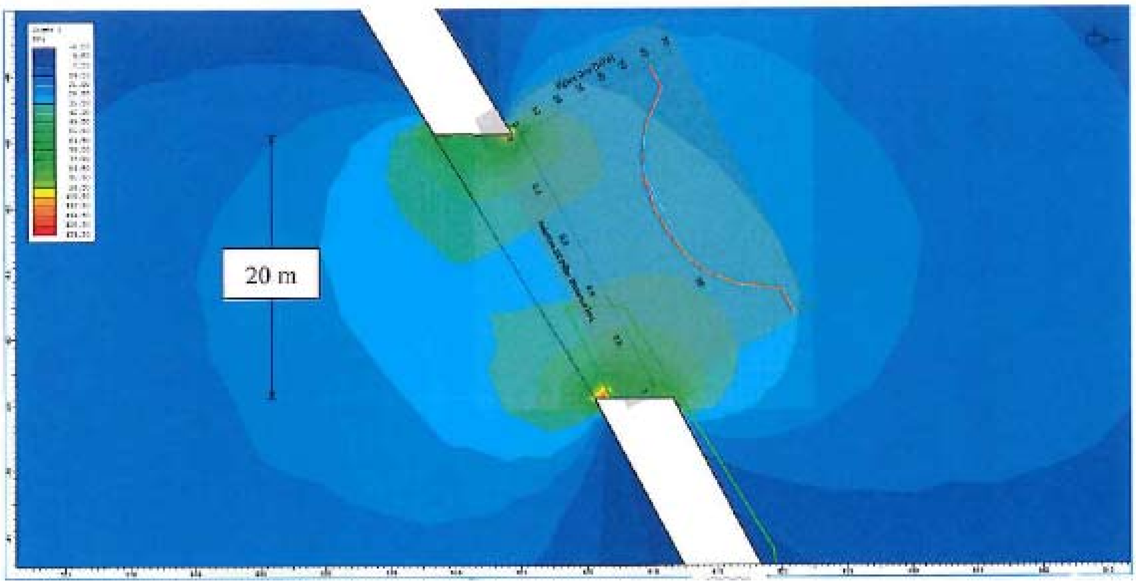

Figure 1-15: Model#2 (20 m sill pillars); Stress concentration in the deepest sill pillar (between stope C and D) (SINTEF, 2016c)

Figure 5-16 charts a comparison between the main stress induced between stopes C and D for the three sill pillar thicknesses analysed. The greater the sill pillar thickness, the lower the peak stress. An average stress value is taken from the central part of the pillar.

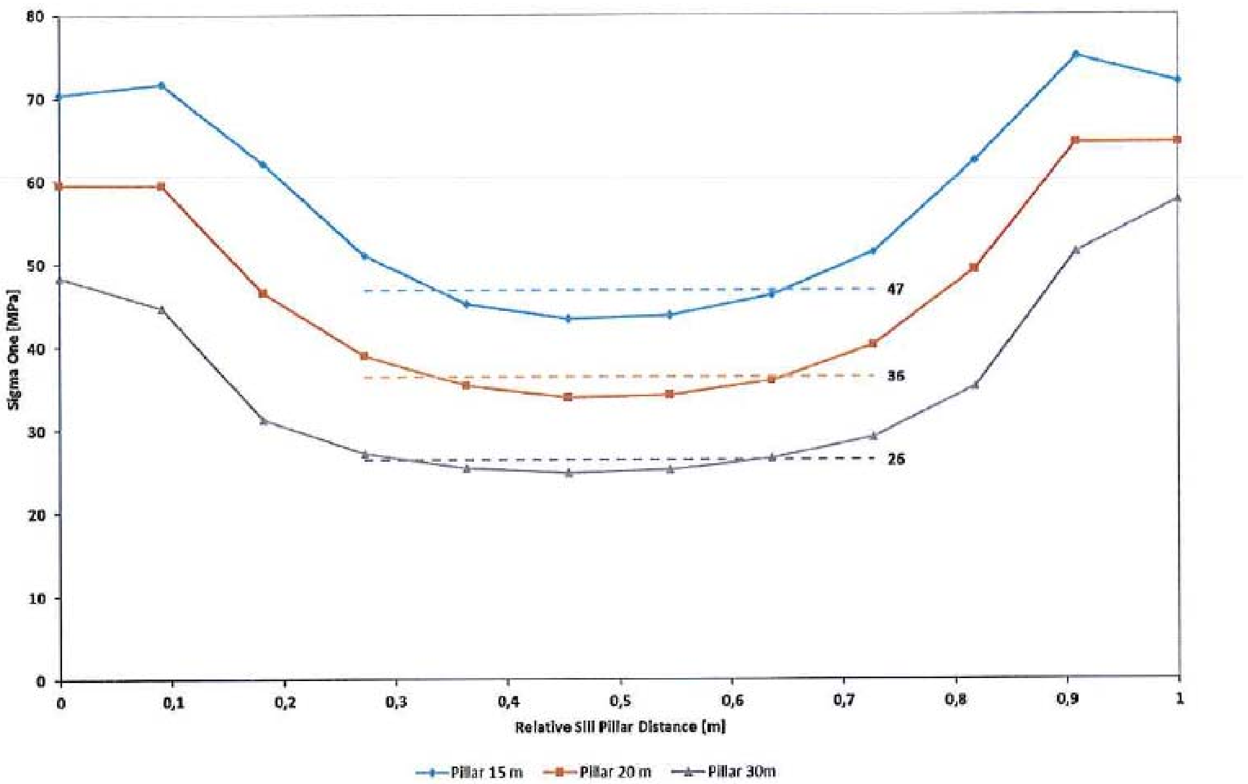

Figure 1-16: Stress concentration between Stopes C and D for 15 m, 20 m and 30 m sill pillars (SINTEF, 2016c)

Model#4 was run using the Mohr-Coulomb failure criterion with elastic-plastic behaviour. The results from the modelling with a 20 m sill pillar indicated yielded zones within Stope D, and potential failures in the roof and floor of Stope D after Stope C has been mined out. The model simulation results are presented in Figure 5-17 to 5-19.

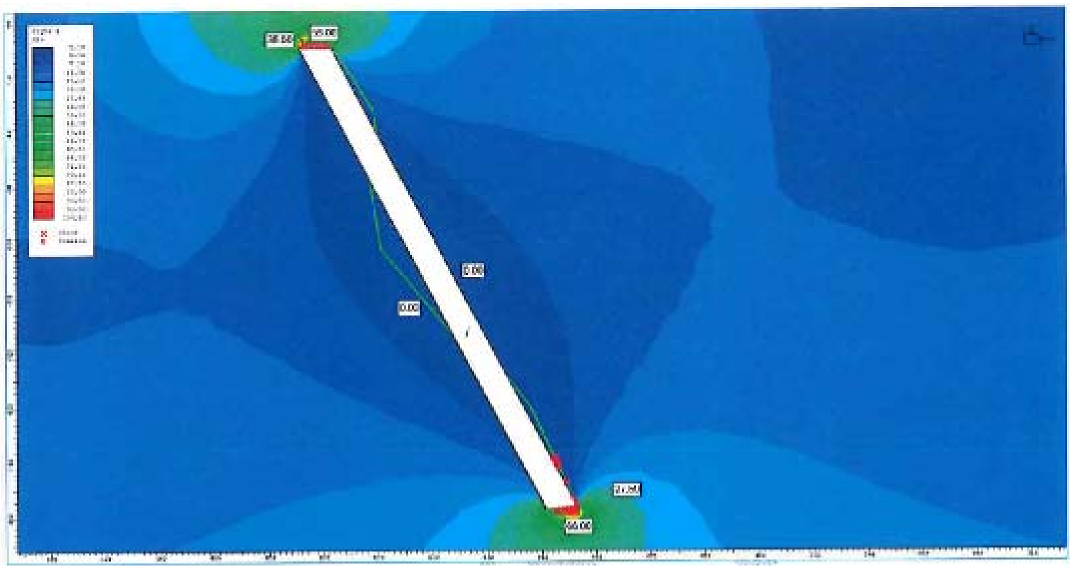

Figure 1-17: Model#4 - Sigma 1 and yielded zone for Stope D (SINTEF, 2016c)

Figure 1-18: Model#4 - Vertical stress and yielded zone for Stope D (SINTEF, 2016c)

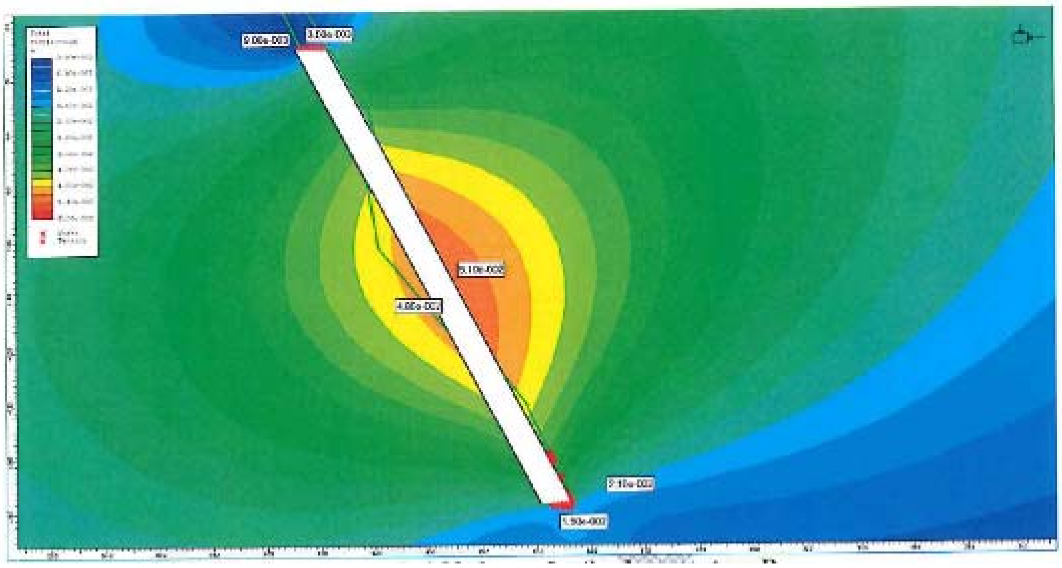

Figure 1-19: Model#4 - Displacements and yielded zone for Stope D (SINTEF, 2016c)

Mathew's Stability Graph for Stope Design

SINTEF used the empirical design method Mathew's Stability Graph to assess the viability of stope geometries and to determine the maximum stable span. The stability graph method is widely used in underground hard rock mines (Ref.5-8: (Mitri et al., 2011)).

The method uses the hydraulic radius (HR) of the critical face and the modified stability number (N'), proposed by (Ref 5-9: (Potvin (1988)), to estimate the stability of unsupported and supported underground openings. The modified stability number (N') is represented by the following equation:

$$N' = Q' \cdot A \cdot B \cdot C$$

Where:

$$Q' = \frac{RQD}{Jn}\frac{Jr}{Ja}$$

- A is the factor to include the effect of induced stress;
- B accounts for the weakness due to the direction of the dominant joint system; and
- C takes into account the orientation of the critical face.

SINTEF analysed the 20 m sill pillar scenario, and the results are presented in Table 5-17. There was no specific information regarding the rock mass structure (joints/faults) at the time of writing and so the parameters B and C were input as minimum and maximum values for the parameters A, B and C.

Table 1-17: Stability analysis for 20m sill pillar (SINTEF, 2016b)

	x	y	Area	Perimeter	HR	σ_{max}	σ/UCS	A	B_{min}	B_{max}	C_{min}	C_{max}	N'_{min}	N'_{max}
Stope Crown	6	100	600	212	2,8	70	0,6	0,18	0,2	1	2	8	4	88
Footwall/ Hangingwall	95	50	4 750	290	16,4	28	0,2	0,46	0,2	1	2	8	11	225

Figure 5-20 and Figure 5-21 show that the crown is stable, and the footwall/hangingwall side is potentially unstable, even in the case with cable bolts as rock support. Note that the use of the term “Caving” on the Stability Graph is interpreted as meaning unstable.

The maximum stable spans for the footwall and hangingwall are presented in Table 5-18. This has been calculated based on the possible range of values that were used to calculate N’.

Table 1-18: Maximum permissible span for the footwall/hangingwall (SINTEF, 2016b)

	x	y	Area	Perimeter	Hydraulic radius	N'
N'_{min}	50	35	1 750	170	10,3	11
N'_{max}	95	60	5 700	310	18,4	225

The maximum stable hangingwall and footwall spans in Table 5-18 are plotted on Figure 5-20 and Figure 5-21, and along with the stope crown and mean design hangingwall and footwall spans.

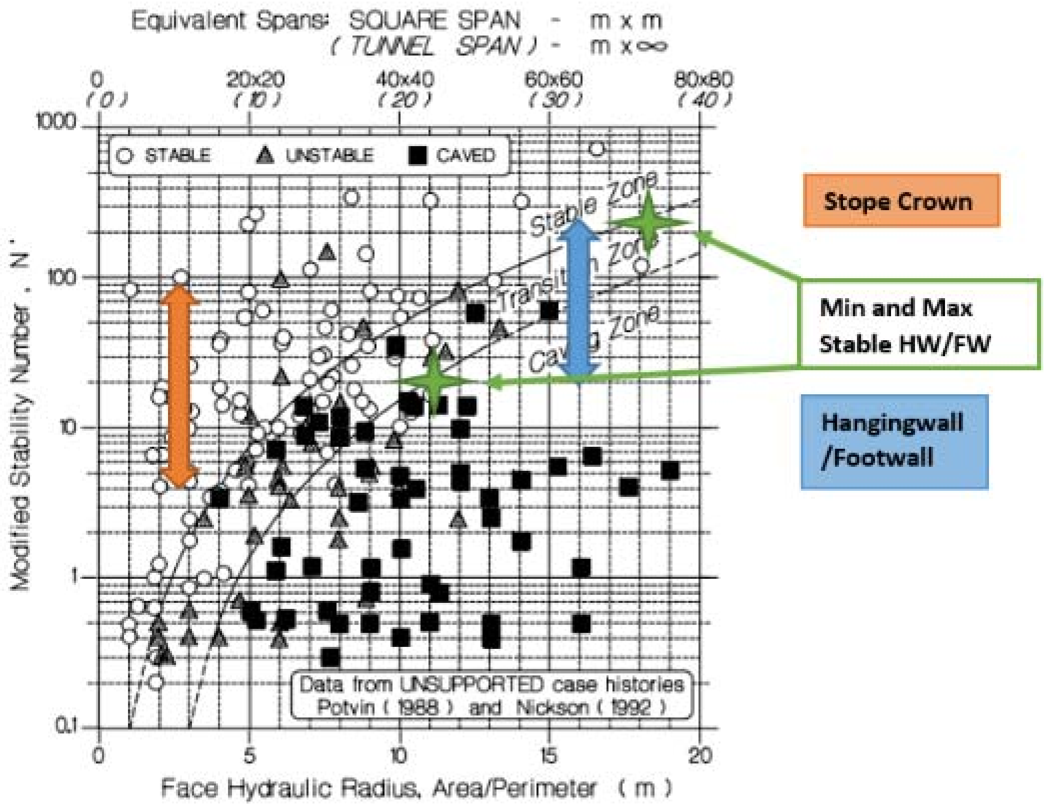

Figure 1-20: Modified stability graph – limits of cable bolt effectiveness as updated by (Ref 5-10 (Nickson (1992)

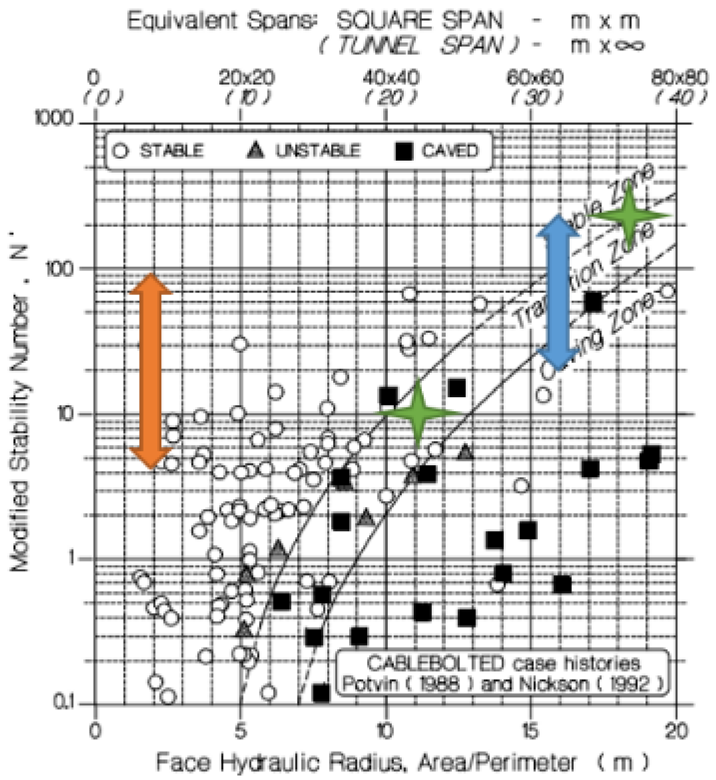

Figure 1-21: Modified stability graph – limits of cable bolt effectiveness as updated by (Ref 5-10 (Nickson (1992))

1.1.4.2 Conclusions and Recommendations

SINTEF concluded the following:

- There is stress concentration in the sill pillars. The induced stress in the deepest sill pillar (between Stopes C and D) reaches an average value of 47 MPa for a 15 m sill pillar, 36 MPa for a 20 m sill pillar, and 26 MPa for a 30 m sill pillar. These values are all less than the minimum rock strength values for the test carried out, but greater than the tensile strengths of the same lithologies;
- The 2D elastic-plastic modelling (Model#4) of the typical cross-section results indicate that there a risk of failure in the roof and floor of Stope D for the scenario which employs a 20 m thick sill pillar after Stope C has been mined; and
- Based on the results of the 2D elastic modelling (Model#2) of the pillars, 20 m thick sill pillars are stable and pose minimal risk of failure.

Golder also noted some conclusions separate from the SINTEF findings.

Golder completed a separate statistical analysis of the SINTEF and Nussir rock quality data to determine the following average values (Table 5-19) for the Q-Values found in Table 5-11. Interval five from 369 to 370 mBS in drillhole NUS-DD15-001 was excluded from the analysis.

Table 1-19: Average Q-Values by Rock Mass Domain (NGI, 2013) by Golder

Domain	Q-Average	Standard Deviation	% Std. Dev.
Hangingwall	28	15,14	54 %
Ore	49	1,11	2 %
Footwall	21	12,45	59 %

The rock mass is comprised of three distinct domains that correlate to the hangingwall, footwall and Nussir orebody. Of the three domains, the orebody has the highest Q-value with the hangingwall second followed by a nominally weaker footwall. Analyses, modelling and design will be based on three distinct geotechnical domains in all future geotechnical and rock mechanics studies of the Nussir deposit.

The Q-system is a logarithmic scale rock mass classification system, so small changes in the Q-value can represent significant changes to the rock mass quality. The SINTEF estimate of 74 for the hangingwall is a full category (Good versus Very Good) higher than the interval length weighted average calculated by Golder. This impacts on the following aspects of the project:

- Maximum stope dimensions including length and height could be overestimated based on the SINTEF findings; and
- Ground reinforcement costs for stopes could negatively impact the mine operating costs.

Golder re-evaluated the Stability Graph to assess the impact of the SINTEF calculations and averages against the updated values produced by Golder. The results of the Golder updated Q-value calculations are presented in Figure 5-22.

The analyses completed by Golder as part of the review of the SINTEF work indicates that the hangingwall Q-values are quite different between the two analyses.

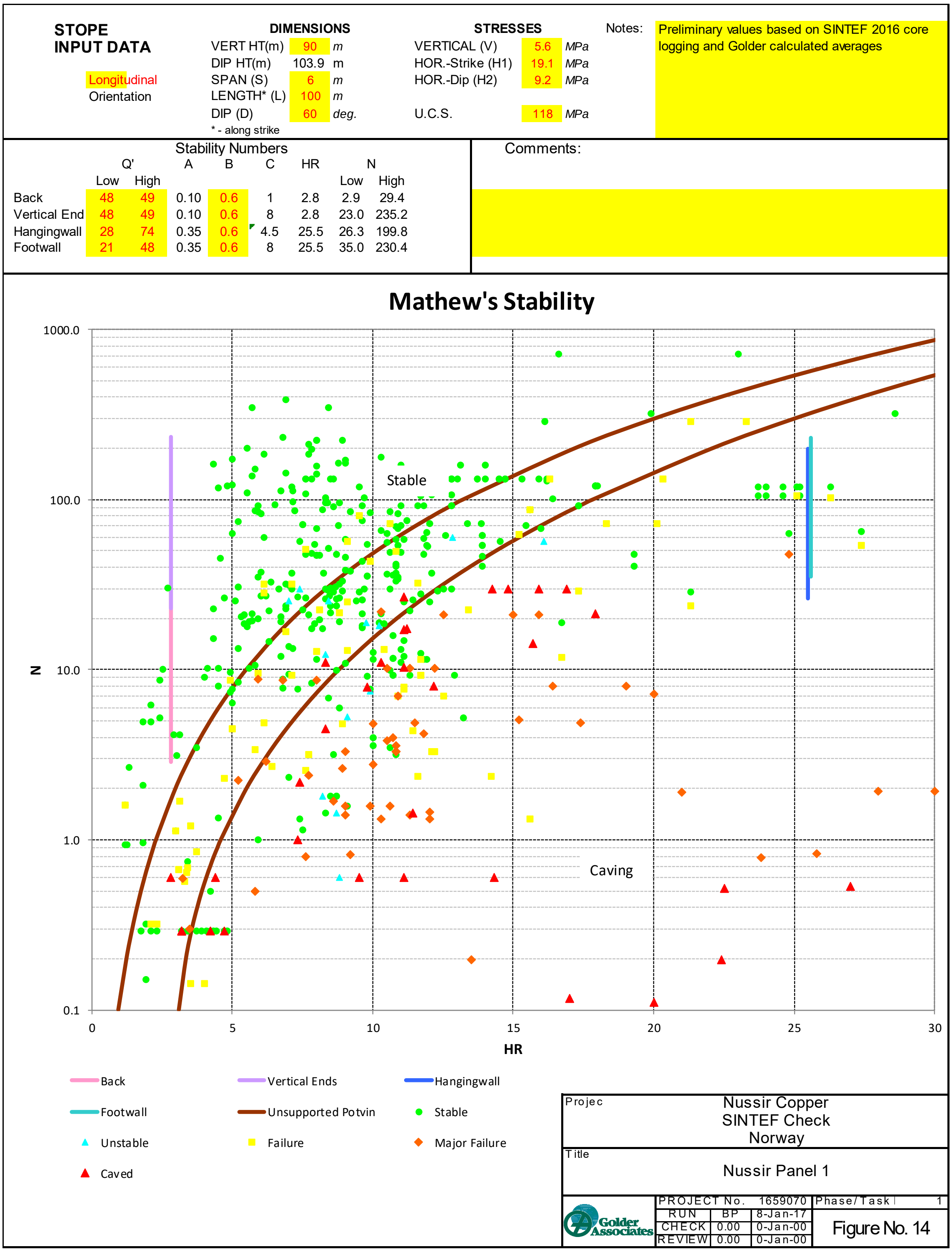

Figure 1-22: Modified Stability Chart (Unsupported) - Golder Analysis

The stope backs and end walls plot in the stable zone of the Mathews Stability Chart based on the Golder analysis. The hangingwall and footwall plot in the unstable region of the chart which could affect the current stope design and have a knock-on effect through the mine design, production scheduling and cashflow model. Golder plotted the results of the Stability Graph plots on probability iso-contours on the Extended Mathews Stability Chart (Figure 5-23) to assess the probability of stability based on the historic database and Nussir data.

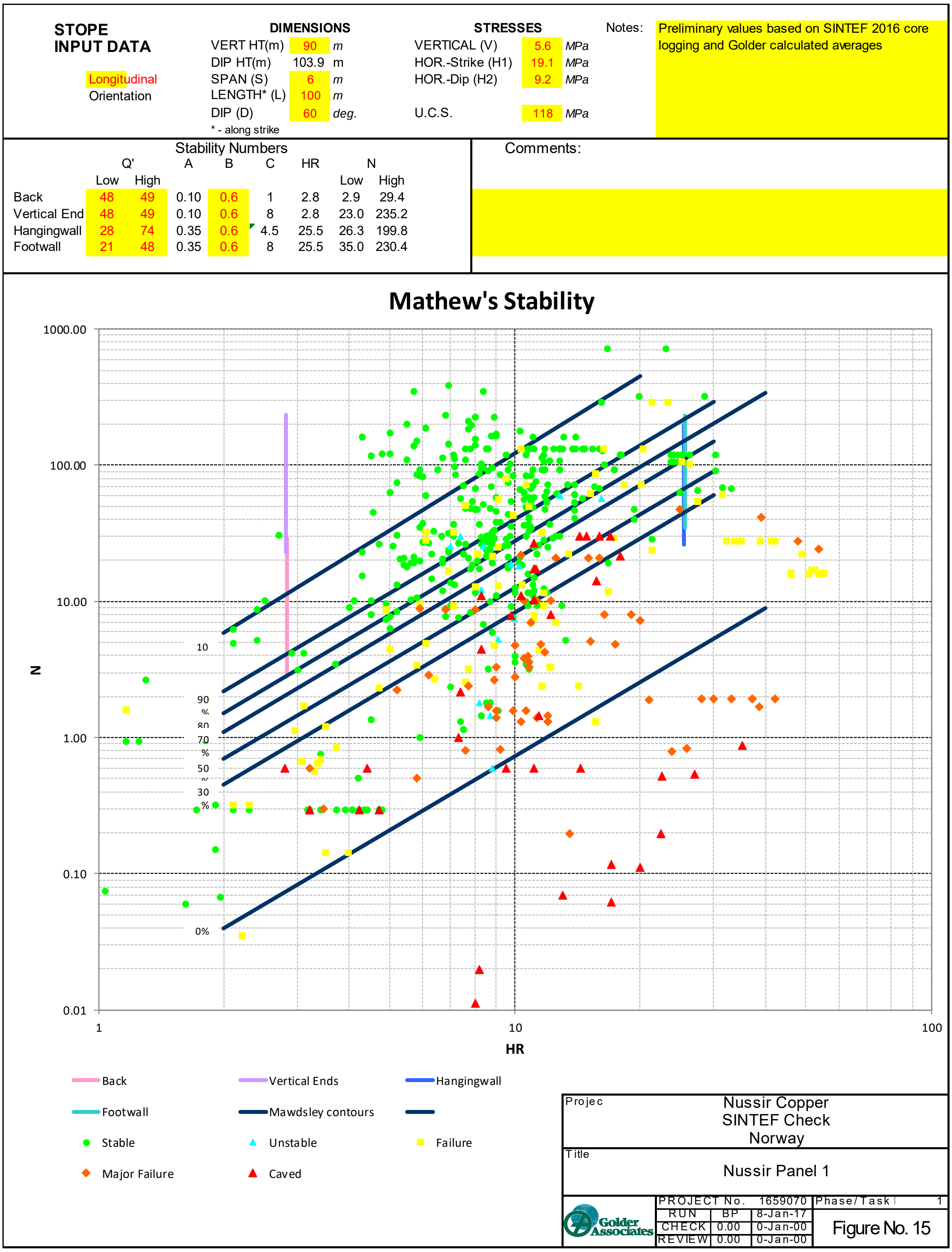

Figure 1-23: Isoprobability Stability Contours of Golder Mathews Stability Graph

The stability isoprobability contours in Figure 5-23 enable an estimate of the probability of a given stope geometry being stable based on where it plots on the Mathews Stability Graph and the logistic regression of the historic data that it plots within. The stope hangingwall and footwall plot roughly between the 30 % and 90% contours. This plot indicates a possible risk of failure of the Nussir stope geometry used in the PFS. Plotting below the 50 % isoprobability contour is interpreted as a greater than 50 % probability of failure or a less than 50 % probability of stability. The stope backs and end walls plot in the 80 % to 100 % regions of the isoprobability chart indicating that these aspects of the design would most likely be stable based on the current data.

Golder estimated a stable (>50 % stability isoprobability plot) geometry for Nussir in Figure 5-24.

STOPE INPUT DATA

Longitudinal
Orientation

DIMENSIONS			STRESSES		
VERT HT(m)	45	m	VERTICAL (V)	5.6	MPa
DIP HT(m)	52.0	m	HOR.-Strike (H1)	19.1	MPa
SPAN (S)	6	m	HOR.-Dip (H2)	9.2	MPa
LENGTH* (L)	75	m			
DIP (D)	60	deg.	U.C.S.	118	MPa

* - along strike

Notes: Preliminary values based on SINTEF 2016 core logging and Golder calculated averages

Stability Numbers

	Q' Low	Q' High	A	B	C	HR	N Low	N High
Back	48	49	0.10	0.6	1	2.8	2.9	29.4
Vertical End	48	49	0.10	0.6	8	2.6	23.0	235.2
Hangingwall	28	74	0.35	0.6	4.5	15.3	26.3	199.8
Footwall	21	48	0.35	0.6	8	15.3	35.0	230.4

Comments:

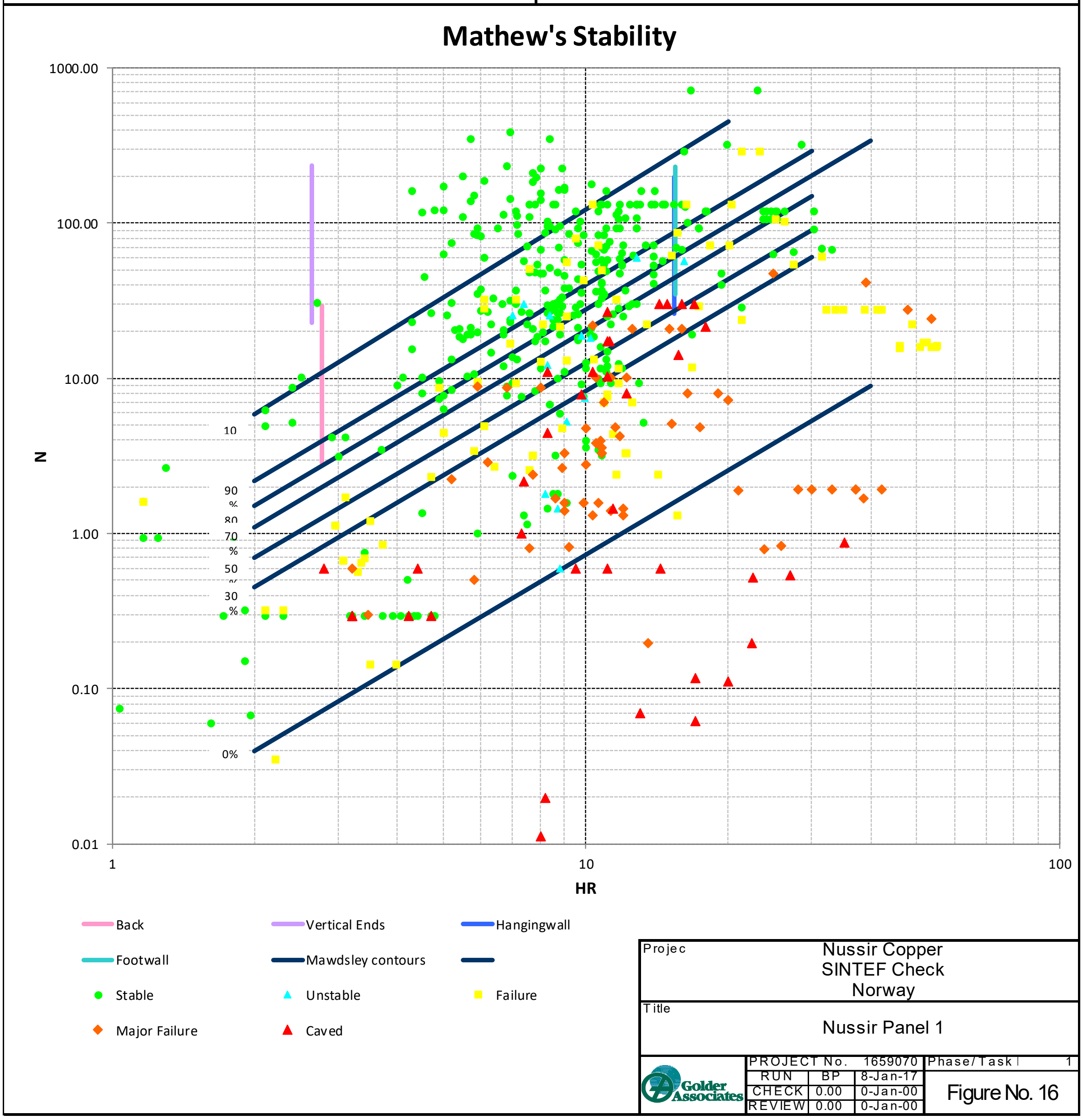

Figure 1-24: Example Stable (>50% isoprobability) Geometry Plot

A hydraulic radius (also referred to as the Shape Factor, S) of approximately 15 results in a greater than 50% stable stope shape. The stope height and length can be varied to maintain an optimal sub-level interval and strike length to sustain production. For example, a 45 m high stope, 75 m long has an equivalent hydraulic radius to a 50 m high stope that is 65 m long or 75 m high and 50 m along strike.

In addition to the above, the following conclusions were made regarding the Nussir rock mechanics and surrounding rock mass by Golder:

- The majority of the orebody, hangingwall and footwall rocks tested ranged between fair to very good, indicating that open stoping and variants of that method are feasible, at least for the Panel 1 area;
- The current stope geometry of 90 m high and 100 m long, along strike, may be unstable based on a brief analysis using the Mathews Stability Graph method for stope design and plotting the output on isoprobability contours. The stope stability is not especially sensitive to the orebody width;
- The use of sill (and rib) pillars may be a source of long term mine instability. The analyses by SINTEF indicates potential for stope crown and hangingwall instability in the lowermost stopes once the mining front advances to the next level above. Rib pillar stability was not analysed; and
- Tensile failure of stope crowns and the hangingwall is a common issue for an open stoping mining method. Installing suitable reinforcement, such as cablebolts, in the stope crown and hangingwall would improve the longer term stability of the stopes.

The following recommendations were made by SINTEF:

- The Mathews Stability Graph empirical stability method has been used to provide an initial indication of key stability issues for Stope D with 20 m sill pillars. Due to a limited structural geology data, minimum and maximum value ranges for rock jointing were analysed. The next phase of the study should consider rock mass jointing structure and induced stress in a 3D model;
- The next stage of the study should include a 3D model to account for the main stress (perpendicular to the model plane), the stress distribution in vertical pillars (in addition to sill pillars) and the global stability along the stop length; and
- Detailed geological mapping of the major joints/faults is highly recommended in order to more accurately assess the structural stability of the stopes.

Golder recommends the following:

- All future drill core is logged and recorded using the Q logging system introduced in 2016;
- Outcrops in the hangingwall, footwall and within the orebody are mapped using the same Q system to increase the geotechnical database and to provide oriented structural data to inform the Stability Graph Method;
- An optimisation of the stope dimensions and hence sub-level interval is undertaken that balances stope stability with the number of mining levels required within each of the panels;
- The mine design, stopes dimensions and pillar locations and dimensions are confirmed with further 2D and 3D modelling;
- Future 2D and 3D modelling should be completed accounting for the mining sequence and appropriate model stages to reflect extraction phases of the stopes;
- A 2D analysis of a regular array of stopes and pillars was completed based on the stope design for Panel 1. A design intended to maximise ore extraction should be analysed as well, which may have variable stope strike lengths and rib pillar widths.
- The offset distance from the footwall of the Nussir orebody to the permanent mine infrastructure (passes, extraction drives and ramps) is determined through a stress modelling exercise;

- The size and relative locations of the key underground infrastructure (workshops, canteen, pump stations, and sumps) is confirmed through modelling;
- The Stability Graphs should be updated periodically to ensure that mine and stope designs are consistent with the most recent geomechanical understanding of the deposit;
- A probabilistic approach to modelling and geomechanical design of the panels outside of the initial 5 year mining period is adapted and used to inform mine planning and design; and,

A similar programme of data collection and analyses should be completed for the Ulveryggen deposit.

1.1.5 Mining Dilution

The Nussir orebody is reasonably uniform, but varies in thickness with some irregularities occurring. These irregularities occur over a scale of 10s to 100s of metres, therefore some ore dilution is to be expected. To simplify the drilling process, stopes should be designed with straight walls to both the hangingwall and footwall, as illustrated in Figure 5-25, where the purple line represents the stope boundary while the thin red line illustrates the orebody.

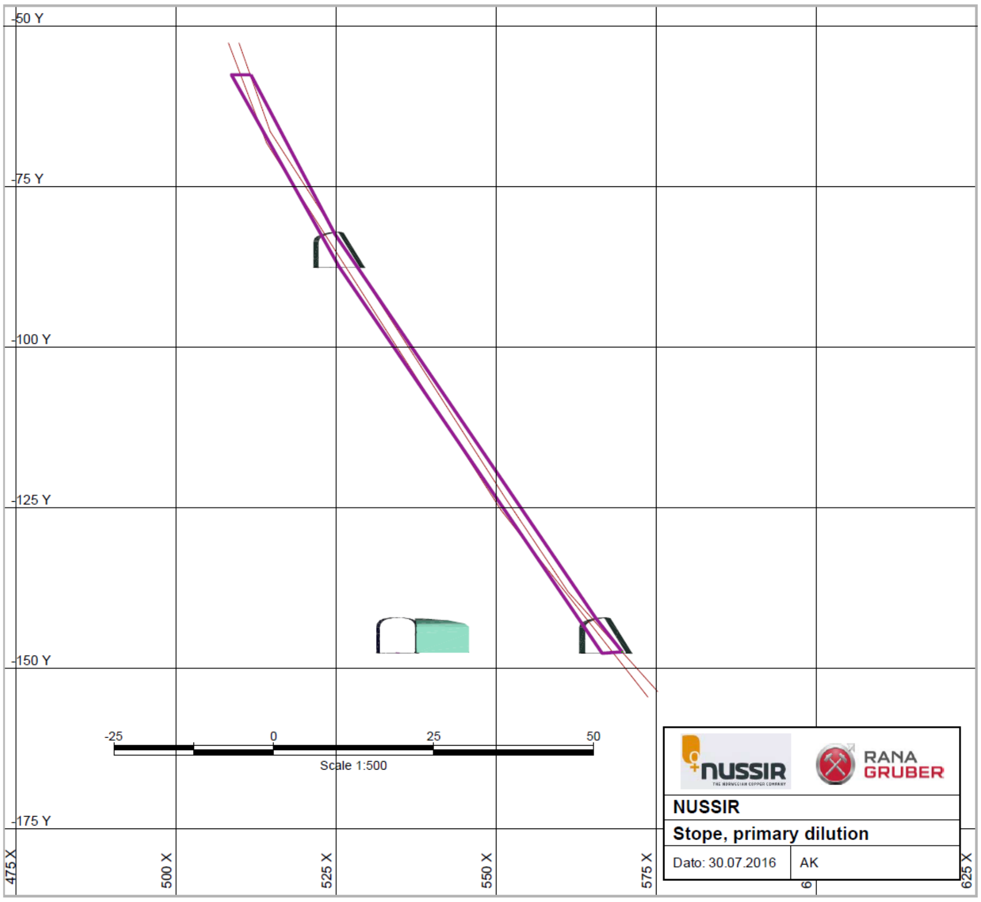

Figure 1-25: Vertical Section through a Nussir Stope.

Panel 1 stopes were designed by Rana Gruber and as such accounted for planned waste and dilution in the stope tonnages and grades as illustrated in Figure 5-25. Dilution for panels 3, 2 and 0 was calculated from the average zone thickness, a block model attribute, and a minimum mining width of 4 m. Stopes containing an average zone thickness of less than 4 m were diluted to 4 m and the grade and tonnage recalculated for the stope. Zone dilution varied from 2 % for Panel 3 to 56 % in Panel 2. Panel 0 average dilution was approximately 3,5 %.

1.1.6 Level Spacing

Spacing between the production and drill levels is dictated by the drilling equipment chosen and the quality of the drilling; holes need to be parallel to one another. With that in mind, level spacing in the Nussir mine is 105 m between 2 extraction levels. Level spacing between an extraction level and its associated drill level is 60 m.

1.1.7 Design Parameters

The design parameters for the different underground excavations at Nussir are presented in Table 5-20. Descriptions for each excavations are also provided.

Table 1-20: Design Parameter for the Development of Tunnels/Drives at Nussir.

Nussir	width / height (m)	face area (m^2)	Incline	Comment
Main decline / Access Tunnel	7 x 6,5	40	-1:10	
Haulage drive	5 x 4,75	22	1:80	inclination follows drill drive
Draw points	4 x 4	16	given	
Ramps / Spirals	5 x 4	18-19	1:8 (1:10 for spirals)	
Drill drives	7 x 4,5	20-25	1:80	asymmetric
Ventilation	3,5m diameter	7		raise drilling
Ore passes	3 x 3	9		into emptied stope
Crusher hall	13m span			varying height
Workshop	15 x 7	100		
Office/Infra	10 x 4	38		

Main Decline / Access Tunnel

The main access tunnel for the Nussir development will be 7 m x 6.5 m high. This results in a face area of approximately 45 m^2, providing enough space to install a road surface (4 m to 5 m wide), a roof supported conveyer belt as well as ventilation ducts and piping for water handling, as illustrated in Figure 5-26. The size of the drill rigs and LHDs are indicated by the coloured squares. The tunnel will be driven with a maximum decline of -1:10. Every 250 m along the tunnel, passing bays will be mined to allow two vehicles to pass one another.

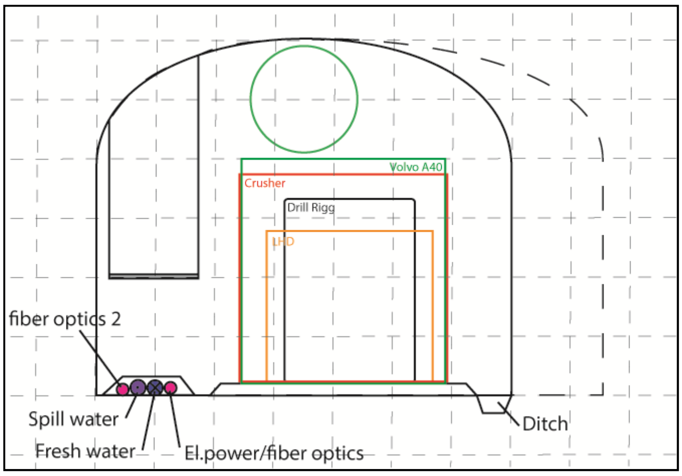

Figure 1-26: Sketch of Main Decline Tunnel Profile

At Ulveryggen the existing transport tunnel which ends below the abandoned open pits will be re-furbished and used for mine access as well as transport of broken ore to the ore pass that feeds into the process plant silo by a conveyor. Before development starts, this tunnel must be inspected and re-surveyed. The result of this survey will help determine whether it is necessary to widen the face area along the whole length of the tunnel or at specific distances to create meeting points where trucks can pass each other.

Haulage Drives

From the access tunnel, 4 m x 4,5 m wide haulage tunnels will run parallel to the drill drives. These haulage drives are planned to have a 1:80 inline. Haulage drives will be constructed every 20 m at stope access / loading drives.

Draw Points

Draw points will be developed from the haulage drives towards the drill drive and are planned to be 4 m x 4 m in size.

Ramps / Spirals

Ramps from the main level on L-150 are planned with a maximum incline of 1:8. This will allow for all mining equipment (LHD's and drill rigs as well as personnel carriages and pick-ups) to move easily between the different levels. Spirals are planned with a 1:10 incline and radius of approximately 20 m.

Drill Drives

Drill drives will be constructed within the mineralized zone. To reduce dilution during development drives will be built asymmetrically along the contact to the hangingwall (Figure 5-27). Drill drive size depends on the width of the ore-bearing zone but should not fall below 7 m x 4,5 m. Some dilution is expected along the contact with the footwall. In general, these drives are planned with an inclination of 1:80.

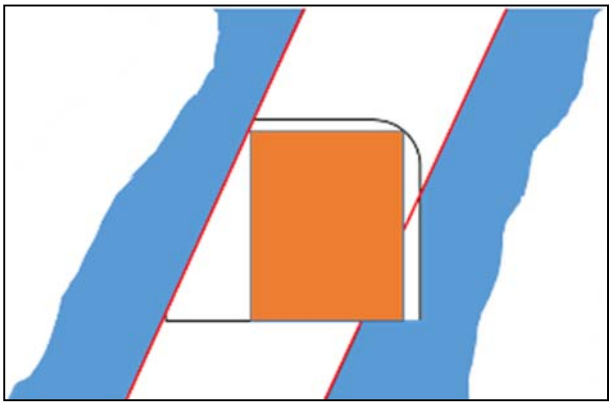

Figure 1-27: Cross-Section Sketch of an Asymmetric Drill Drive

In the equipment envelope for a Simba ME7C drill rig is illustrated in orange, the stope boundary in red, and the hangingwall and footwall are shaded in blue.

At the Ulveryggen deposit, drill drives will be built with the same 1:80 incline but in contrast to the Nussir drives with a more symmetric shape (4 m x 4,5 m).

Ventilation

The main ventilation shaft is planned to be a raise shaft with a diameter of 3,5 m.

Ore Passes

Ore passes will be 3 m x 3 m slots and will deliver ore into an empty stope on the level below.

Crusher Hall

The crusher hall will be situated at the main level (L-150) and close to all transportation drives in the mine. The platform the crusher sits on will be lowered by 2 m to 3 m (depending on model used) compared to the parallel-orientated transportation drive, as illustrated in Figure 5-28. This should allow easy tipping into the feeder by both LHDs and mining trucks.

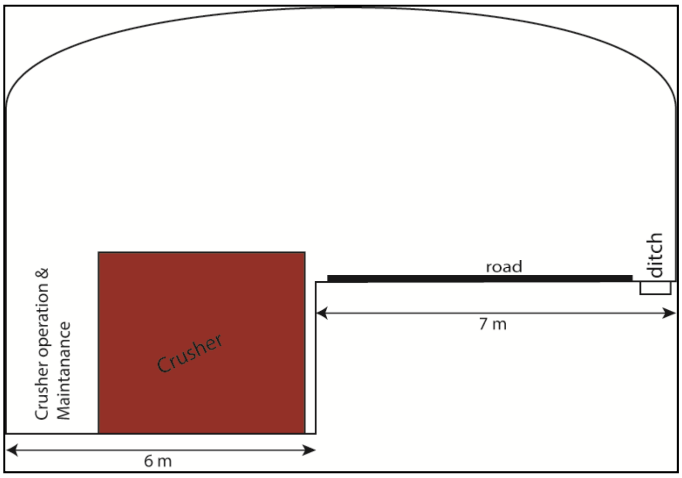

Figure 1-28: Crusher Hall at L-150 in the Nussir mine.

Work Shop

The work shop will be 15 m wide by 7 m high. The entrance to the work shop will be 5 m x 5 m.

Canteen

The underground workings for the canteen and office facilities will have a span of 10 m, and be 4 m high.

1.1.7.1 Mine Layout

The Nussir deposit is planned to be mined by sub-level open stoping in 4 panels, as shown in Figure 5-29 and summarised in Table 5-21. The mine will be developed from the east to the west starting with Panel 1 in the east. The main entrance into the mine will be situated at the industrial area close to the fjord, which also houses the processing plant, office buildings, and barracks. An approximately 2,6 km long main decline with a maximum gradient of -1:10 will lead to the main level (L-150) ending in the centre of mining Panel 1 (Figure 5-30). A sketch illustrating the layout of Panel 1 is presented in Figure 5-31. At the main level where the decline enters there will be a crusher hall, canteen with offices, workshop and a water-settling basin (Figure 5-32). Additional openings will be made to house transformators and ventilation. The ramp to the mining levels is situated in the central area of the panel. The central area of each level will also be the lowest point where spillwater will be collected. From there, haulage and drill drives will spread outwards to the east and west and a ramp system will lead to production levels at higher elevations. On each level, 6 to 9 stopes will be developed on either side of the entrance. In total 21 stopes to the east and 33 stopes to west will be developed, this difference is due to changes in surface elevation. For each stope, one level containing both a drill and a haulage drive (extraction level) and one level for drilling will be developed (Figure 5-33 and Figure 5-34). Drill and haulage drives will be connected by a series of drawpoints. The sequence of extraction level and drill level will be repeated for all levels.

A fourth panel will be planned west of the intepreted fault zone once mineral resources in that area have been better defined.

Ventilation will be provided through a 3,5 m wide shaft from the surface; exhaust air will rise up through the ramp system and be released through a raise (the ventilation system is described in more detail in Section 1.3).

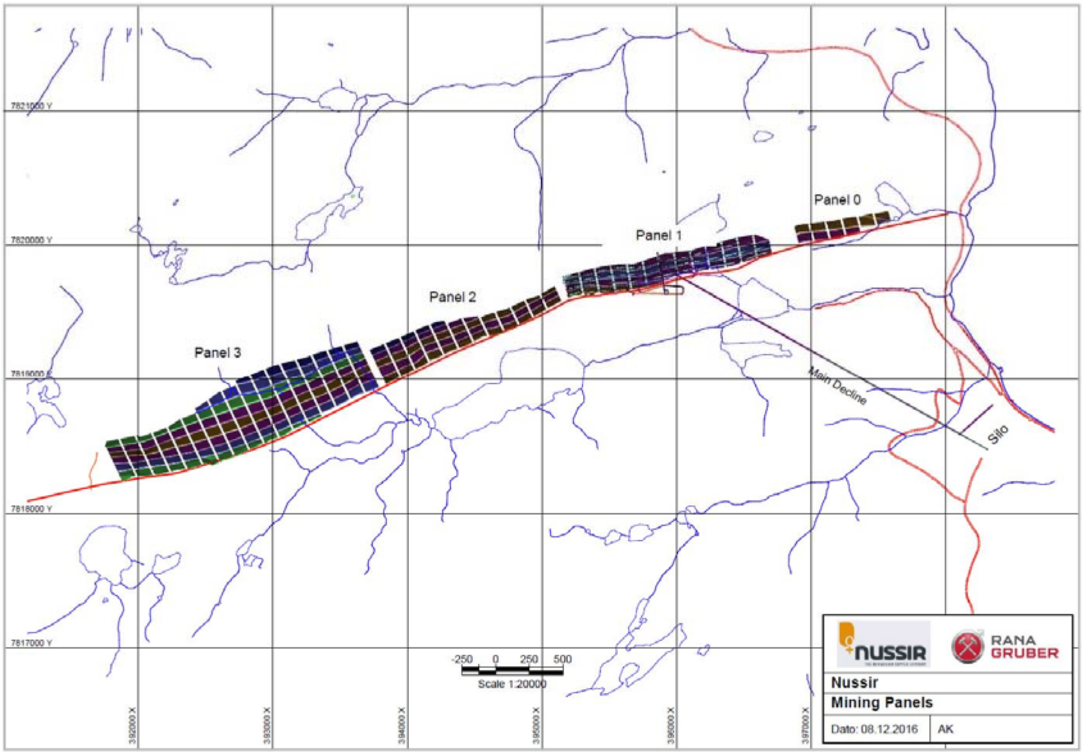

Figure 1-29: Map Illutrating the Location of the Nussir Mining panels and their Stopes (Panel 0, 1,2 and 3).

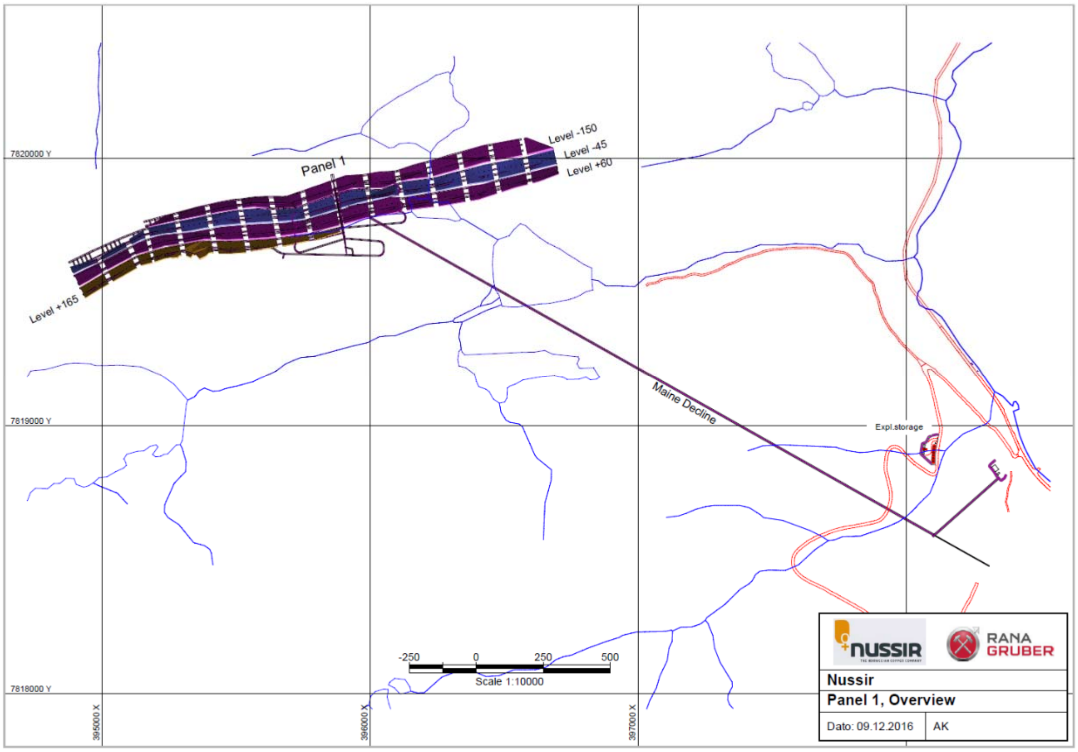

Figure 1-30: Overview over Panel 1 Including the Main Decline, Nussir Mine. Roads (red) and Water Courses (blue) are Shown for Reference.

Table 1-21: Tonnages Nussir Mining Panels 0, 1, 2, 3.

Mining Panel	Level	Scheduled Resources			
		Tonnage	%Cu	Ag	Au
Panel 0	L-60	649 431	1,040	8,740	0,077
Panel 0	L45	492 758	1,257	11,885	0,127
Sub Total		**1 142 188**	**1,138**	**10,167**	**0,100**
Panel 1	L-150	725 638	1,178	16,463	0,165
Panel 1	L-45	1 193 398	1,007	13,593	0,140
Panel 1	L+60	973 793	0,988	12,134	0,130
Panel 1	L+165	436 023	1,144	8,503	0,310
Sub Total		**3 328 852**	**1,057**	**13,125**	**0,165**
Panel 2	L-225	521 440	0,806	15,482	0,139
Panel 2	L-120	728 270	0,702	13,439	0,119
Panel 2	L-15	780 215	0,653	11,097	0,120
Panel 2	L+90	805 697	0,704	9,424	0,123
Panel 2	L+195	754 068	0,571	6,965	0,107
Sub Total		**3 589 690**	**0,679**	**10,966**	**0,120**
Panel 3	L-435	758 680	1,019	6,050	0,132
Panel 3	L-330	1 213 421	1,105	8,293	0,159
Panel 3	L-225	1 875 033	1,136	8,459	0,128
Panel 3	L-120	1 917 905	1,144	9,496	0,108
Panel 3	L-15	1 579 846	1,090	10,023	0,099
Panel 3	L+90	1 224 246	1,112	8,926	0,102
Panel 3	L+195	1 167 609	1,188	9,596	0,099
Panel 3	L+300	565 354	1,035	8,231	0,054
Sub Total		**10 302 094**	**1,099**	**8,727**	**0,112**
Total		**18 383 058**	**1,010**	**10,038**	**0,122**

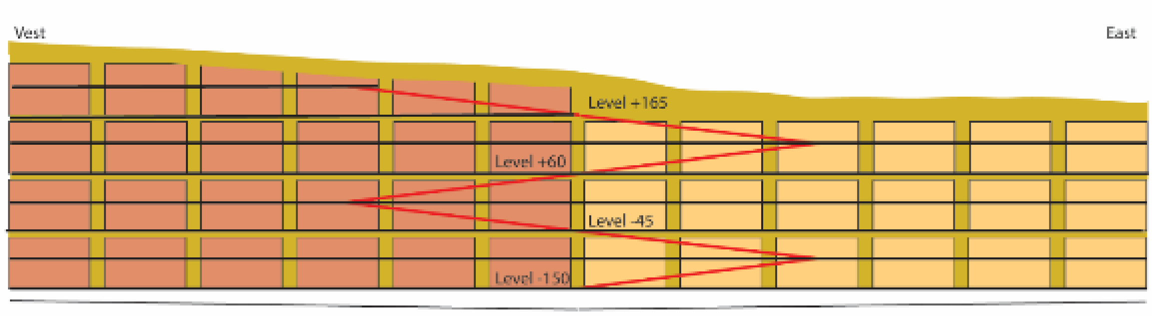

Figure 1-31: Sketch of Panel 1as a Projection Seen from the Footwall (Not to Scale).

Figure 5-33 illustrates development and mining activities in two different stopes on two levels. While mining started on the lowermost level, development of the extraction and the drill level will be in progress on the level above.

Figure 1-32: Mine Layout, Main Level L-150 Nussir Mine.

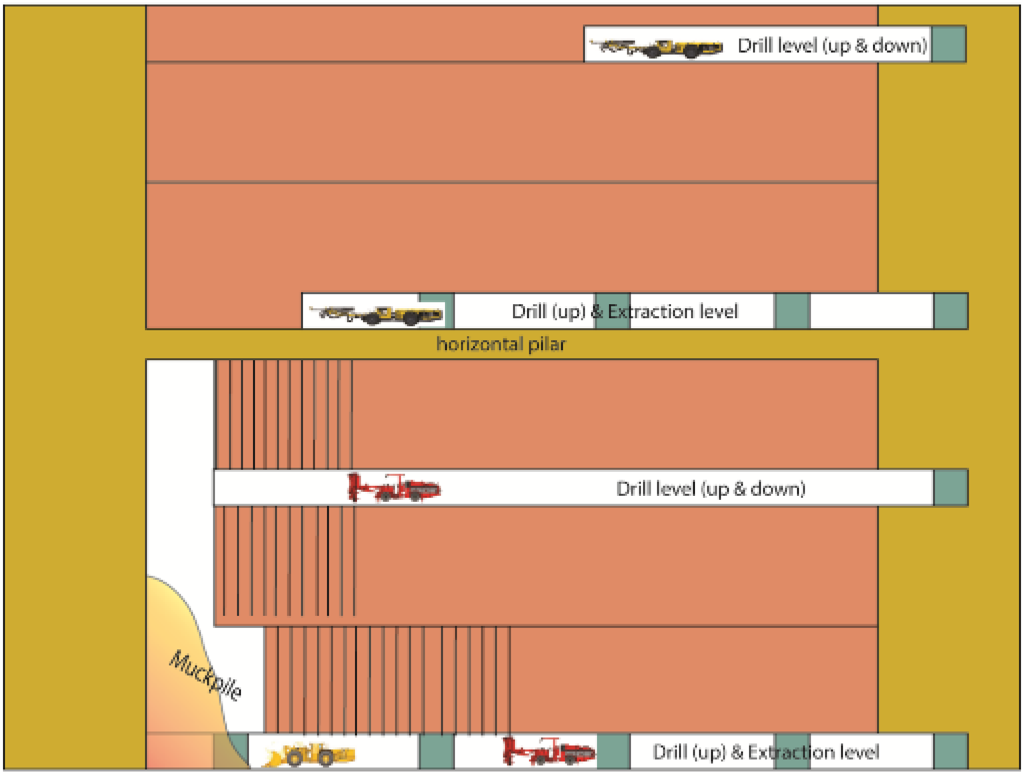

Figure 1-33: Illustrating Mining with Simultaneous Activities in Two Stopes on Two Levels.

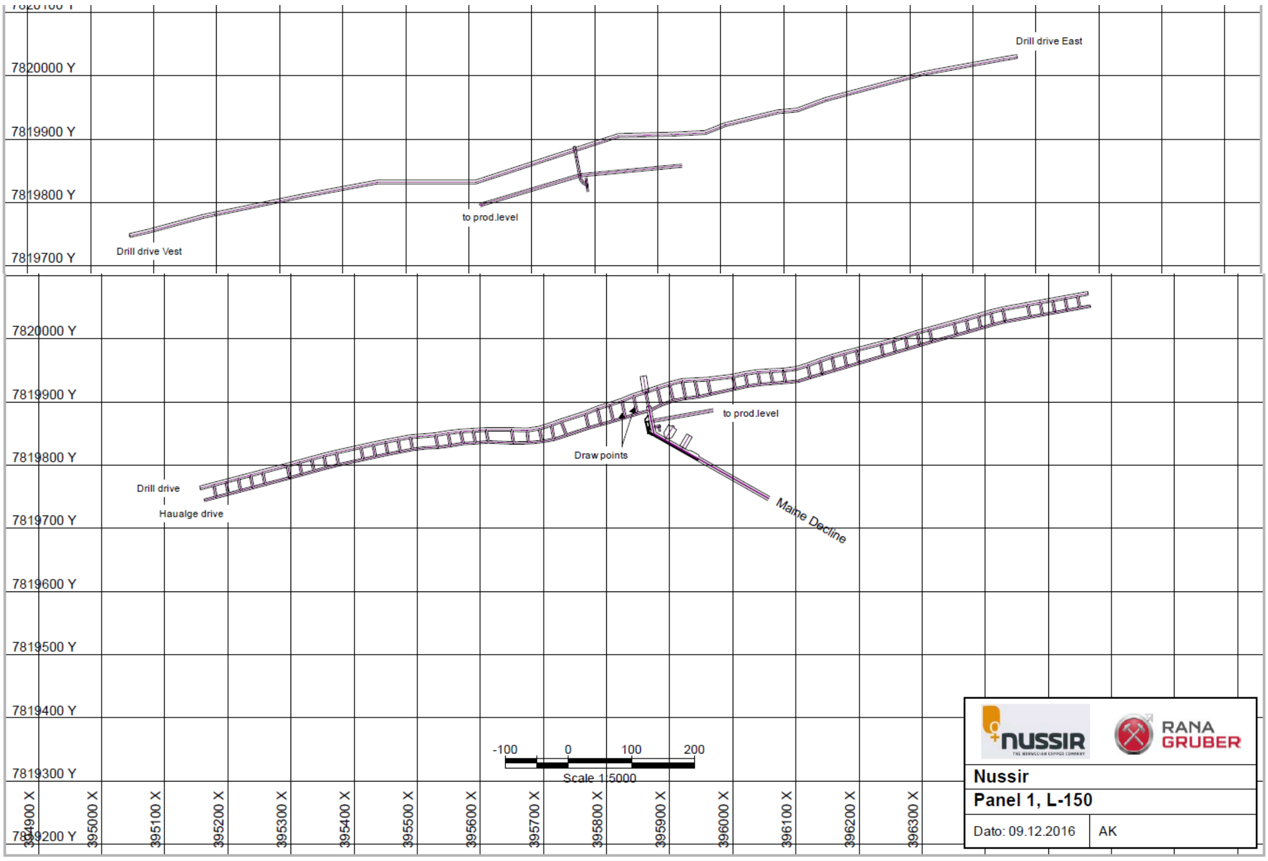

Figure 1-34: Layout of Level -150 (Drill and Extraction) and Level -90 (Drill), Nussir Panel 1.

1.1.7.2 Mining schedule

The mining schedule is based on production starting as soon as the first drill level and its associated stope are developed.

Throughout the life of mine there will be at least 2 stopes in simultaneous production. It would be advantageous to mine one stope close to the crusher and one further away to minimise the transportation distances and costs. It is planned to start production on level –150 with stopes 6 and 12, from there production will proceed towards the west. Panel 1 will be active trough 4 years of production. During that time, development should commence and successively open panels 0, 2 and 3 for production.

The expected Cu grade of the ore to be extracted from the stopes in Panel 1 ranges from 0,804 % to 1,551 % Cu (Figure 5-36).

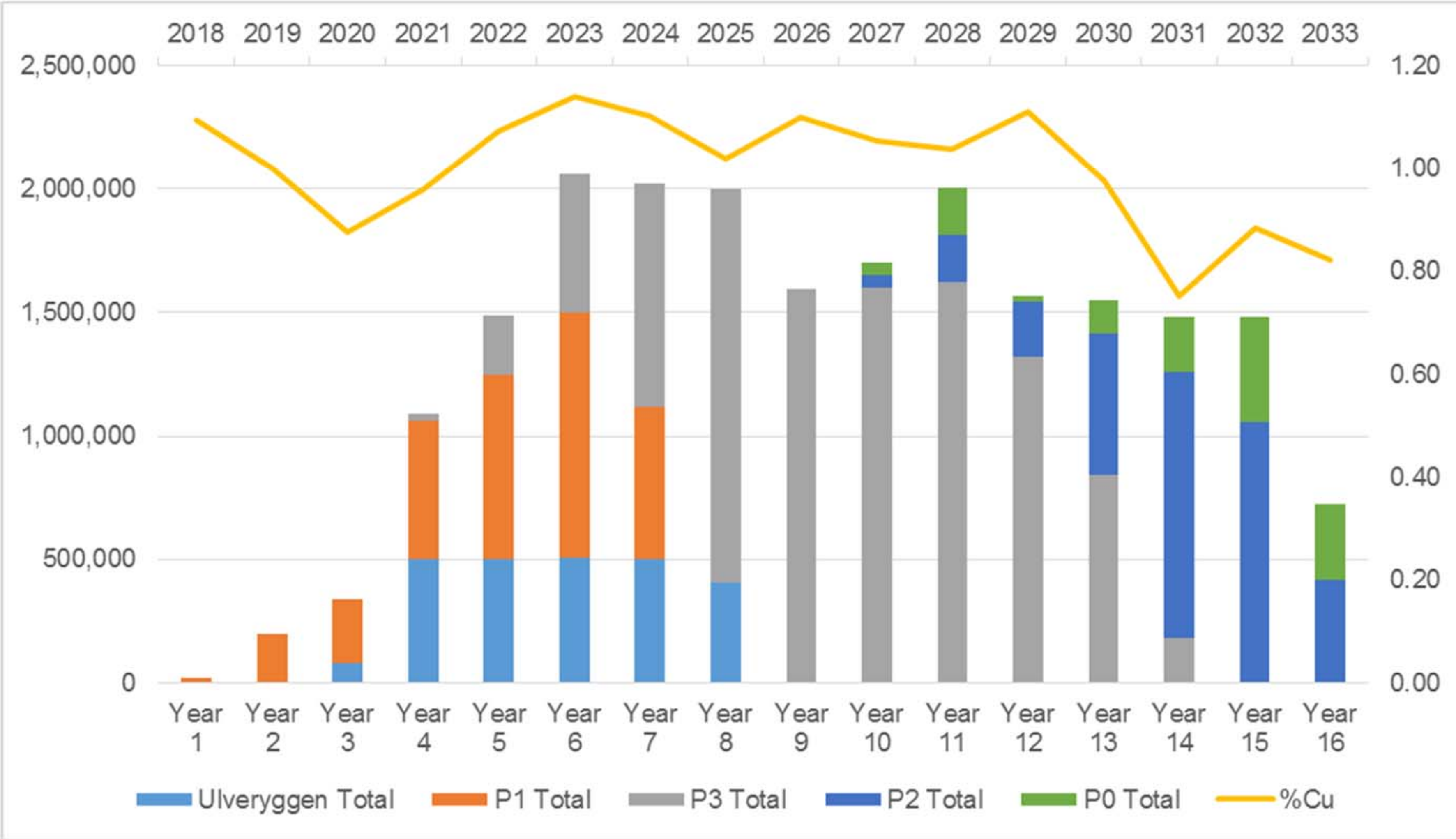

Figure 1-35: Annual Life of Mine Production Forecast by Mining Area

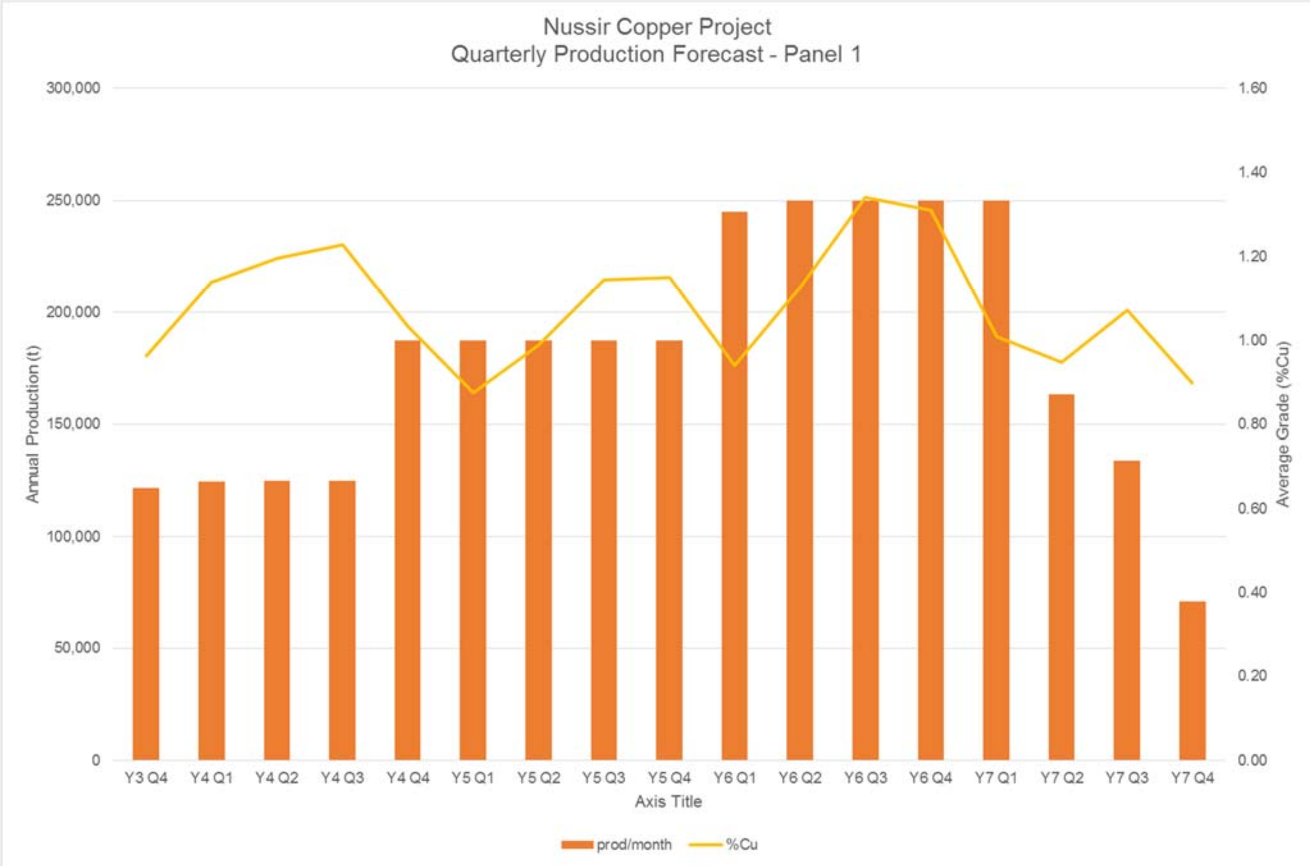

Figure 1-36: Expected Quarterly Copper Grade, Nussir Panel 1.

1.1.8 Ulveryggen Mine Design

The Ulveryggen mineralization differs in both size, shape and grade in comparison to the Nussir orebody. While Nussir is a several kilometres long, continuous and rather narrow, the Ulveryggen deposit consists of several distinct and shorter along strike but wider orebodies. A common feature in these orebodies is that they become smaller with increasing depth. Ulveryggen has been previously mined from small open pits. These pits are today abandoned and one of the pits is partly filled with water while a part of another is used by Norsk Gjenvinning as a waste drilling material disposal area. Figure 5-37 illustrates the open pits at Ulveryggen as well as the projected boundary of the orebodies at the current surface. The red line illustrates the projected outline of the orebodies at the surface. The bottom of the pits is not well defined as they are filled with water (water level at 360 m) or used and re-filled by Norsk Gjenvinning. It was not possible to find maps illustrating the last stages of mining that could help to constrain the final depth of the open pits.

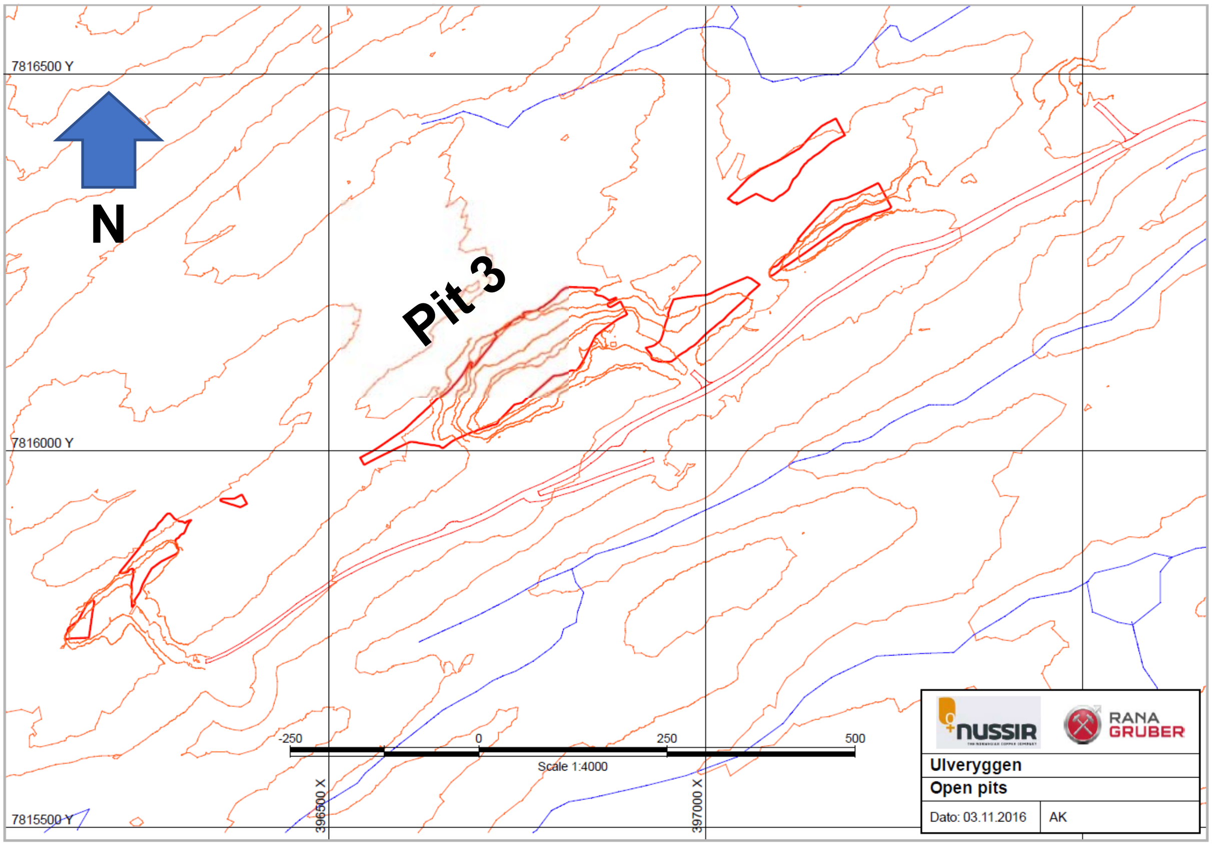

Figure 1-37: Existing Open Pits at Ulveryggen (Numbered 1 to 4 from East to West).

Underground, at approximately at 225 mASL, a ca. 2.4 km long tunnel, two ore shafts and one ventilation shaft exist, which was used for ore transport to the processing plant during open pit operations in the 1970's. These underground workings will be re-used for any new mining activities at Ulveryggen.

1.1.8.1 Ulveryggen Underground Mining

Underground mining at Ulveryggen is planned to be by sub-level open stoping involving 4 stopes. Stope dimensions will be approximately 90 m high x 100 m long, however, the length of the stopes will change to accommodate the variation in width of the orebody. The stopes will be developed with an extraction / drill level and a secondary drill level 60 m above the extraction level. Draw points will be developed from a haulage drive on the extraction level into the drill drive situated directly below the stope.

The planned drift sizes for the Ulveryggen Mine are summarised in Table 5-22.

Table 1-22: Drift sizes, Ulveryggen Mine

Ulveryggen	Width / Height (m)	Face Area (m^2)	Incline	Comment
Main Decline/ Access Tunnel				Existing access tunnel
Haulage drive	5m x 4,75m	22	1:80	inclination follows drill drive
Draw points	4m x 4m	16	given	
Ramps / Spirals	5m x 4m	18-19	1:8 (1:10 for spirals)	
Drill drives	4m x 4,5m	33	1:80	Symmetrical

A general layout of proposed Ulveryggen stopes and development is presented in Figure 5-38.

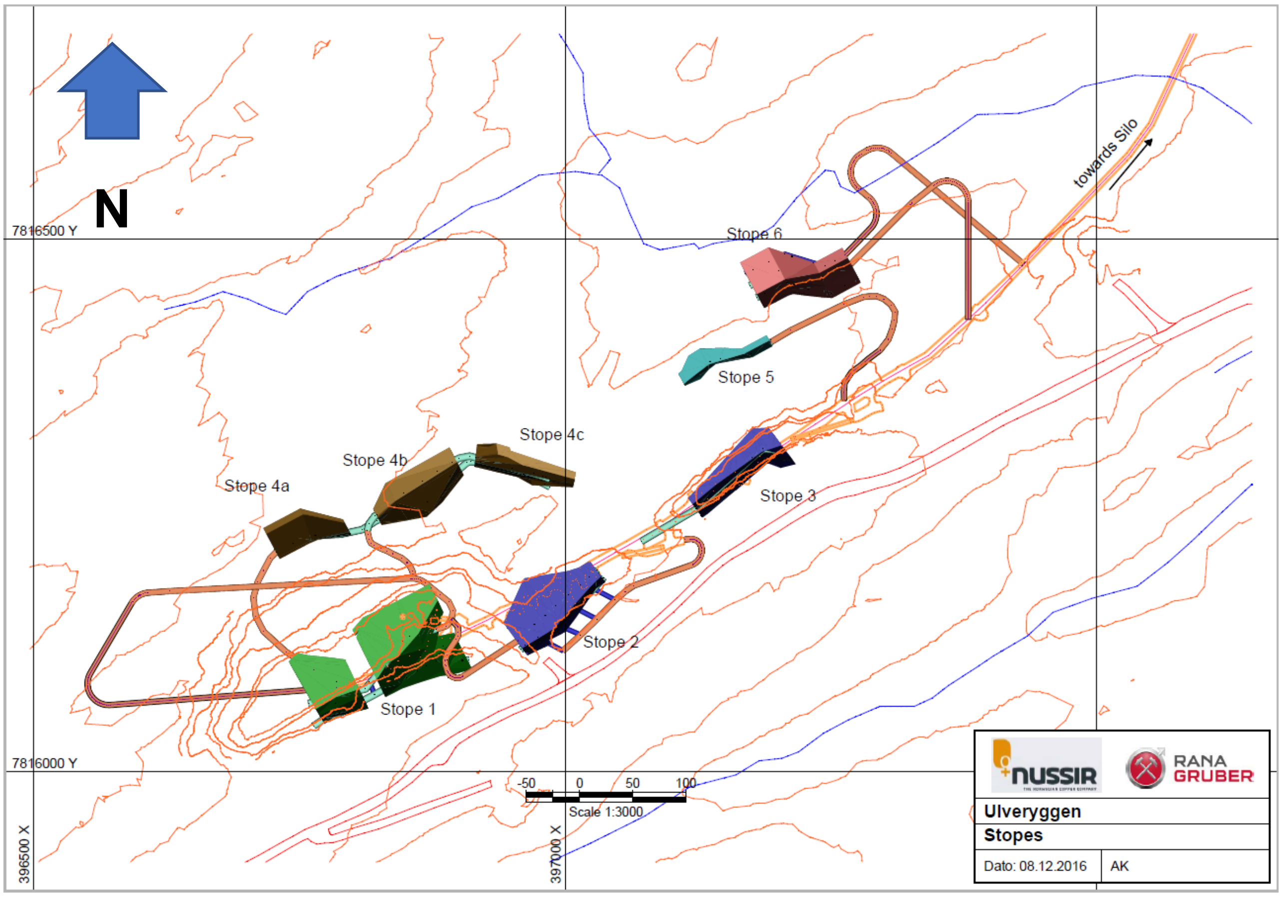

Figure 1-38: Ulveryggen Stopes (Numbered 1 to 6 from West to East).

Illustrations of Ulveryggen indicate that there are 9 stopes in total, however once economics and other criteria are applied to the stope shapes, only 4 of the stopes are included in the mining schedule. The tonnages for each of the Ulveryggen stopes are summarized in Table 5-23.

Table 1-23: Ulveryggen Stope Tonnages

Stope	Resource Category	Tonnage	% Cu
Stope 1a	Indicated	297 672	0,317
Stope 2	Inferred	1 742 733	1,147
Stope 4a	Inferred	85 825	0,636
Stope 4b	Inferred	345 387	0,530
Development	Inferred	27 240	1,017
Total		**2 498 857**	**0,944**

The life of project mine production schedule for Ulveryggen is presented in Figure 5-39. The planned total tonnage to be mined from the Ulveryggen stopes between years 3 and 8 of the life of mine schedule (for Nussir and Ulveryggen combined) is 2,5 Mt with an average grade of 0,94 % Cu.

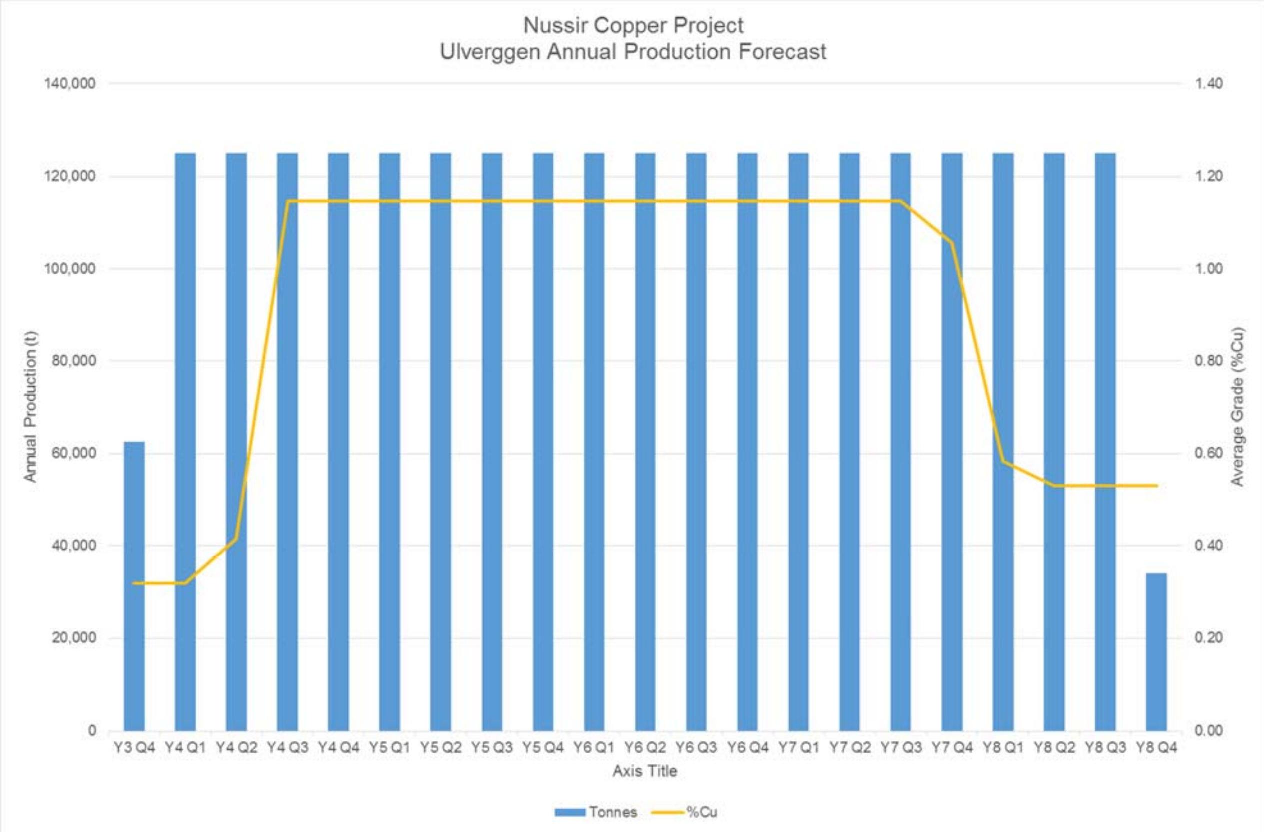

Figure 1-39: Ulveryggen Life of Project Mine Production Schedule

1.2 Mining Equipment

Equipment selection can have a substantial effect on operating costs and production rates of an operation. The chosen types of equipment are presented below and summarised in Table 5-24 .

Development drilling will be carried out using a fleet of 4 jumbo drills, 2 will be located at Nussir and 2 at Ulveryggen. To accommodate the large and small drives, each mine will require two sizes of jumbo, for example, an [redacted] and an [redacted]. It is assumed that these will have utilisation of around 50 %.

Development loading and hauling will include a fleet of 4 LHDs, 2 will be located at Nussir and 2 at Ulveryggen. To accommodate the large and small drives, each mine will require two sizes of LHD, for example a [redacted] and a [redacted].

Production drilling will be provided by a fleet of 4 drill rigs which can drill parallel holes up to 30 m long from one position, such as the [redacted]. Three of these drill rigs will be located at Nussir, and 1 at Ulveryggen.

These drill rigs are capable of drilling parallel up and down holes with a separation distance of up to 6.4 m. Holes can be drilled with diameter up to 89 mm. The drilling unit is situated on a boom for maximum range and flexibility. The rigs are capable of automatic full-ring drilling and can be controlled remotely from a unit situated close to the rig or even from a distant office facility.

A raise drilling rig, such as the [redacted] may be an option for drilling the opening slots. The [redacted] can drill up to 60 m long holes with a diameter up to 75 cm, without the need of a pilot hole.

Production loading and hauling will be done using a fleet of 4 LHD machines; 2 at Nussir and 2 at Ulveryggen.

Two types of LHD machine are available, (a) the conventional diesel powered machines and (b) the more environmentally friendly battery powered LHD machines.

1) Diesel Powered

This alternative would include 4 [redacted], with a 14 t capacity for hauling material from draw points to the crusher, or alternatively to an ore pass. The [redacted] may be operated remotely. An alternative LHD machine from another supplier would be the [redacted] from [redacted].

2) Battery Powered

Suppliers of mining machines are developing battery powered LHD machines. Both [redacted] and [redacted] now have "green" alternatives in their portfolio. In the case of [redacted], their [redacted] Battery is currently only available in North America, but the company is working on making these machines CE (Conformité Européene) certified, [redacted] plan to release the [redacted] Battery model to the European market in late 2017/early 2018. Similarly to the [redacted], the battery powered [redacted] is remote capable. One disadvantage of the battery powered LHD's is their low tram capacity. [redacted] reports a tram capacity of 6,8 t per scoop for their battery powered machines, whilst the [redacted] units can only manage 5 t per scoop. [redacted] are currently developing a 10 t battery powered LHD.

Mining truck sizes are restricted by small drift sizes (4 m x 4,5 m). A small unit such as the 20 t [redacted] is recommended. There will be a fleet of 3 mining trucks which will transport ore from the outermost stopes in each panel to the crusher or ore pass. One of these will be operating at Nussir and 2 at Ulveryggen. An alternative to the [redacted] would be a 40 t truck, such as the [redacted], or articulated trucks from e.g. [redacted], such as the [redacted]. To transport ore from an ore pass / magazine to the crusher a häggloader will be used, along with a series of heavy-duty conveyor belts running directly into the crusher.

The explosives charging rig will carry the loading unit which will be rented from the explosives supplier, which will be either [redacted], [redacted] ([redacted] have temporarily discontinued the production of these rigs). Any four-wheel drive lorry that can carry the tank and loading unit will suffice; [redacted], [redacted] and [redacted] supply such equipment.

A **cable-bolting rig**, potentially the [redacted] from [redacted] will be required on site. This is to ensure an effective way of installing necessary rock support.

Other mobile fleet equipment includes a shotcrete wagon ([redacted] or similar) a grader for road maintenance, a small wheel loader ([redacted]), a small excavator on wheels (with hydraulic hammer), maintenance vehicles, a couple of pickups, and two busses for personnel transport will also be required. Table 5-24 shows the complete list of mobile fleet equipment required. Raise drilling / boring and development will be complete by contractors.

Table 1-24: Mobile Equipment Fleet

Item	No.	Model / Make	Comment
Development Jumbo	2	[redacted]	Contractor
Development Jumbo	2	[redacted]	Contractor
Production Drill rig	4	[redacted]	Nussir ASA

Item	No.	Model / Make	Comment
Development LHD	2	[redacted]	Contractor
Development LHD	2	[redacted]	Contractor
Cable Bolter	2	[redacted]	Contractor
Shotcrete Wagon	2	[redacted]	Contractor
Raise Drill	1	[redacted]	Contractor
Production LHD	3	[redacted]	Nussir ASA
Mining truck	3	[redacted]	Nussir ASA
Hæggloader	1	[redacted]	Nussir ASA
Charging rig/truck	2	[redacted]	Nussir ASA
Grader	1	[redacted]	Nussir ASA
Wheel loader	2	[redacted]	Nussir ASA
Small Excavator	1	[redacted]	Nussir ASA
Medium sized excavator (wheels)	1	[redacted]	Nussir ASA
Pick-ups	10	[redacted]	Nussir ASA
Bus (9 seats)	2	[redacted]	Nussir ASA

1.3 Mining Operations

1.3.1 Grade Control

Grade or ore control has to start during development through a program of mapping and systematically sampling faces during tunnelling advance in drill drives and other headings within the orebody. This is crucial to ensure the optimal position of ore drives. During mapping and sampling, it is important to log sample quality (size / consistency / comparability).

Daily samples should be collected at all active draw points during production and assayed for Cu. These analyses should be used for reconciliation and stock pile management.

1.3.2 Mine Survey

During development, all drives and excavations underground must be surveyed on a regular basis. Ideally this should be every second day, especially in drill drives which need to follow the strike of the ore. Surveying should be carried out using a total station and a set of established control stations. Prior to the start of development, a set of control stations (2 to 3 stations) should be installed just outside the main entrance to the portals, preferably mounted on a pole (Figure 5-41 c)). These can be installed as bolts in the wall or surveying poles installed on flat but solid ground (Figure 5-41 a) and b). Ideally, these stations are placed in a secured/protected area, safe for any impact of traffic and snow handling during the winter season.

a)	c)

Figure 1-40: a) Permanent control station in tunnel wall. b) Short term control station using a bolt in the wall. c) Control station mounted on a pole outside the mine entrance.

During development, control stations should be established continually and located at least every 200 m on both sides of the drives. They must be clearly marked and ideally, they should also be located to avoid damage during ongoing development and later mining operations. A set of survey control stations needs to be established on each level. These need a higher degree of protection than ordinary control stations. All control stations need identifiable ID's and should be stored in a database together with their status (OK, re-surveyed, removed, destroyed). Control stations that are destroyed should be removed.

Centrelines of the roof and floor, and wall and arch lines should be surveyed regularly. Larger excavations which house important mining installations, such as the crusher hall, should be surveyed in more detail either by laser scanning or photogrammetric methods. Total stations and laser scanners are supplied by several companies; both Leica Geosystems and Trimble are well represented with dealerships and support in Norway.

In drill drives and during development within the ore, the face of the drive should be mapped by a mining geologist and main features such as contacts to hanging and footwall or major fault or fracture zones should be surveyed to continuously refine the geological model of the orebody.

1.3.3 Ground Control / Rock Mechanics

To ensure a safe working environment in the mine, ground control must be an integral part both during the planning, development and operational phases of the mine. It is suggested that recommendations by
Ref. 5-11: (NGI 2015) are followed, using the Q-system to evaluate ground support requirements, as shown in Figure 5-42. Different support categories are indicated by numbers ranging from 1 (unsupported) to 9 (special evaluation). The blue rectangle indicates the support category for access and haulage drives, the orange rectangle highlights support required for larger openings (20 m to 30 m span). The red rectangle indicates the range of rock mass quality (Q) encountered.

In addition to the Q-Value, the NGI recommendations include two other factors to evaluate support design in underground excavations. These factors are safety requirements and dimensions (span or height of the opening). Generally speaking, there will be need for increased ground support with increasing span or height of the excavation. NGI uses a factor called Excavation Support Ratio (ESR) to express safety requirements. Low ESR values indicate the need for high level of safety while higher

ESR values indicate that lower levels of safety will be acceptable. Together with the value for span, ESR gives the Equivalent dimension in the following way:

$$Equivalent\ Dimension = \frac{\text{Span or Heigth in m}}{\text{ESR}}$$

Typical ESR values used in mining projects range from ESR=1 for permanent excavations such as workshops, offices and storage rooms, ESR=1.3 for access tunnels, ESR=1.6 for tunnels, drifts and headings for large openings, to ESR=3 for temporary mine openings.

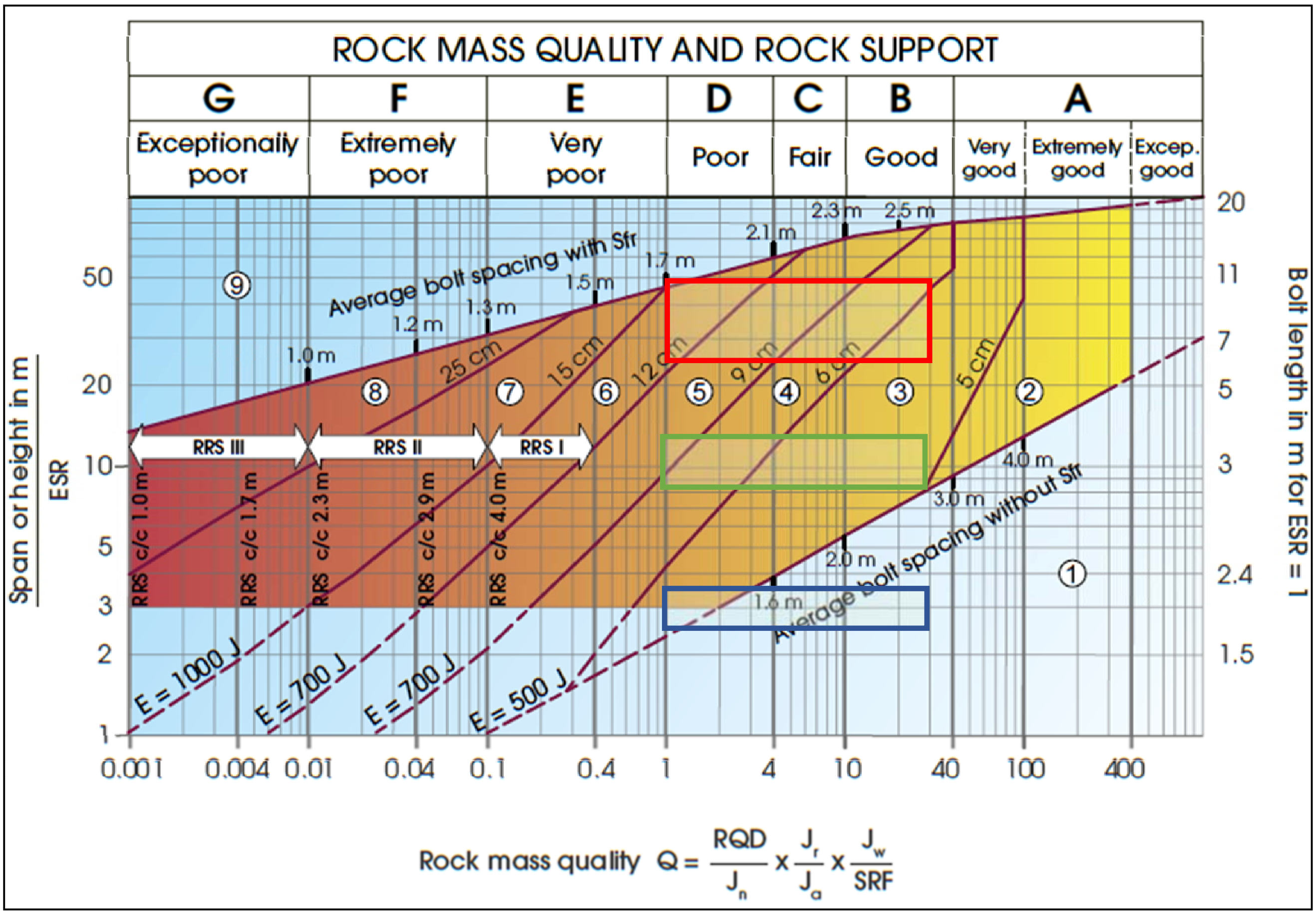

Figure 1-41: Support recommendations based on Q-values and span/ESR.

1.3.3.1 Ground Control during Development

During development all drives must be assessed by a rock mechanics engineer or geologist to map fractures and faults and to assess as well as document the rock mass quality according to the Q-value system

Ref. 5-11: (NGI 2015). Necessary rock support exceeding standard support must be adjusted according to mapping/observation results. Most of the rock types in the investigated area show compressive strength in the range between 90 and 120 MPa. Indications from Q-values show good rock quality, generally in the range between fair to very good and excellent. However, RQD and Q values are currently only available in close vicinity to the ore. There is almost no data available that cover the footwall and the area that house the main decline to the Nussir mine. At Ulveryggen, an approximately 1.4 km long transportation drive from the 1970’s provides a good indication of what ground conditions to expect during development and operation. This drive is still in very good condition despite the absence of a standard bolting pattern (there is limited spot bolting in the tunnel). Most of it is not supported, only spot bolting is used to keep rocks in place and to secure the area for safe traffic. Only one area requires scaling, shotcrete and bolting, and this is where the tunnel cuts through a fracture.

This suggests that a relatively low degree of rock support is required. Controlled drilling and blasting is the first step to establish stable drives in both the Nussir and Ulveryggen mines. Drilling and blasting should be followed by thorough scaling. An engineering geologist/rock mechanics engineer should then assess the rock quality at the face and give his recommendation for ground support. These recommendations have to be followed in order to insure a safe work environment both during development and throughout mining operations. The degree of rock support required is also dependent on the particular area underground. Long-term installations such as crusher hall, workshop, offices and rooms for electrical installations should be better supported than drives that are only in use for a limited amount of time.

The NGI ground reinforcement scheme, as presented in Figure 5-42 indicates that most of the drives (access tunnel, ramps and haulage drives) planned for the Nussir and Ulveryggen mine fall within support category 1. In these drives it is suggested that scaling and spot bolting is carried out where necessary. In less stable regions a rock mechanics engineer must assess the need of further support. In general, a standard pattern of 4 to 6 bolts in a ring in the roof of the drive with 4 m distance between each ring is recommended. Resin grouted bolts at 2,4 m long should be installed using a specialised bolting rig, not using a standard tunnelling rig. That is to ensure the correct installation of the bolts and to achieve the required angle between the bolts and the wall/roof (see Figure 5-43).

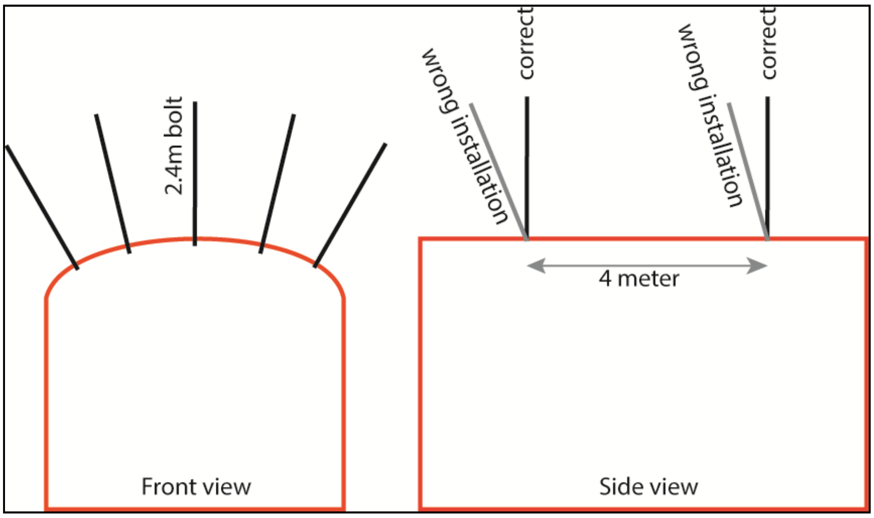

Figure 1-42: Standard ground support in access tunnel, ramp and haulage drives.

Drill drives are to be developed in ore and Q-values from within the ore often show lower rock mass qualities (fair to good) than in both the footwall and hangingwall. However, some drives may need to be mined along the contact to the hangingwall, which may lead to less stable conditions. The drives will be highly affected by stress and wear due to blasting in the stopes. In drill drives it is recommended that a standard pattern is used, which includes 4 to 6 resin grouted bolts at 2,4 m long with a 4 m separation. These are to be installed in the roof and the asymmetric wall in contact with the hangingwall, as illustrated in Figure 5-44.

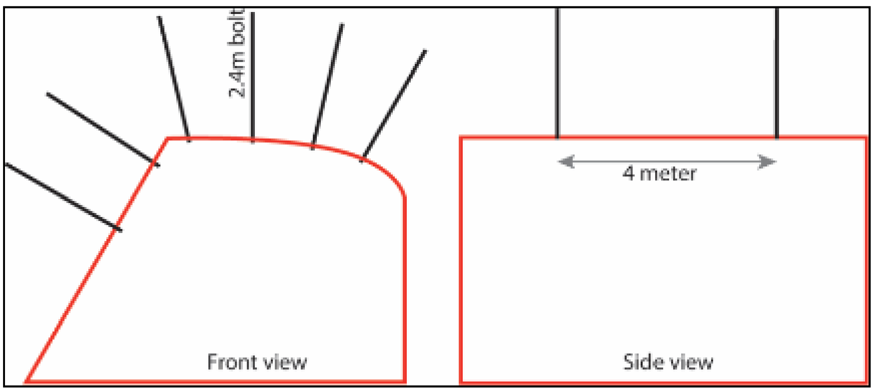

Figure 1-43: Standard ground support in drill drives.

Draw points are subject to high stress and wear during blasting and material drawing, therefore they need a higher degree of ground support including cement grouted bolts and a 4 cm to 6 cm thick layer of fibre armed shotcrete.

For permanent installations underground (crusher hall, offices, workshops) a higher degree of support is recommended. Following the NGI system these openings fall in the support category 3 to 4 which require a combination of fibre armed shotcrete and bolting. The standard pattern, as shown in Figure 5-43 , will be further supported by 5 cm to 6 cm (category 3) or 6 to 9 cm (category 4) thick fibre armed shotcrete and a 4 m x 4 m pattern of 3 m long cement grouted bolts, as illustrated in Figure 5-45 .

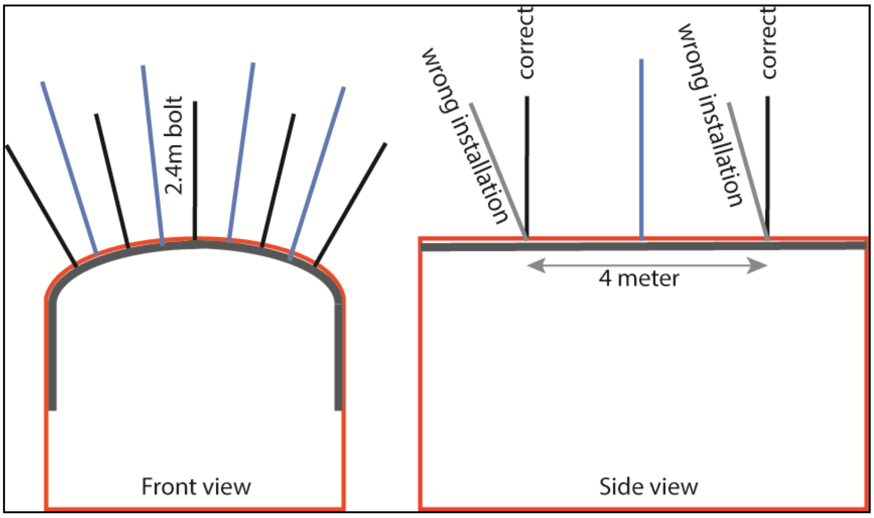

Figure 1-44: Standard ground support in permanent installations

Shafts with a diameter of 3 m to 5 m do not require ground support as they are inaccessible to workers. This is independent of the cross-section shape (circular or rectangular).

1.3.3.2 Ground Control during Production

During the production phase, all drives must be regularly monitored for changes to their rock mass quality. Orientation and strength of the local stress field can change due to mining activities, and this will affect the stability of the rock mass. All monitoring must be documented. Monitoring frequency can vary depending on the type of drive/excavation. Drives and excavations housing permanent installations (workshops, crusher hall, and offices) should be monitored more frequently (weekly) than drives which are not as frequently used. Inactive areas or areas left after production must be clearly closed off to prevent traffic or personnel entering.

If necessary, in addition to the ground support discussed in this section, further ground control such as scaling, support by shotcrete, bolting must be considered and installed following recommendation from a rock mechanics engineer or engineering geologist.

1.3.3.3 Testing of Bolts and Shotcrete Thickness

All measures taken to support ground must be controlled and documented in a log. This is normally completed during development by the contractor. The contractor should document that all ground support is installed according to specifications. This documentation must include the approximate position of bolts, length and types of bolts used, and thickness of shotcrete. Thickness of shotcrete and the effectiveness of the bolts by pull-out tests needs to be assessed. The locations of where these assessments have been performed needs to be located (surveyed) on the development plans and documented.

Details on frequency for testing are outlined in Ref. 5-12 NS-ISO 2859-1 (1999). All routines for control and testing must be implemented in a QA/QC system.

1.3.3.4 Rock Mechanics Monitoring and Modelling

As mining progresses the opening of large stopes will lead to local changes in the regional stress field; both direction and size of the horizontal stress (σ_H) may be change. This in turn can lead to higher stress and subsequent instabilities in nearby drives and other openings required for the mine. A surveillance system using 2D stress measurements at strategic points throughout the mine helps provide an early warning system. These instruments should be equipped with a data logger and readings should be taken at least once a week.

To survey the stability of the footwall it is recommended that 2 or 3 extensometers be installed. These can be installed at a 40 degree angle in the haulage drive pointing towards the stopes. Alternatively, they could be installed in the ramp up to production and drill levels. 2D and 3D models should be used for stress modelling for a better understanding of rock mechanics challenges which may be encountered. These models should be updated and updated with monitoring results. This will help to provide an improved understanding of rock mass behaviour that can be applied to mine planning.

Installation cost for 2D-doorstopper instruments is 146 000 NOK per unit, fully installed with equipment for data transfer and interpretation of data. The costs for these items were not specifically included in the mining capital or operating costs, however an annual allowance for testing, consulting and other technical support was included in the operating costs.

1.3.4 Development

Development of drives and ramps completed by a contractor should follow the New Austrian Tunnelling Method (NATM) approach. This involves a rather conventional tunnelling method applying drilling, blasting and trucking of blasted/broken material to a waste dump. The tunnelling advance must be observed by an engineering geologist or rock mechanic engineer in order to map and assess the rock mass quality. The amount of necessary rock support will be based on these observations as described in Section 1.3.3.

Drill plans and blast hole loading must be designed in order to not overload and subsequently over-blast holes. A good “contour” is required as this is the first step in avoiding high ground support costs. Half barrels of the drill holes should be visible after blasting. Excessive over-breaking cannot be accepted and corrective methods must be applied. Advance with each blast should be between 4,5 m to 5 m in the main access tunnel.

Tunnel directions are set out by a mine surveyor according to the most recent mine plan, or alternatively by surveyors with the contractor. To maintain the correct direction and gradient, rigs should be equipped with laser equipment. Despite correct set-out and available laser technology for guidance, the tunnelling advance needs to be controlled and surveyed. This is done by surveying the tunnels on a regular basis (alternate days). Centrelines of floor and roof, wall lines and arch lines need to be surveyed. This is to ensure the correct direction and dimension of the tunnels. When deviations are measured/observed corrective methods must be applied.

Development rates for Nussir and Ulveryggen are as follows:

- Nussir: 800 m per month or 400 m per single heading (2 to 3 development rigs); and
- Ulveryggen: 400 m per month (1 development rig).

Experience shows that these rates are fully achievable as there are many areas which can be developed simultaneously, therefore there should be minimal idle time of the development rigs.

1.3.5 Production Drill and Blast

The drilling plan for Nussir involves rings of parallel blast holes drilled as both up and down holes (Figure 5-46). Up holes would be drilled from the extraction levels, and both up and down holes may be drilled from the drill level above. The distance between the rings varies depending on the size of the stope. To open a stope for production a 1,2 m x 1,2 m slot is required at one end of the stope (see Figure 5-47 and Figure 5-48 showing examples for the Nussir and Ulveryggen mines). An alternative to drilling and blasting a slot, is raise drilling using a [redacted] or [redacted] drill. One advantage of using this method is a higher confidence of obtaining straight holes and the slots are guaranteed to reach the planned length. The slot is then widened to the full width of the stope. The drills ream slot holes of up to 0,8 m diameter.

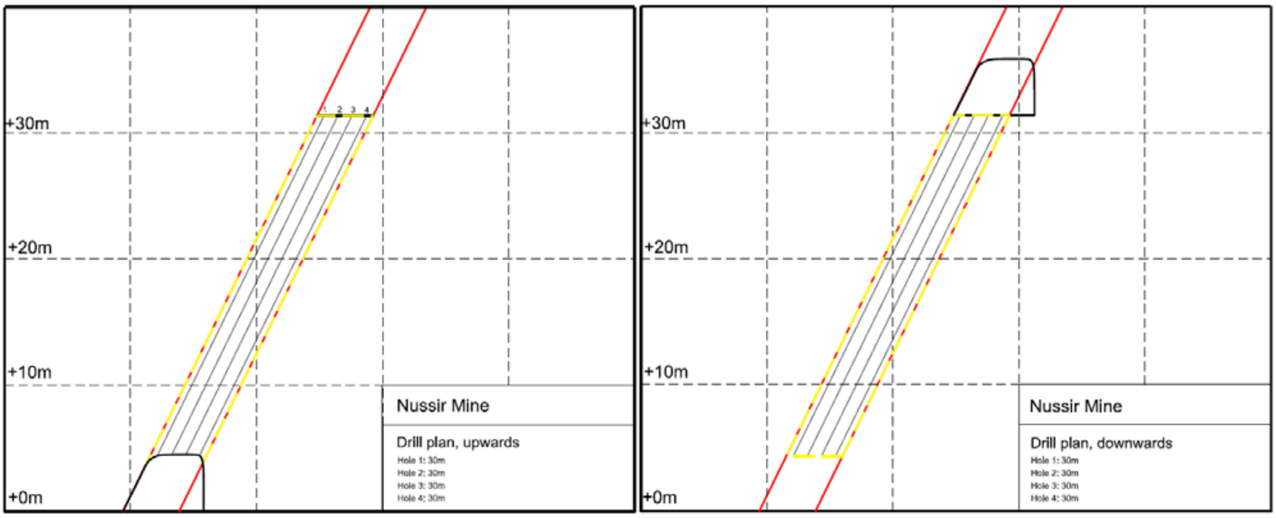

Figure 1-45: Drill Plan of Typical Blast Hole Rings at the Nussir Mine.

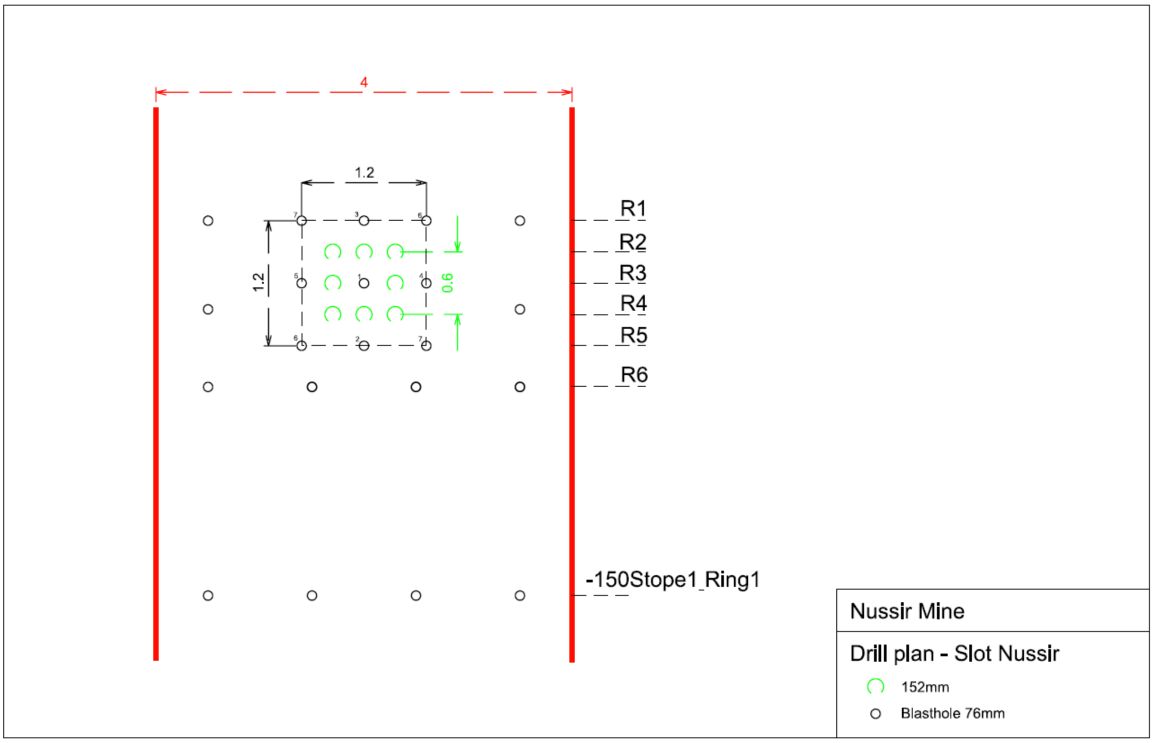

Figure 1-46: Drill Plan for a Slot and Subsequent Widening to Match Width of the Stope for the Nussir Mine.

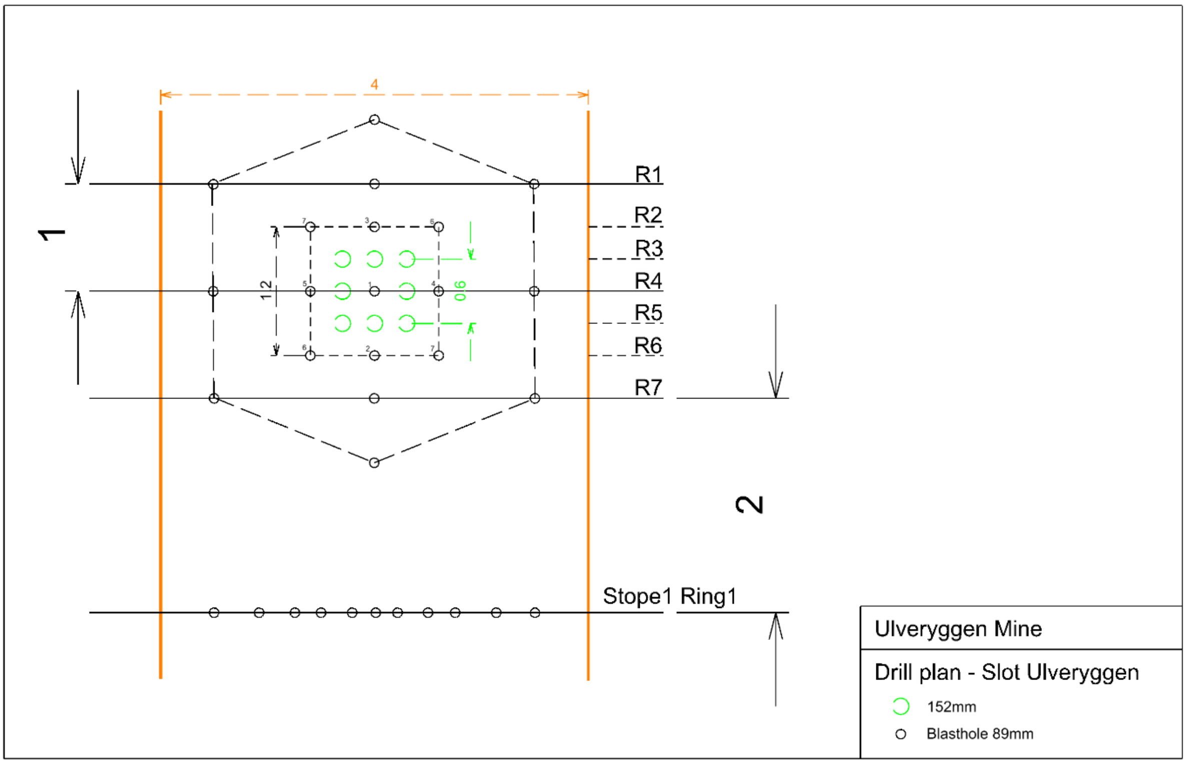

Figure 1-47: Drill Plan for a Slot and Subsequent Widening to Match Width of the Stope for the Ulveryggen Mine.

Rings of holes are loaded and blasted, starting at the slot-opening and retreating back towards the end of the stope. Rings drilled from the extraction level are blasted first, followed by rings drilled form the drill level. These up and down-hole rings are to be fired simultaneously to create a double lift stope. This process and blasting sequence is illustrated in Figure 5-49.

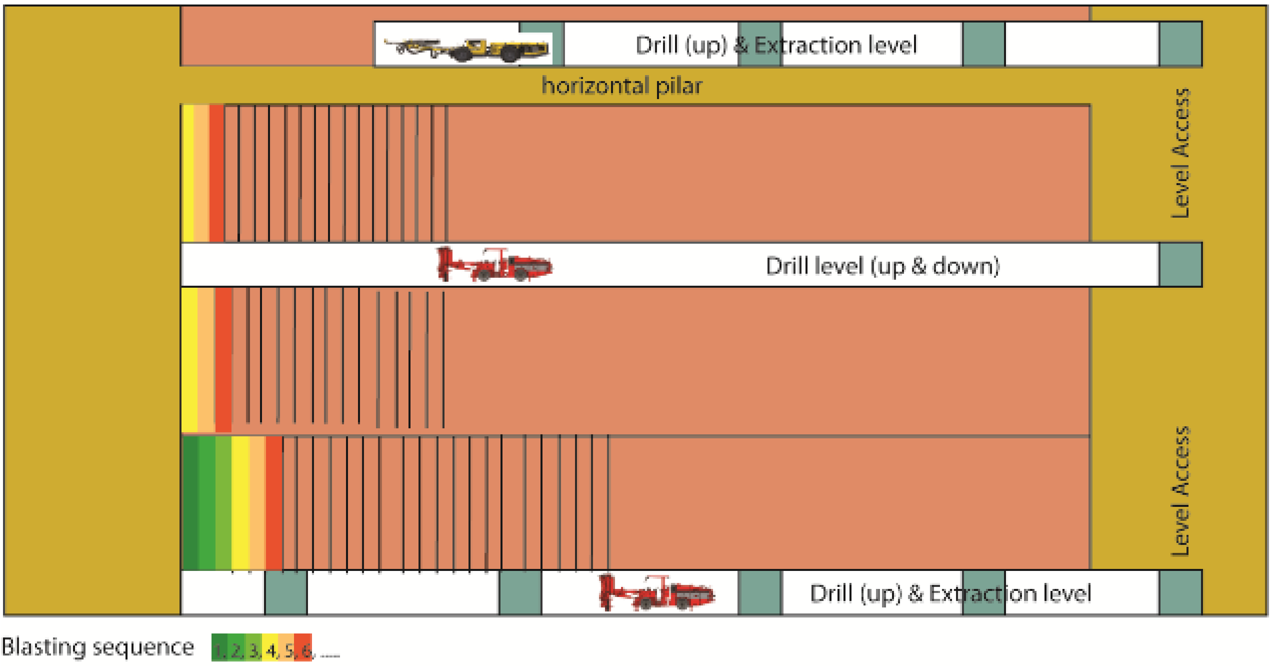

Figure 1-48: The Process of Drilling and Blasting at the Nussir Mine, the Sequence of Blasting is Illustrating by Changing Colors Assigned to the Rings.

1.3.6 Load and Haul / Transport.

Ore is mucked from draw points at the extraction level using LHDs. LHDs will then haul the ore to the crusher or alternatively to the nearest ore pass. Ore from stopes farthest away from the crusher will be mucked by LHDs and loaded onto mine trucks.

Abandoned stopes on the lowermost production level might serve as ore shafts, magazines for short-term storage, or ore mixing for grade control.

From stopes farthest away from the crusher, ore will be transported using mining trucks. Ore will be transported from ore pass to crusher by LHD, alternatively by Hæggloaders and conveyors feeding directly into the crusher.

[redacted] produced a proprietary simulation model of the proposed haulage and transport plan. The modelling of truck LHD capacities suggest that 2 load and haul machines (40 t capacity) and one mining truck (42 t capacity) are sufficient to produce up to 1 Mtpa.

Ore will be transported from the underground mine to the process plant via wide belt conveyor in the main access tunnel. The conveyor will be fed from an ore pass and chute on the -150 Level at Panel 1. The same system will be in use over the life of the mine in each of the panels with the conveyor extended to Panels 2, 3 and 0. At Panel 2 an ore pass and transfer point will be installed.

1.3.7 Mine Services

1.3.7.1 Explosive Storage

The explosive store will be located close to the access road that leads up to the Ulveryggen area, just above the industrial area housing the process plant, offices and entrance to the Nussir mine (see Figure 5-50). The area is currently used as a lay down area but will be re-contoured, flattened, and berms will be constructed between storage containers to separate explosives and detonators. A second berm will be constructed and act as a shield towards the industrial area and national road to the east. The area will need to be fenced by non-flammable material, at least 2 m high with two lines of barbed wire at the top. The fence must be placed at least 2 m away from the storage containers. Containers used must comply with government regulations for storage of explosives (Forskrift om håndtering av eksplosjonsfarlig stoff, FOR-2002-06-26-922).

A separate emulsion storage area will be established in close vicinity to the Nussir mine entrance. The emulsion tank should be stored in a 12 m x 15 m tent, large enough to house the truck/vehicle used for blast hole loading, see Figure 5-51 .There should be an 8 m stand-off distance around the tent from other facilities.

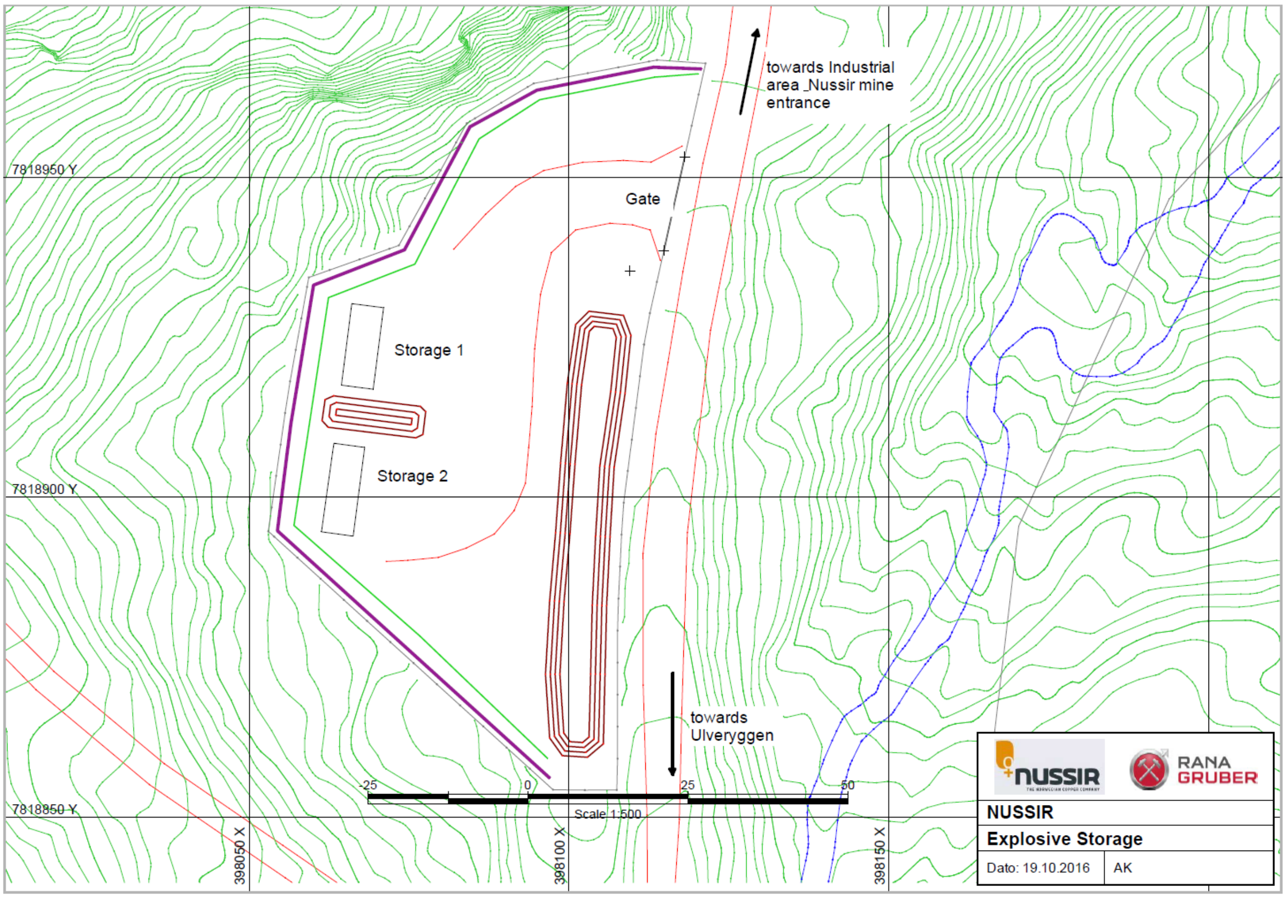

Figure 1-49: Explosive Storage at the Nussir Industrial Area.

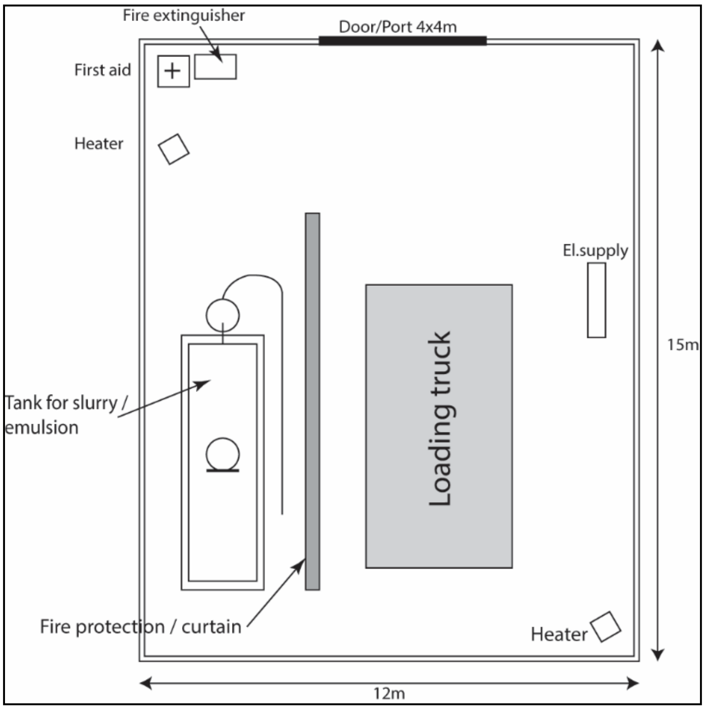

Figure 1-50: Layout of Slurry/Emulsion Storage Tent.

1.3.7.2 Fuel Storage

Fuel storage areas will be located outside the mine entrances to both the Nussir and Ulveryggen mines. The tanks will serve the fleet of mining vehicles. All tanks and fuel pumps must comply with government regulations (ref. Tankforskriften" and "Veiledning til tankforskriften"), which requires that all tanks must be placed within a leak-proof berm that can house 110 % of the volume of the tank. If several tanks are installed in one basin, this basin must be big enough to house 110 % of the largest tank installed.

1.3.7.3 Workshops

A workshop will be excavated and equipped on the main level in the Nussir mine (L-150) to service the mining fleet for Panel 1. It should contain 3 bays and a 2-axled overhead crane. A storage facility should be located nearby for close access to consumables such as air filters, oil filters, motor oil/hydraulic oil, hydraulic tubes etc. Access to water and compressed air will also be required and the workshop must be equipped with electricity and adequate lighting.

In order to be prepared for battery powered mining vehicles the workshop should be constructed to easily close off an area for battery charging and handling. This area does not need any extra requirements concerning ventilation other than that it is closed off for general traffic.

An oil/fuel trap must be installed to catch any spills before it reaches the general water settling basin.

Drill bits will be ground and maintained in a specialised container (compressed air, water, good ventilation).

Given the 2 km distance between the two mining areas, these facilities are duplicated at Panel 3 at the -225 Level Canteen and Offices.

A canteen with toilet facilities, washroom and 3 offices is planned at the main level (L-150) in Panel 1 at the Nussir mine, as illustrated in Figure 5-52. At least three offices for shift supervisors, the mining geologist and rock mechanics engineer should be provided. A meeting room for mining crews is also recommended. It could be installed as part of the cafeteria area and closed off using folding doors.

A sewage tank will be installed in close proximity to collect used water from the canteen before it reaches the general system for water handling. The canteen will be ventilated through a ventilation duct. Internal ventilation will supply the offices and washrooms with fresh air.

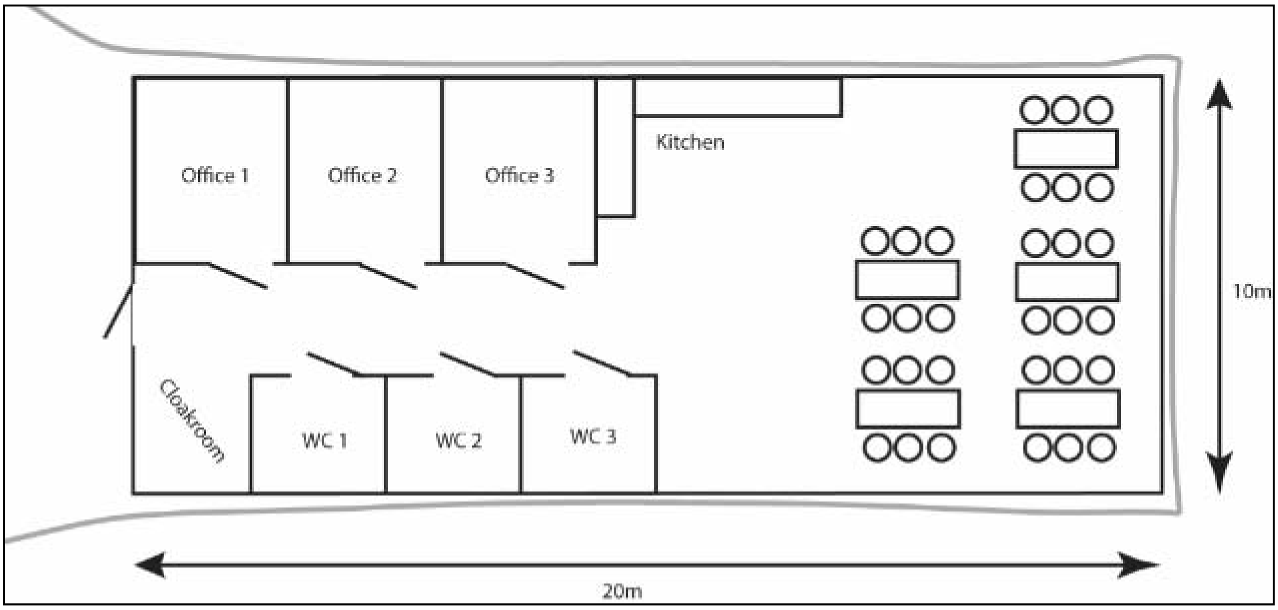

Figure 1-51: Canteen Underground Nussir.

The canteen and offices will be duplicated in Panel 3 on the -225 Level due to the 2 km distance between the two mining areas.

1.3.7.4 Electricity and Communication

Electrical power will be required in the mine for ventilation, pumping, drilling and eventually charging battery powered mining vehicles. Power is also required in the underground office facilities, canteen, workshop, and for the communication system. Power will be delivered to the electrical equipment used in the mines via a system of transformers, switch gear and cables. A list of the mining equipment with respective power consumptions is presented in Table 5-25 and Table 5-26.

Table 1-25: Nussir Mining Equipment List and Power Consumption.

Nussir Mine	Number of	Power Rating	Installed Power	Load	Effective Load	Utilisation
Electrical Load List	**Units**	**(kW)**	**(kW)**	**Factor**	**(kW/d)**	
Main Ventilation Fans	2	160	320	95 %	304	100 %
Heater (Ventilation)	4	1 000	4 000	95 %	3 800	100 %
Exhaust fan	1	132	132	95 %	125	100 %
Secondary Ventilation Fans	11	75	825	95 %	784	75 %
Crusher	2	132	264	70 %	185	50 %
Conveyor	2	200	400	80 %	320	60 %
Workshop	1	80	80	50 %	40	40 %
Canteen and Offices	1	30	30	80 %	24	20 %
Development Drill Rig (Boomer L2)	1	79	79	80 %	63	60 %
Development Drill Rig (Boomer M)	1	79	79	80 %	63	60 %
Cable Bolter ([redacted])	1	105	105	80 %	84	60 %

Nussir Mine	Number of	Power Rating	Installed Power	Load	Effective Load	Utilisation
Electrical Load List	Units	(kW)	(kW)	Factor	(kW/d)	
Raise Drill ([redacted])	1	580	580	80 %	464	60 %
Production Drill rig	3	55	165	80 %	132	60 %
Häggloader	1	76	76	80 %	61	75 %
Pumps (waste water)	3	40	120	80 %	96	80 %
Pumps (fresh water)	1	47	47	75 %	35	67 %
Lighting	1	45	45	90 %	41	100 %
Sub-Total	**37**	**2 915**	**7 347**	**90 %**	**6 621**	**86 %**
Contingency	10%	292	735		662	
Total		**3 207**	**8 082**	**90 %**	**7 283**	**86 %**

Table 1-26: Ulveryggen Mine Electric Power Requirements

Ulveryggen Mine	Number of	Power Rating	Installed Power	Load	Effective Load	Utilisation
Electrical Load List	Units	(kW)	(kW)	Factor	(kW/d)	
Main Ventilation Fans	1	160	160	95 %	152	100 %
Heater (Ventilation)	1	1 000	1 000	95 %	950	100 %
Exhaust fan	1	132	132	95 %	125	100 %
Secondary Ventilation Fans	5	75	375	95 %	356	75 %
Canteen and Offices	1	30	30	80 %	24	20 %
Production Drill rig	1	55	55	80 %	44	60 %
Pumps (waste water)	1	40	40	80 %	32	80 %
Pumps (fresh water)	1	47	47	75 %	35	67 %
Lighting	1	45	45	90 %	41	100 %
Sub-Total	**13**	**1 584**	**1 884**	**93 %**	**1 759**	**91 %**
Contingency	10%	158	188		176	
Total		**1 742**	**2 072**	**93 %**	**1 935**	**91 %**

Communication and surveillance in the mine will be based on a digital radio communication system, [redacted]. The radio/data signal will be distributed through a network of fibre optics cables and wireless base stations. The main fibre cable suppling the mine with net access is placed in a tube alongside the road in the main decline. There should also be a back-up fibre cable installed. An operating center will be installed in the mine, and a back-up center will be installed on the surface in case of emergency (preferably situated close to process plant or nearby administration building). The system can be easily adapted and expanded as the operation expands.

The communication system will cover the following items (also illustrated in Figure 5-53):

- Tracking of miners' location inside the mine.

- Radio functionality.
- Text messaging.
- Surveillance cameras (located at the crusher / mine entrance, etc.).
- Mine system surveillance.
- Remote programming.
- The collection of data concerning air quality.
- Administrate alarms (air quality, emergency).
- Email alerts.

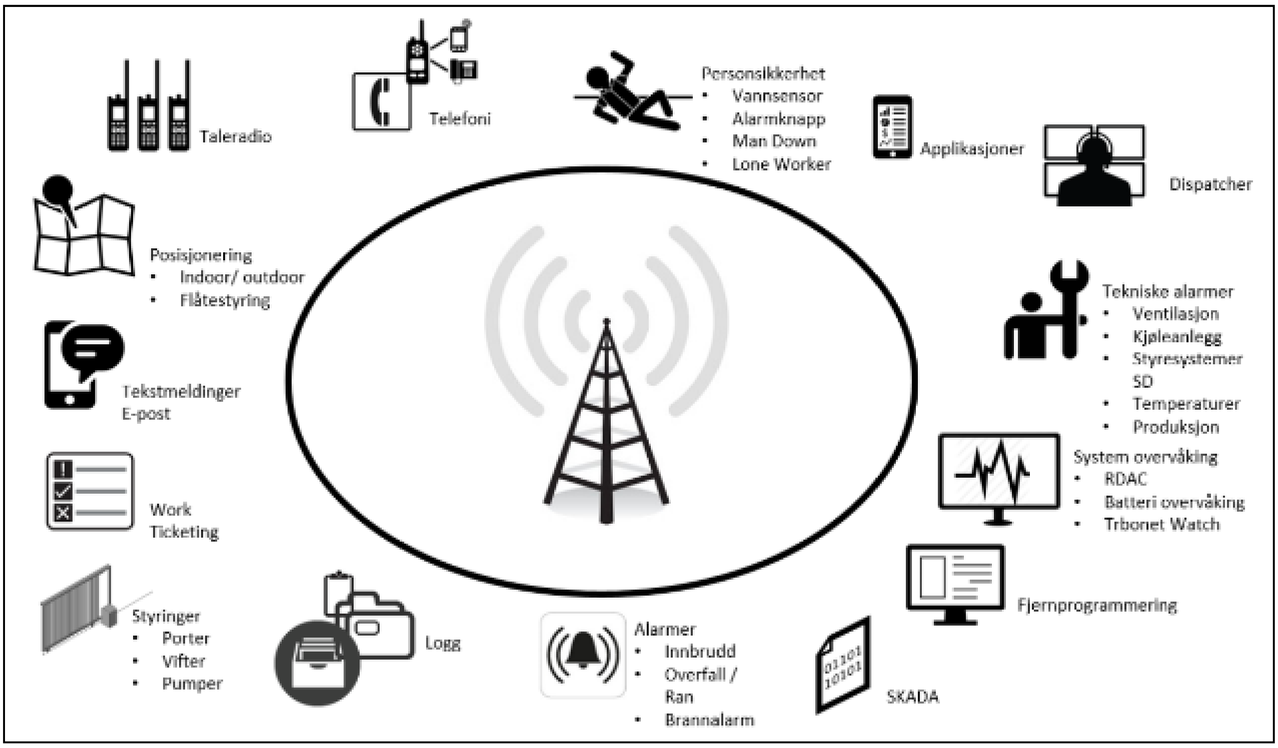

Figure 1-52: Functionality of the Motorola DMR System.

1.3.7.5 Health, Safety and environment

To ensure a safe working environment in the Nussir and Ulveryggen mines, rules for Health Safety and Environment (HSE) will be established. These rules will cover all aspects concerning safety, from personal protective equipment (PPE) and air quality monitoring, to the safe use and maintenance of large mining machines.

Air quality in the underground working environment will be regularly checked (on a 2 to 3 year basis) with regard to mineral particles (silica etc.), fibrous minerals (asbestos), and radon content.

To secure workers and operators during mine accidents, fires and rock falls several refuge chambers will be installed throughout the mine. Theses safety chambers will be self-contained underground capsules made of re-enforced steel which provides a safe retreat for workers in the event of an emergency. These chambers should safely hold 6 to 8 people for up to 36 hours. They will be air tight, stocked with water and breathable air/oxygen and hardwired for electrical power. They will contain a phone that connects to the mines communication system. In addition, each worker/operator/visitor to the mine will carry their own self-rescue devise, which can produce oxygen for up to one hour.

Emergency plans and an emergency team ("industrivern organsisajon" in Norwegian) will be established, which can be mobilised in short notice. Mine accidents and emergency situations will be simulated on a regular basis for training purposes.

1.3.8 Ventilation

Primary ventilation to the Nussir mine Panel 1 will be provided by a 3,5 m diameter ventilation shaft. Fans with silencers will be mounted on the shaft collar. The silencer will reduce the noise of the fans to 69 db. The fan will be equipped with an electrical heating unit in order to warm the air during winter. These installations will have a footprint of approximately 90 m^2. The exhaust air-flow will follow the ramp to the uppermost production level. From there an exhaust shaft will be constructed which connects to the surface. Secondary ventilation consists of fans on each extraction and drill level and flexible ventilation ducts which deliver fresh air to the active production areas.

During the development phase, especially while the main access tunnel is constructed, fresh air will be supplied from the main entrance of the mine. Air will be pushed by a fan to the heading of the access drive through flexible ducts with exhaust air flow following the incline to the mine entrance.

For a more detailed description see ventilation report (Ref. 5-13: (Bolsøy 2016)) and the associated VentSim model, a screenshot of the VentSim model is also provided in Figure 5-54.

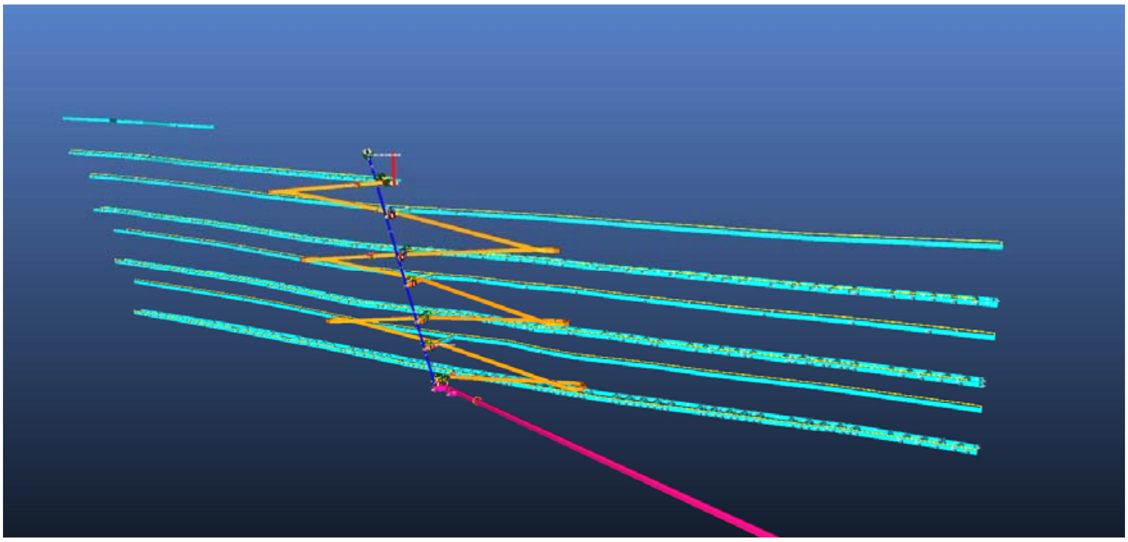

Figure 1-53: Screen Shot of the VentSim Model used to Model Air Flow in Panel 1, Nussir Mine.

The size of surface installations is restricted by certain planning conditions summarised in the Zoning Plan, therefore two options to reduce the 90 m2 footprint of the fan installations on the surface have been explored:

1) The fans could be installed in an underground working at the uppermost part of the ramp approximately 20 m below surface, as shown in Figure 5-55. From there, a second raise will be drilled or blasted which will be used for ventilation. Once finished, the original shaft must be capped to avoid air leakage. Installation above surface can then be reduced to a rather modest sized housing (approximately 20 to 25 m^2) on top of the shaft where heaters would be installed to keep the shaft free of ice and challenges associated with frequent freezing and ice.

2) Alternatively, a new 5 m diameter ventilation shaft could be constructed. This would eliminate the need for fans at or close to the surface. All air would be drawn down into mine by the secondary fans which would be placed on the extraction and drill levels.

For costing purposes, the ventilation system proposed for Panel 1 will be duplicated in Panel 2 to service access to Panel 3, and at Panel 3 to ensure adequate ventilation during production. More detailed analysis of the ventilation network will be completed during the detailed mine design phase of the project (i.e. during the feasibility study).

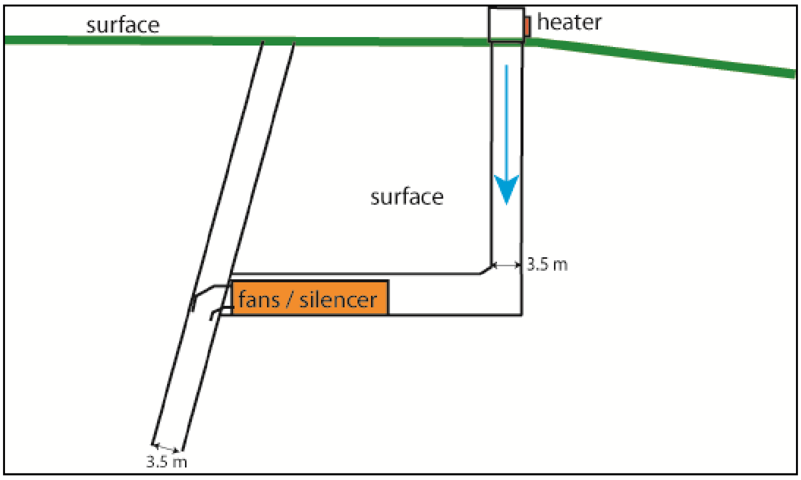

Figure 1-54: Alternative 'A' - Ventilation Fans Installed Underground.

Primary ventilation to the Ulveryggen mine will be provided by existing shafts previously used as ore passes; a fan will deliver air into the mine. The fresh air will be distributed around the mine by secondary fans and flexible ducts to the working areas. Exhaust air will exit through the access tunnel. Similarly to the Nussir mine the primary ventilation fan will be equipped with an electrical heater to warm the air during winter.

1.4 Hydrogeology

1.4.1 Hydrogeology and Mine Dewatering

Studies relating to the Nussir Project undertaken prior to this PFS have included minimal information regarding hydrogeological conditions surrounding the proposed mines and the potential effects of groundwater on operations.

The objectives of this component of the PFS study are to:

- Draw together existing regional climate, geology and hydrogeological information and local hydrogeological data (where available) to develop a hydrogeological conceptual model for the area of the Nussir and Ulveryggen ore bodies.
- To estimate the groundwater inflow to the Nussir mine workings.
- To propose preliminary strategies for groundwater management.
- To outline additional investigations required to develop the understanding of the local hydrogeological regime and improve accuracy of predictions of the nature and quantity of groundwater inflow to the mine.

1.4.1.1 Mine Context: Location and Topography

The Nussir Project is located in northern Norway in an area of Pre-Cambrian bedrock exposure. Superficial deposits (Quaternary drift and other more recent unconsolidated sediments) are thin across much of the Project area. Bedrock permeability is low and the area receives comparatively high rainfall and low evaporation. In this environment, groundwater levels are close to surface and marshy areas and surface water bodies are frequent: these can be considered "groundwater outcrops" where boggy ground or open water has developed as a result of the shallow water table. There are two fundamental drivers to groundwater flow in this type of environment:

- Structural features in the low permeability bedrock which may act as conduits for groundwater flow.
- Topography, which results in gravitational drainage.

The proposed mining area is situated approximately 3 km from the coast. Topography falls steeply to the north-east toward the coastline from an elevation of between 300 m above sea level (mASL) to 500 mASL in the mine area to close to sea level at the coastal location of the processing plant and port (Figure 5-56). Locally, a steep south-facing slope marks the edge of the Nussir ridge falling from 500 mASL at the top of the Nussir ridge to 300 m in the valley. The proposed Nussir mine lies to the north-east of the steepest part of the ridge line.

A strong south-west to north-east alignment in surface topography is visible in topographic (LIDAR) data for the area.

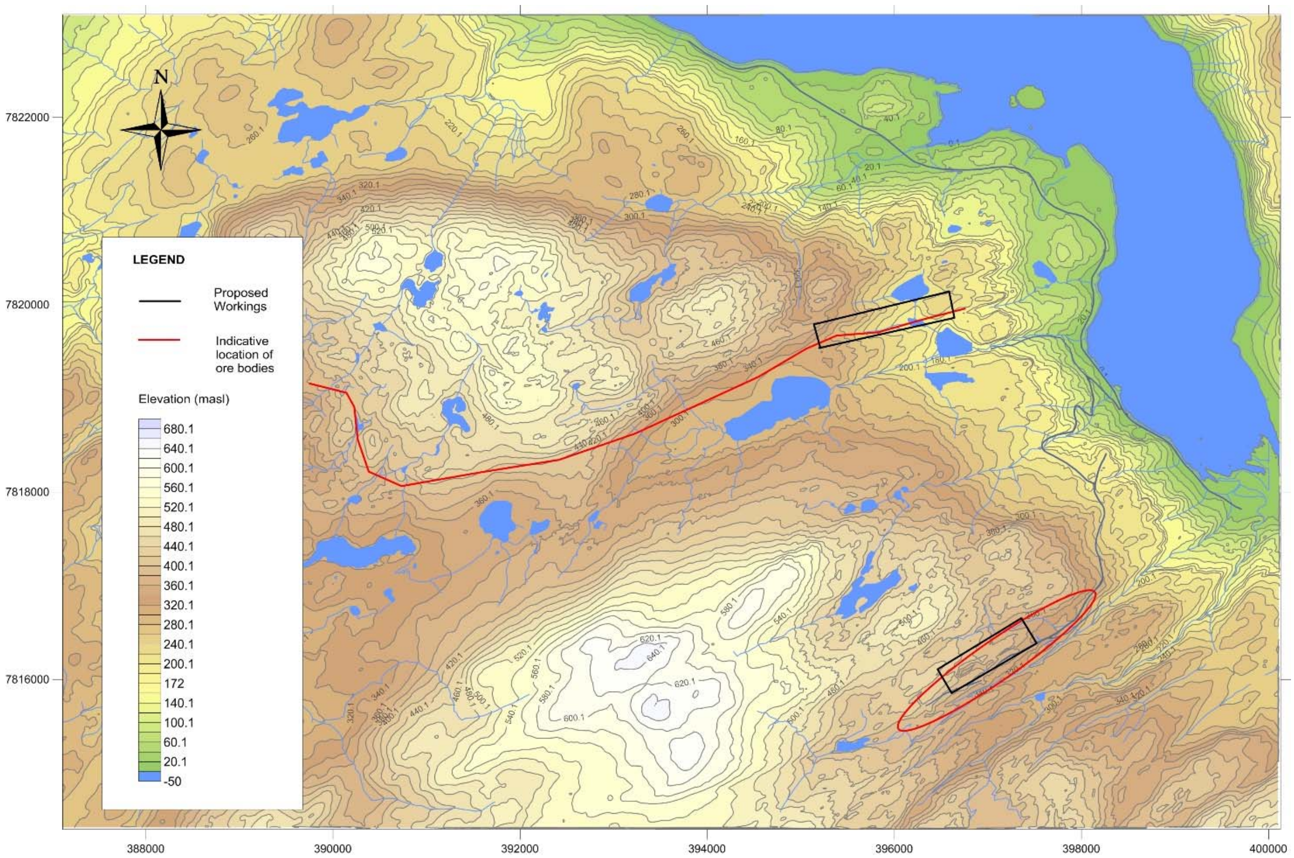

Figure 1-55: Topography of the Mine Vicinity

1.4.1.2 Climate and Groundwater Recharge

Given the geological and hydrogeological setting of the Site, with predominantly thin unconsolidated cover, low permeability bedrock, and low evaporative losses, groundwater recharge will often be restricted due to saturated conditions at surface. For the purposes of understanding the availability of groundwater to the mine, it is considered that at this stage in the mine's design an assumption that an effectively unlimited source of water is provided at surface by infiltration, maintaining groundwater levels close to surface, is reasonable for this site setting. It is improbable that rate of infiltration to the underground mine could exceed the availability of water as a result of infiltration at surface. No estimation of groundwater recharge has therefore been undertaken within this study, though this should be considered at future stages of the project development.

1.4.1.3 Hydrology and Surface Water Quality

Surface water courses and water bodies in proximity to the mine are indicated in Figure 5-56 . As noted above, there are numerous surface water bodies in proximity to the mine, considered to be indicative of a shallow water table and generally boggy conditions. The area south of Nussir ridge is indicated to be boggy in an extended region surrounding the lake (Ásajárvi). Surface water flows predominantly to the north-east toward the fjord and coastline.

The water courses most likely to be influenced by the proposed mining activity are:

- The Dypelva (“Deep River”) from the Damvatn reservoir and downstream.
- Unnamed streams and water bodies in the vicinity of the Ulveryggen ore body.
- Øyennvannen lake and Geresjohka stream situated between the Nussir and Ulveryggen ore bodies.
- Repparfjord.
- The upstream Dypelva including Ásajárvi Lake, and adjacent lakes on adjoining tributaries (“Fiskevatna” and “Langvatnet”). This area may potentially be affected by changes to the groundwater flow regime, but is less likely to be affected due to its location upstream of the mine entrance.

Limited data regarding surface water flow rates has been identified in the published sources of information regarding the project area. However, a baseline surface water quality evaluation has been conducted (Ref. 5-14 (NIVA 2011)) which incorporated sampling and analysis of streams across the project area. Routine monitoring is also undertaken by Finnmark Recycling of the flows entering and discharging from the Ulveryggen haulage adit. Whilst the sampling completed was limited in scope and may not capture temporal variability or localised effects, generally the waters sampled were of circum-neutral pH, with comparatively high alkalinities/carbonate content and low metals concentrations. Discharge from the Ulveryggen mine has elevated copper concentrations in comparison to other water courses, and in excess of Environmental Quality Standards.

1.4.1.4 Hydrogeology

Hydrogeological Units

Figure 5-57 illustrates the distribution of unconsolidated deposits (published mapping, Ref. 5-15 NGU 2016) overlying the crystalline basement across the project area. Much of the area has no superficial deposits at surface or a thin cover of colluvium derived from bedrock weathering. Residual moraine and accumulated peat bog is present in valleys in the upland area. Adjacent to the coast, marine beach and tidal deposits are present. Along the inland margins of Repparfjord, alluvium in present in the main river valley and stream outlets.

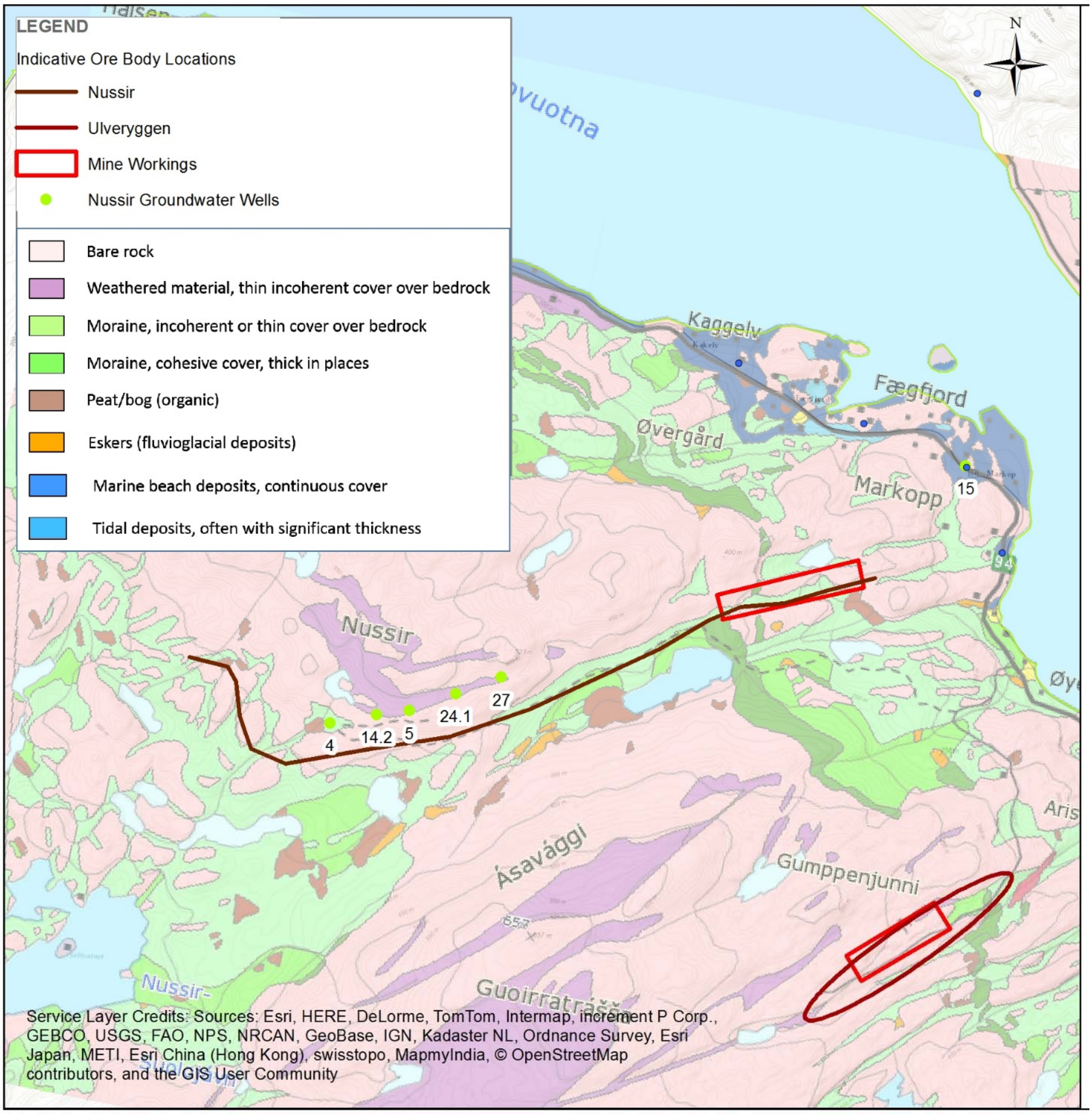

Figure 1-56: Superficial Deposits (NGU, 2016) and Depth to Groundwater

In the areas of proposed mine working, groundwater in superficial unconsolidated deposits is unlikely to be significant as a potential resource for water supply (though it will act as a conduit in flow from the bedrock zones potentially influenced to surface waters). Thicker unconsolidated sediments along the fjord coastline may potentially form local aquifers, but all of the wells mapped by Ref. 5-15: (NGU 2016) in the project area are recorded to be installed in bedrock.

Figure 5-58 illustrates the published 1:50,000 bedrock geology of the project area. The Nussir Group comprises a sequence of metabasalts, tuffs, tuffites and mafic volcanics. The Nussir Group comprises a sequence of metabasalts, tuffs, tuffites and mafic volcanics. The crystalline rocks of the Nussir Group are likely to have very low intrinsic porosity or permeability, but will permit groundwater flow through the development of secondary fracture porosity. The permeability of such deposits is nonetheless expected to be low. Groundwater flow is likely to be dominated by flow in more permeable zones associated with geological structures (fault and fracture zones). The Nussir Group is steeply dipping to the north, and permeability is likely to be influenced both by the steeply angled bedding within the deposit, and anisotropy developed as a result of phases of structural deformation.

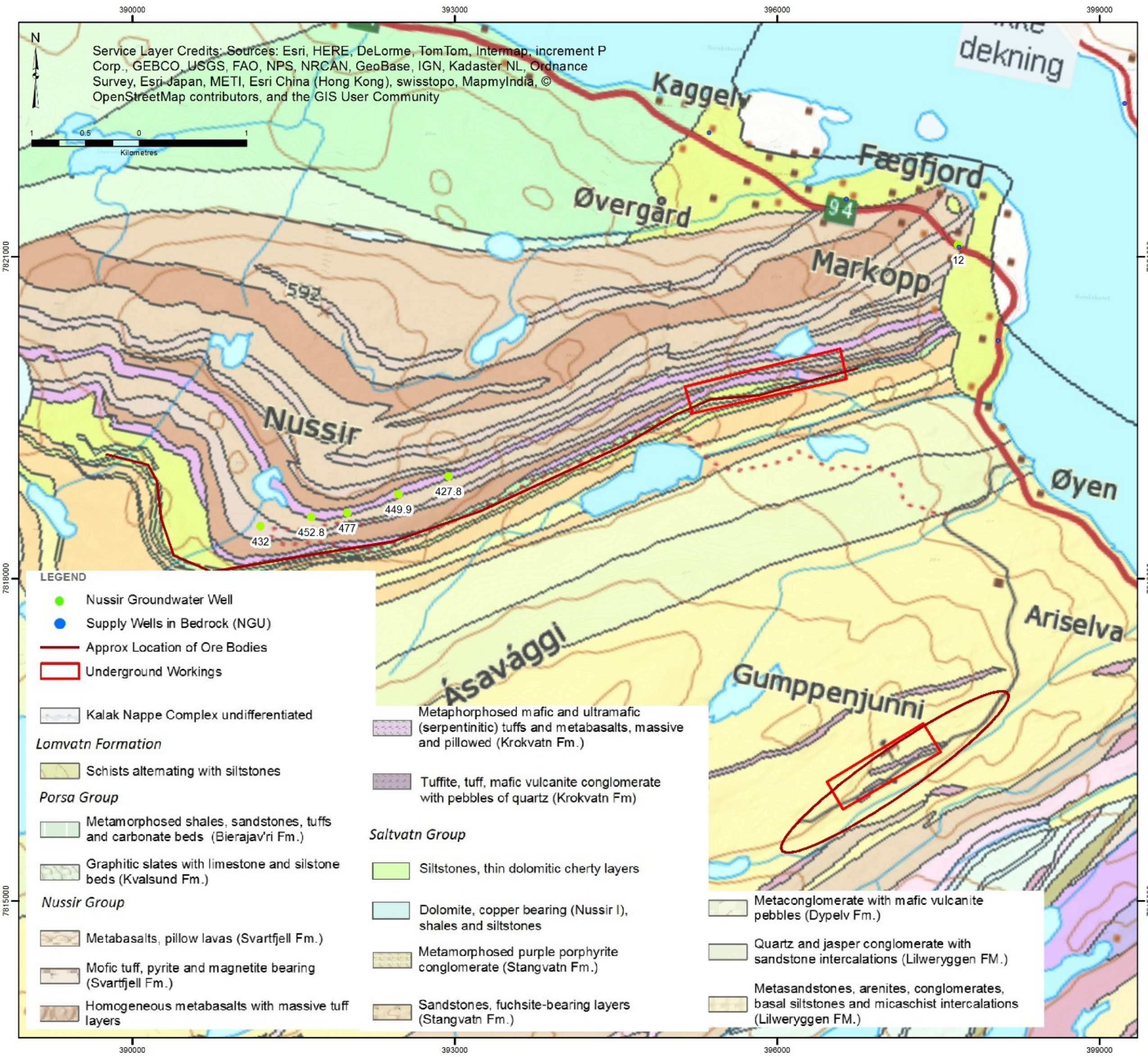

Figure 1-57: Regional Bedrock Geology (NGU, 2016), and Groundwater Elevation Data

The Nussir Group is bounded (and overlain) to the north by the Porsa Group, a series of metamorphosed shales, sandstones, and slates with interbedded carbonates and siltstones. A dolomite horizon lies at the base of the Porsa Group. Whilst these metasediments have the potential to have primary porosity and permeability to support groundwater flow, these rocks are part of the Reppardfjord window of Precambrian rocks and as such are likely to have little residual primary porosity as a result of metamorphosis and secondary cementation. It is considered that, similar to the Nussir Group, groundwater flow in the Porsa Group will be dominated by fracture porosity and structurally controlled zones of higher permeability.

The Nussir Group is bounded (and underlain) to the south by the Saltvatn Group, a series of metamorphosed siltstones, sandstones, conglomerates and dolomite. The Nussir ore body itself is hosted within dolomites which are situated in the upper part of the Saltvatn Group sequence. As with the Porsa Group, whilst the Saltvatn Group comprises metasediments which would have had significant primary porosity and permeability prior to metamorphosis and burial, due to age and the extent of tectonic influence on this area, it is unlikely that significant primary permeability remains. It is considered that, similar to the Nussir Group, groundwater flow in the Saltvatn Group will be dominated by fracture porosity and structurally controlled zones of higher permeability.

Geological Structure

In hydrogeological terms, the current geological interpretation has significance for groundwater flow:

- Fault zones bounding the Nussir Group, particularly in the south, are potentially key in controlling groundwater flow in this area.
- The thrust fault bounding the Nussir Group to the south, if present, is likely to be in close proximity to the mine workings, where this fault is intersected, this has the potential to significantly increase groundwater flow, and groundwater flow in proximity to the fault zone is also likely to be greater than in the rock as a whole.
- Secondary transpressional faulting and smaller scale faults are likely to also influence groundwater flow, for instance exploration has identified a fault situated between boreholes NUS-DD-15-026 and NUS-DD-15-020, the far left and second left wells in Figure 5-58 .
- There are numerous tectonic events which will have produced cleavage and jointing in the host rock, in addition to larger scale fracturing and faulting. The orientation of folding is predominantly between ENE to WSW and northeast to southwest. Both axis parallel jointing and joint sets developing in response to the overall compressional regime are likely to enhance permeability in the general north-east to south-west (i.e. aligned to the topographically driven flow direction) in comparison to that in the north-west to south-east orientation.
- Bedding/layering orientation in the Nussir Group is also likely to promote south-west to north-east flow whilst inhibiting flow to the north-west or south-east.

The degree of influence of the fault bounding the Nussir Group to the south on groundwater flow will depend on the properties of this fault zone. Lithological descriptions in wells subjected to hydraulic testing do not identify any brecciation or fault rock at the contact. The preserved contact zone also observed in numerous drill cores, confirms that there is a continuous primary depositional contact between the Saltvann sedimentary units to overlying Nussir volcanic units.

At a regional scale, the geological structure is considered likely to promote groundwater flow in the Nussir project area toward the coast, consistent with the expected groundwater flow regime. However, the geological structure may inhibit localised flow (e.g. from the Nussir ridge to the south-east toward Ásajárvi Lake and Dypelva) to the north-west or south-east.

The ore bodies of the Ulveryggen mine are indicated to be fault-controlled, probably associated with north-east to south-west orientated faulting in this area. The fault zone which hosts the ore body is likely to act as a conduit for groundwater flow and is indicated by geophysics to provide a preferential flow pathway. There is therefore a potential for rates of groundwater inflow into the Ulveryggen mine to be higher than into the Nussir mine.

Groundwater Levels and Flow

Groundwater level has been sporadically recorded (typically at the time of drilling/commissioning) by the Norwegian Geological Survey (NGU) in catalogued domestic supply wells in proximity to the project area

(Ref. 5-15 NGU 2016). In addition to this data, groundwater levels were recorded in 5 deep boreholes along the Nussir ridge (intersecting the Nussir ore deposit) in April 2016, (Poyry report, 2016). Groundwater level data for wells in the proximity of the project area are shown in Table 5-27 and illustrated in Figure 5-58 (groundwater elevation) and Figure 5-59 (depth to groundwater).

Table 1-27: Groundwater Levels Recorded April 2016 (Poyry 2016)

Borehole	Easting	Northing	Elevation (mASL)	Azimuth (degrees)	Inclination (degrees)	Length (m)*	Date of Measurement	Groundwater Elevation (mASL)	Depth to Groundwater (m)
NUS-DD-15-026	391192	7818489	436	175	83	502 *(521)*	24/04/2016	432	4
NUS-DD-14-030	391666	7818572	467	162	83	583,6 *(637)*	26/04/2016	452.8	14.2
NUS-DD-15-020	392001	7818613	482	160	84	561 *(552)*	21/04/2016	477	5
NUS-DD-15-021	392475	7818787	474	160	83	595	19/04/2016	449.9	24.1
NUS-DD-15-022	392942	7818955	454,8	160	83	575 *(575)*	28/04/2016	427.8	27

** Lengths given are assumed to be the open borehole lengths at the time of dipping/hydraulic testing, values in brackets are the total drilled depth indicated by the Nussir geology database.*

The available data suggests that groundwater is close to ground surface across the project area, reaching 20 m below ground level (mBGL) to 30 mBGL on higher ground (as indicated by wells on the Nussir ridge), and reducing to 10 mBGL to 15 mBGL below ground closer to the coast. It is expected that the water table will become shallower approaching lakes and streams, and that these features are in connection with groundwater and are groundwater fed.

Based on the available data, groundwater elevations are expected to broadly mirror topographic elevation, with groundwater flow occurring locally to the north and south to local streams/valleys and at a wider scale to the north-east toward Repparfjord. It is noted that three lakes are in close proximity to the proposed Nussir workings, these lakes form a tiered sequence falling south. The most northerly lake, slightly north of the proposed mine, has a surface elevation of 204 mASL in recent survey data, the middle lake (overlying the mine workings) a surface elevation of 186 mASL and the lower lake (south of the mine workings) an elevation of 170 mASL. It is not improbably that groundwater flow to the south in this area is hindered by geological structure, and there will be a similar gradient in the groundwater profile.

As described above, the geological structure of the area is considered likely to promote groundwater flow to the north-east toward Repparfjord. However, locally geological structure could have a strong influence on groundwater flow, and where mine workings encounter fault or fracture zones, this is likely to dominate groundwater inflow.

Hydraulic Properties

Nussir Mine

The hydraulic properties of the Nussir Group have been investigated through Posiva Flow Log/Difference (PFL DIFF) method in 5 boreholes along the Nussir ridge, at locations indicated in Figure 5-58 (Poyry, 2016). The following wells investigated are described in Table 5-27, further details are as follows:

- NUS-DD-15-020: encountered Nussir Group volcanites to 462 m, hanging wall sediments (Saltvatn Group) to 532 m, ore to 537 m, footwall sediments to 543 m and sandstone to 552 m. Flow logging was undertaken to 500 m.
- NUS-DD-15-021: No lithological data available, flow logging was undertaken to 260 m.
- NUS-DD-15-022: encountered Nussir Group volcanites to 495 m, hanging wall sediments (Saltvatn Group) to 559 m, ore to 572 m, footwall sediments to 575 m. Flow logging was undertaken to 431 m.

- NUS-DD-15-026: encountered Nussir Group volcanites to 428 m, hanging wall sediments (Saltvatn Group) to 513 m, ore to 517 m, footwall sediments to 521 m (clay). Flow logging was undertaken to 439 m.
- NUS-DD-15-030: encountered Nussir Group volcanites to 545 m and hanging wall sediments (Saltvatn Group) to 637 m, ore to 537 m; a brecciated zone was identified in 626 m. Flow logging was undertaken to 449 m.

The flow logging in general did not extend to the elevation of the ore deposit, generally being completed within the Nussir Group volcanites, and upper sedimentary sequence of the Saltvatn Group. In three of the five boreholes, flow logging did not reach the Nussir Group/Saltvatn Group contact (putative fault zone location). In one of the two wells in which logging extended to the Saltvatn Group, a net outflow from the borehole was reported in the lower 300 m (despite pumping from the borehole) and no test results could be obtained in this section. As a result, only one of the five boreholes tested (NUS-DD-15-026) provides permeability data relating to the Nussir Group/Saltvatn Group contact, and none provide data relating to the ore body and underlying deposits.

Mining of the Nussir ore deposit is proposed to be undertaken via a bench-stoping method. The ore deposits itself is a few metres wide and dips steeply (at around 60° to 65°) to the north. The access ramps for the workings are proposed to be situated in the footwall, to the south of the stopes, i.e. in the Saltvatn Group. The stopes themselves will extend parallel to the ore body, which is stratigraphically bound. If the hydraulic properties of the bedrock vary stratigraphically, the testing undertaken in the Nussir Group volcanites may not be directly relevant to the properties of the bedrock in the vicinity of the proposed workings. If, on the contrary, the hydraulic properties are primarily uniform throughout the Saltvatn and Nussir Groups, values for the rock mass to the north of the proposed area of working may be relevant to the mined area.

Logging in NUS-DD-15-026 identified a single point of inflow around or below the elevation of the Nussir Group/Saltvatn Group contact. Inflow in this zone was not markedly high in comparison to other inflow points. In this well, the frequency of flowing fractures decreased considerably with depth, such that the reported transmissivity in the contact zone was considerably lower than closer to surface (this pattern is not replicated in other wells). There is no evidence, from testing in this single well, that the (potentially faulted) contact between the Nussir Group and the Saltvatn Group is particularly permeable in comparison the surrounding country rock.

The reported frequency of flowing features and calculated transmissivity and permeability in each of the tested boreholes is summarised in Table 5-28.

Table 1-28: Summary of Results of Posiva Flow Log (PFL) Testing, All Boreholes

Elevation	Depth below ground (m)	Number of Flowing Fractures (/100 m except where indicated)	Transmissivity (m^2/s)	Hydraulic Conductivity (m/s)
NUS-DD-15-020				
Average	*All*	*28*	*$3{,}41 \times 10^{-7}$*	*$2{,}20 \times 10^{-9}$*
382,55	100	7	$3{,}17 \times 10^{-7}$	$3{,}17 \times 10^{-9}$
283,10	200	8	$2{,}35 \times 10^{-8}$	$2{,}35 \times 10^{-10}$
183,64	300	6	N/A*	N/A*
84,19	400	4	N/A*	N/A*
-15,26	500	3	N/A*	N/A*
NUS-DD-15-021				
Average	*All*	*30*	*$2{,}64 \times 10^{-7}$*	*$6{,}89 \times 10^{-10}$*
374,7	100	13	$5{,}81 \times 10^{-8}$	$5{,}81 \times 10^{-10}$

Elevation	Depth below ground (m)	Number of Flowing Fractures (/100 m except where indicated)	Transmissivity (m^2/s)	Hydraulic Conductivity (m/s)
275,5	200	11	1,88x10^{-7}	1,88x10^{-9}
176,2	300	6	1,76x10^{-8}	1,76x10^{-10}
77,0	400	0	0	0
53,2	424	0	0	0
NUS-DD-15-022				
Average	*All*	*51*	*1,34x10^{-5}*	*3,39x10^{-8}*
355,5	100	5	8,18x10^{-6}	8,18x10^{-8}
256,3	200	8	1,74x10^{-6}	1,74x10^{-8}
157,0	300	13	2,13x10^{-6}	2,13x10^{-8}
57,8	400	19	1,3x10^{-6}	1,3x10^{-8}
20,1	438	6	4,39x10^{-8}	1,16x10^{-9}
NUS-DD-15-026				
Average	*All*	*52*	*9,02x10^{-6}*	*1,87x10^{-8}*
336,7	100	16	6,12x10^{-6}	6,12x10^{-8}
237,5	200	17	2,48x10^{-6}	2,48x10^{-8}
138,2	300	11	1,62x10^{-7}	1,62x10^{-9}
39,0	400	6	2,13x10^{-7}	2,13x10^{-9}
-60,3	500	2	3,37x10^{-8}	3,37x10^{-10}
NUS-DD-14-030				
Average	*All*	*32*	*4,08x10^{-7}*	*8,19 x10^{-10}*
367,7	100	8	1,3x10^{-8}	1,3 x10^{-10}
268,5	200	10	1,53x10^{-7}	1,53x10^{-9}
169,2	300	3	1,79x10^{-8}	1,79 x10^{-10}
70.0	400	6	1,25x10^{-7}	1,25x10^{-9}
-59.0	530	5	9,87x10^{-8}	7,59 x10^{-10}

** An outflow from the borehole was recorded in these intervals, despite pumping from the well, such that a result value could not be determined.*

The hydraulic conductivities reported by the testing are generally low, between 1x10^{-8} m/s and less than 1x10^{-10} m/s. A reduction in hydraulic conductivity with depth is evident in one of the five boreholes tested. The variation in hydraulic conductivity between boreholes is considerably greater than the variation with depth in most cases. Overall, the data suggests that no strong depth-dependency occurs.

NUS-DD-15-030 lies close to the position of a putative north-south trending fault identified in the mine geological model/resource evaluation. The conductivity in this borehole is not notably higher than other locations. The hydraulic conductivity in NUS-DD-15-022 is greater than in other wells by at least an order of magnitude. Based on the available geological mapping and lithological descriptions, no specific feature is identified to account for this difference.

An outflow from NUS-DD-15-020 was recorded between 212 m and 500 m, despite pumping from the well, such that a result value could not be determined. This results is suggestive of non-negligible permeability in this zone. Two brecciated zones were identified in this well at around 500 m (below the Nussir/Saltvatn contact, but above the Nussir ore zone).

The hydraulic testing data suggests that the hydraulic conductivity of the bedrock hosting the Nussir mine is likely to be relatively low ($<1x10^{-8}$ m/s). However, the data is not sufficient to determine:

- Whether the upper Saltvatn Group is more permeable than the Nussir Group.
- Whether the contact between the Saltvatn Group and the Nussir Group is a permeable zone (although the limited data available and particularly lithological description suggests that is not).
- Whether either the earlier or later phases of faulting identified in the structural interpretation is associated with zones of high permeability.

Ulveryggen Mine

- The testing data available is of limited value in understanding hydraulic properties surrounding the proposed Ulveryggen Mine. This mine is at considerable distance from the testing site and in a different lithology. Furthermore existing structural interpretations suggest that the ore deposit is associated with faulting and that that faulting is also associated with groundwater occurrence. As such, extrapolation of the hydraulic testing in the Nussir area to the Ulveryggen deposit is considered inadvisable.

- Away from faulted zones, given the age and degree of alteration of the rocks, it is probable that the hydraulic conductivity of the lower Saltvatn Group is not much higher than that identified in the Nussir area (i.e. less than $1x10^{-8}$ m/s). However, where faulting has occurred, permeability may be considerably higher.

Groundwater Quality

No groundwater quality information was available for review as part of the current study. However concentrations in surface water are likely to be both generally lower than in groundwater due to dilution effects, and more influenced by surface activities.

The GeoDE Consult AS, 2016 report (Appendic 5-9) presents an assessment of the potential for contaminated mine drainage from the Nussir Project mines. The findings of this study are summarised as follows:

"*Abundant calcite and dolomite minerals are found in the ore zone and wall rocks. These carbonate minerals have a buffering effect and would prevent the generation of acid mine drainage. Sulphides in the deposits are dominated by the copper minerals bornite and chalcocite. These sulphides are documented to be resistance to oxidation. Acid-base accounting results conducted on a tailings sample indicate that mine waste will not generate acid rock drainage. Leach testing showed that very little copper was leached from the sample.*

Investigations of other analogous sediment-hosted copper district such as Kupferschiefer and Central African Copperbelt show that acid mine drainage is rare. Baseline geochemical studies of drainage from the historical workings at Ulveryggen and creeks and rivers in the area show neutral pH and low concentrations of copper. It is therefore predicted that a neutral drainage with minor copper concentrations can be generated after cessation of mining. The geometry of the ore bodies, good mining practices, and good mineral waste management will limit the amount of copper sulphides that will remain after mining and be exposed to oxidation and leaching. Filling of the mine workings with groundwater after closure will most likely also mitigate the release of copper by reducing the potential for oxidation of copper sulphides."

Whilst this conclusion is valid based on the data presented, it should be noted that:

- The assessment considered acid-base accounting undertaken only on a single tailings sample, this may not be representative of the wall rock drainage or other waste rock.

- The volcanites which form the 'hanging wall' of the Nussir Mine (the Nussir Group) are likely to contain less carbonate minerals than the Saltvatn Group and may behave differently to both the tested sample and the existing Ulveryggen Mine.
- Elevated copper (more than 60 times the Norwegian MAC EQS and typical surface water baseline concentration) has been observed in the current discharge from the Ulveryggen mine, interpreted to be associated with the existing open pits and waste dumps; based on existing information, it must reasonably be assumed that discharge from future workings will have similar copper concentrations.

Conceptual Model Summary

Recent hydrogeological testing has refined the understanding of hydrogeological conditions in the vicinity of the Nussir Project, confirming that volcanites of the Nussir Group are of low permeability associated with flow in the secondary fracture porosity and that groundwater is close to ground surface in the project area.

The overall features of the hydrogeological regime are relatively well constrained:

- Unconsolidated superficial deposits are absent across higher ground in the project area, with generally sporadic accumulation in valleys, groundwater flow in alluvial or superficial aquifers is unlikely to significantly influence the groundwater flow regime controlling mine inflow or dewatering.
- Groundwater is close to ground surface across the project area, locally discharging the north and south to lakes and streams and at a wider scale flowing north-east toward Repparfjord.
- The Pre-Cambrian rocks of the Repparfjord window are of low permeability with flow dominated by secondary fracture porosity; groundwater flow is likely to be dominated by geological structures. Given the understanding of the deformation history of the area, these structures are largely likely to promote groundwater flow to the north-east toward Repparfjord.
- Given the high rainfall, thin/absent superficial cover and low bedrock permeability, groundwater recharge rates are considered unlikely to place a limitation on inflow the mine workings: flow will be limited by the hydraulic properties of the host rocks. At this stage in the mine development, it is considered appropriate to assess groundwater inflow based on an assumption of a constant near-surface water table.

There are, however, many details of the groundwater flow regime which require refinement:

- The hydraulic properties of the Saltvatn Group hosting the Ulveryggen Mine have not been characterised and may differ considerably from the Nussir Group which has been tested.
- Testing in the Nussir Group identified a range in permeability between different boreholes of at least two orders of magnitude, the cause of this variability and possible links to structural features cannot be refined without further investigation.
- Whilst this and all previous studies have identified that structural features are likely to dominate groundwater flow, this assumption and the locations of features which are significant, cannot be tested without both further spatially distributed groundwater monitoring to resolve the features of the groundwater flow regime, and a better understanding of the geological structure in the vicinity of the proposed mining locations.
- No information regarding groundwater quality is yet available for the project area, and further characterisation of groundwater quality and mine geochemistry is required to give robust predictions of future discharge quality for operational management.

The risks associates with the hydrogeological regime are generally assessed as low.

1.4.1.5 Implications for Mining

The section presents an outline of the issues to be considered during the ongoing design of the mine and development of a groundwater investigation programme, based on the information currently available.

Dewatering and Engineering

Dewatering of the ore body to allow working is the principal driver for water management in the mine with the main objectives of:

- Creating conditions appropriate for productive mining.
- Mitigating the risk of mud rush and water inrush.
- Reducing pore pressures to improve the geotechnical stability of the rock.
- Providing a source of water with the potential to supply part (or all) of the project's total water demand.

Based on the information available it is evident that the proposed mine workings will be some depth below the water table and there is a clear need to manage groundwater inflows during the mining operation. However, the status of the Nussir and Ulveryggen mines is very different, as the Nussir workings lie below the elevation of the portal of the haulage adit and will require active dewatering. In contrast, approximately 90% of the workings associated with the Ulveryggen Mine lie above the elevation of the existing haulage adit and will be passively drained.

Nussir Mine

Inflow rates for the Nussir workings have been estimated by Golder using the method of (Ref. 5-16 (Goodman et al (1965)) for calculation of steady state inflow to an underground tunnel. The estimated rate of inflow at each level, and for the total workings combined are indicated in Table 5-29. The predicted inflows over the mine life are shown graphically in Figure 5-59.

Table 1-29: Calculated Mine Inflow, Nussir Mine, Stopes and Adits

Operational Level	Contribution From	Total Inflow, Base Case [L/min]	Total Inflow, Sensitivity A [L/min]	Total Inflow, Sensitivity B [L/min]
Level 1 (230 mASL)	Haulage adit and Level 1	410	800	170
Level 2 (170 mASL)	Haulage adit and Level 2	430	850	180
Level 3 (125 mASL)	Haulage adit and Level 3	440	880	180
Level 4 (65 mASL)	Haulage adit and Level 4	490	960	200
Level 5 (20 mASL)	Haulage adit and Level 5	490	960	200
Level 6 (-40 mASL)	Haulage adit and Level 6	530	1050	220
Level 7 (-85 mASL)	Haulage adit and Level 7	520	1030	220
Level 8 (-145 mASL)	Haulage adit and Level 8	570	1130	240

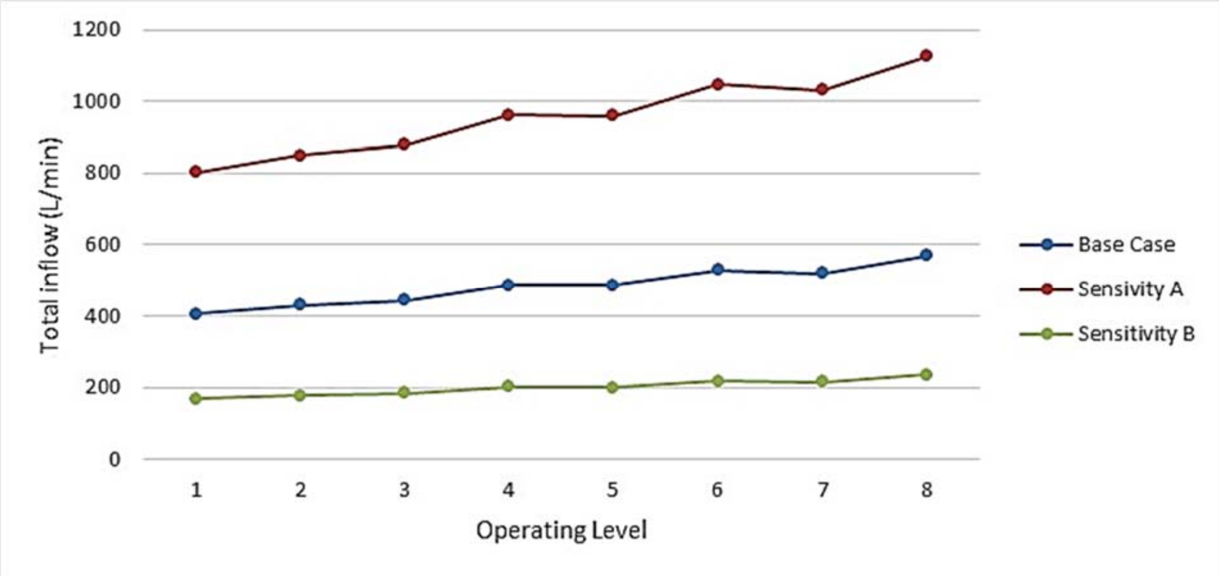

Figure 1-58: Calculated Groundwater Inflow to Nussir Mine

This calculation is subject to the following assumptions regarding the mine geometry:

- That 40 % the tunnel length intersects more permeable rock (hydraulic conductivity $1x10^{-8}$ m/s) and 60 % of the tunnel intersects less permeable rock (hydraulic conductivity $1x10^{-9}$ m/s) based on the variability encountered in hydraulic testing.
- During mining of single stopes a zone 40 m high and 5 m wide will be mined, the effective radius of single stopes post-mining is 8 m.
- During mining of single stopes a zone 40 m high and 20 m wide will be mined, the effective radius of double stopes is 16 m.
- The effective radius of other tunnels (access ramp, haulage adit) is 5 m.
- That the full length of the access ramp and haulage adit will be constructed prior to the start of stope mining.
- Dewatering associated with drainage of the haulage adit will depressurise the overlying zone such that the access ramp and sections within the radius of the ramp will make a minimal contribution to groundwater inflow.
- Sensitivity analysis calculations have been undertaken on the following basis:
  - Sensitivity Analysis A: 90 % of the rock encountered is in the higher permeability range.
  - Sensitivity Analysis B: 10 % of the rock encountered is in the higher permeability range.

The mine inflow is expected to increase progressively as the workings are developed, peaking during mining of the deepest stope (-145 m level). Inflow from the haulage adit is predicted to be by far the greatest contribution to the total inflow. Steady state inflow from the haulage adit alone is estimated to be between 165 L/min and 790 L/min with a most likely estimate of 400 L/min (24 m^3/hr). The peak inflow rates are calculated to be between 240 L/min and 1 130 L/min with a most likely estimate of 570 L/min (34 m^3/hr).

It is currently recommended that the Nussir mine workings are dewatered passively with water entering the mine being collected and pumped out to surface through pipe work installed either in the declines or haulage adit. Active dewatering by pre-drainage wells external to the mine working is to be considered only as a contingency in the event that inflows from identified highly conductive features become unmanageable. For the purpose of this project passive dewatering is defined as draining of water bearing formations/structures by drilling of small diameter drain holes out from the workings into the rock mass and then diverting the water into sumps for removal. The drains are to be drilled with standard mining equipment in drifts off the main working areas. Consideration should also be given to the use of grouting to seal any localised highly conductive zones encountered during mine development.

Ulveryggen Mine

The layout of the proposed Ulveryggen workings are illustrated in Figure 5-60 . The ore will be worked from nine distinct stopes. Of these stopes, eight are located at elevations above the existing haulage adit and will be connected by access tunnels to the haulage adit. Groundwater discharging from these stopes and tunnels will discharge passively to the haulage adit and from there to Aresbakti stream and Repparfjord. According to the surface water management strategy, discharge from the mine will be captured and piped to the processing plant at the point of exit from the haulage adit if the rate of discharge increases beyond current conditions in the future.

One of the stopes, the most easterly, is located below the elevation of the haulage adit. Active groundwater control is likely to be required in tunnels at the base of this area of working, with pumped discharge to the haulage adit. Engineered separation of flows in the haulage adit at the entrance of access tunnels to this area of working will be required to prevent flooding of this area of the mine by discharge from other mine areas.

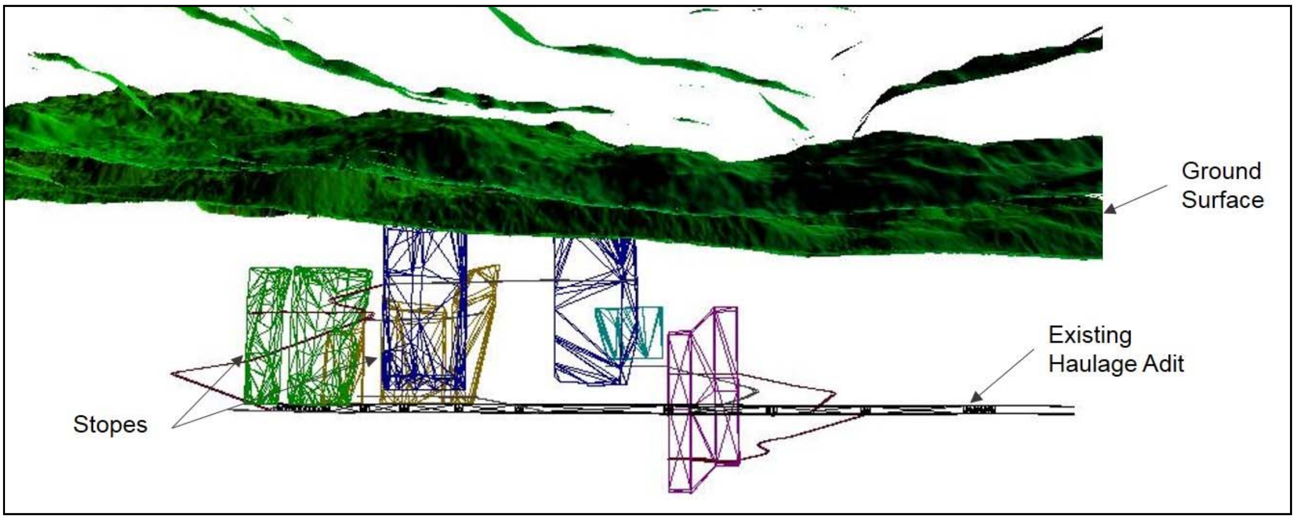

Figure 1-59: Ulveryggen Workings, Horizontal Section View from the South

As discharge from the Ulveryggen mine will be primarily passive, and the mine may also be largely under-drained by the existing haulage adit, no attempt has been made at the current stage of the mine development to quantitatively predict the future discharges from the Ulveryggen mine.

It is considered that the best indication of future groundwater flows from these working is given by discharge from the existing haulage adit. Limited data is available regarding flow in this adit. Ref. 5-14 NIVA 2011 indicates that flow in the Arebakti at the monitored location is 7 L/s and has a copper concentration of 118 µg/l, and also that the copper concentration at the point of discharge from the haulage adit is, on average, 470 µg/l. Based on the dilution ratio inferred by this data, the current discharge from the haulage adit is approximately 100 L/min (1.7 L/s). As both flow and concentration in the Aresbakti stream will fluctuate, a considerable error is associated with this estimate.

The degree to which discharge from the Ulveryggen Mine area will increase as a result of future development is a function of the extent of the existing drawdown associated with the historical haulage adit. Development in the area dewatered by the existing adit is unlikely to significantly increase flows. Development outside the existing cone of depression will increase the rate of discharge. Extension of the mine workings may also increase flow to the adit as the potential for intersection of larger flowing features (fractures or faults) is increased as the worked area extends. The effect of further development of the Ulveryggen Mine can be better estimated with improved characterisation of piezometric levels around the existing pits and haulage adit and in the proposed areas of working, and better characterisation of the location of faults and flowing features.

As for Nussir mine, it is currently recommended that the Ulveryggen Mine workings are dewatered passively. In most the workings, it is anticipated that passive drainage to the main haulage adit will occur without pumping. In the most eastern workings, water entering the mine should be collected and pumped out to surface through pipe work installed either in the declines or local haulage adits. Active dewatering by pre-drainage wells external to the mine working is to be considered only as a contingency in the event that inflows from identified highly conductive features become unmanageable. For the purpose of this project passive dewatering is defined as draining of water bearing formations/structures by drilling of small diameter drain holes out from the workings into the rock mass and then diverting the water into sumps for removal. The drains are to be drilled with standard mining equipment in drifts off the main working areas. Consideration should also be given to the use of grouting to seal any localised highly conductive zones encountered during mine development.

Operational Dewatering Methods in Both Mines

Currently there is no hydrogeological information for the mine development hence the following dewatering strategy is proposed to mitigate the risk of localised high inflow rates, particularly in areas of known faulting:

- During construction of the declines and levels it is proposed that probe holes are drilled ahead of face advancement.
- In the event that significant water inflows are encountered in probe holes, and if the rate of flow does not quickly reduce if the probe hole is allowed to flow freely, then two principal measures could be considered:
  - Drilling of drain holes angled out from the face in a fan pattern, to drain the rock around area to be mined. The rate of pressure reduction would be monitored in adjacent probe holes; and/or
  - The probe hole may be plugged and grout holes drilled out from the decline/level at cement-based grout injected to form a low permeability barrier around the decline/level during construction. Due to the likely small opening size of fissures in the rock there may be a requirement to use micro fine cement grouts rather than conventional Ordinary Portland Cement (OPC).

1.4.1.6 Conclusions and Recommendations

Recent hydrogeological testing has refined the understanding of hydrogeological conditions in the vicinity of the Nussir Project, confirming that volcanites of the Nussir Group are of low permeability associated with flow in the secondary fracture porosity and that groundwater is close to ground surface in the project area.

The overall features of the hydrogeological regime are relatively well constrained:

- Unconsolidated superficial deposits are absent across higher ground in the project area, with generally sporadic accumulation in valleys, groundwater flow in alluvial or superficial aquifers is unlikely to significantly influence the groundwater flow regime controlling mine inflow or dewatering.
- Groundwater is close to ground surface across the project area, locally discharging the north and south to lakes and streams and at a wider scale flowing north-east toward Repparfjord.
- The ancient rocks of the Repparfjord window are of low permeability with flow dominated by secondary fracture porosity; groundwater flow is likely to be dominated by geological structures. Given the understanding of the deformation history of the area, these structures are largely likely to promote groundwater flow to the north-east toward Repparfjord.
- Given the high rainfall, thin/absent superficial cover and low bedrock permeability, groundwater recharge rates are considered unlikely to place a limitation on inflow the mine workings: flow will be limited by the hydraulic properties of the host rocks. At this stage in the mine development, it is considered appropriate to assess groundwater inflow based on an assumption of a constant near-surface water table.

There are, however, some details of the groundwater flow regime which require refinement:

- The hydraulic properties of the Saltvatn Group hosting the Ulveryggen mine have not been characterised and may differ from the Nussir Group which has been tested.
- Testing in the Nussir Group identified a range in permeability between different boreholes of at least two orders of magnitude, the cause of this variability and possible links to structural features cannot be refined without further investigation.
- Whilst this and all previous studies have identified that structural features are likely to dominate groundwater flow, this assumption and the locations of features which are significant, cannot be tested without both further spatially distributed groundwater monitoring to resolve the features of the groundwater flow regime, and a better understanding of the geological structure in the vicinity of the proposed mining locations.
- No information regarding groundwater quality is yet available for the project area, and further evaluation of groundwater quality and mine geochemistry is required to give robust predictions of future discharge quality.

Preliminary calculation of inflow to the Nussir mine workings based on estimated bedrock permeability from testing in the Nussir Group indicate that for the entire mine workings, inflows may range between 40 m^3/hour and 190 m^3/hour, with a most likely value based on existing characterisation of around 100 m^3/hr.

It is currently recommended that the Nussir mine workings are dewatered passively with water entering the mine being collected and pumped out to surface through pipe work installed either in the declines or shafts. Currently there is no significant hydrogeological information for the mine development. Recommendations have been made in this report regarding a dewatering strategy to mitigate the risk of localised high inflow rates.

Preliminary geochemical studies have concluded that drainage from the mine is likely to be neutral with low copper concentrations, and can be suitably managed through good management practices and appropriate closure. However, copper concentration in the current discharge from the Ulveryggen pits is considerably in excess of the Norwegian MAC EQS as a result of historical activities (this discharge is known to the authorities and monitoring of the discharge is undertaken by the existing owners). Based on the existing information, it should be assumed that this quality is likely to be representative of future discharge from both Nussir and Ulveryggen mines. It is currently planned that during operations discharge from the Nussir mine, and discharges from Ulveryggen if they exceed current flows, will be directed to the processing plant and will not be discharged to the environment. Closure considerations may need to incorporate measures to manage the quality of mine water discharges, but these will also be guided by further studies of groundwater quality and mine geochemistry.

Currently the hydrogeology of the Nussir and Ulveryggen deposits is poorly characterised. Further investigation and monitoring is recommended to:

- Facilitate an improved understanding of the hydrogeological regime.
- Establish baseline hydrogeological conditions.
- Facilitate an assessment of the risk to mining from water inrush.
- Facilitate an assessment of the rate of inflow to the mine during its development.
- Allow an assessment of the impact of the underground mine on groundwater and surface water.

1.4.2 Mine Water Management

1.4.2.1 Dewatering

At Nussir, all drives are planned with a slight incline to drain water to the centrally located lowest point on each production level. From these points, water is passed through drilled drainage holes to the main level of the mining panel. A settling basin (clarification sump) will be located at the lowest part at L-150 of Panel 1 of the Nussir mine (Figure 5-61).

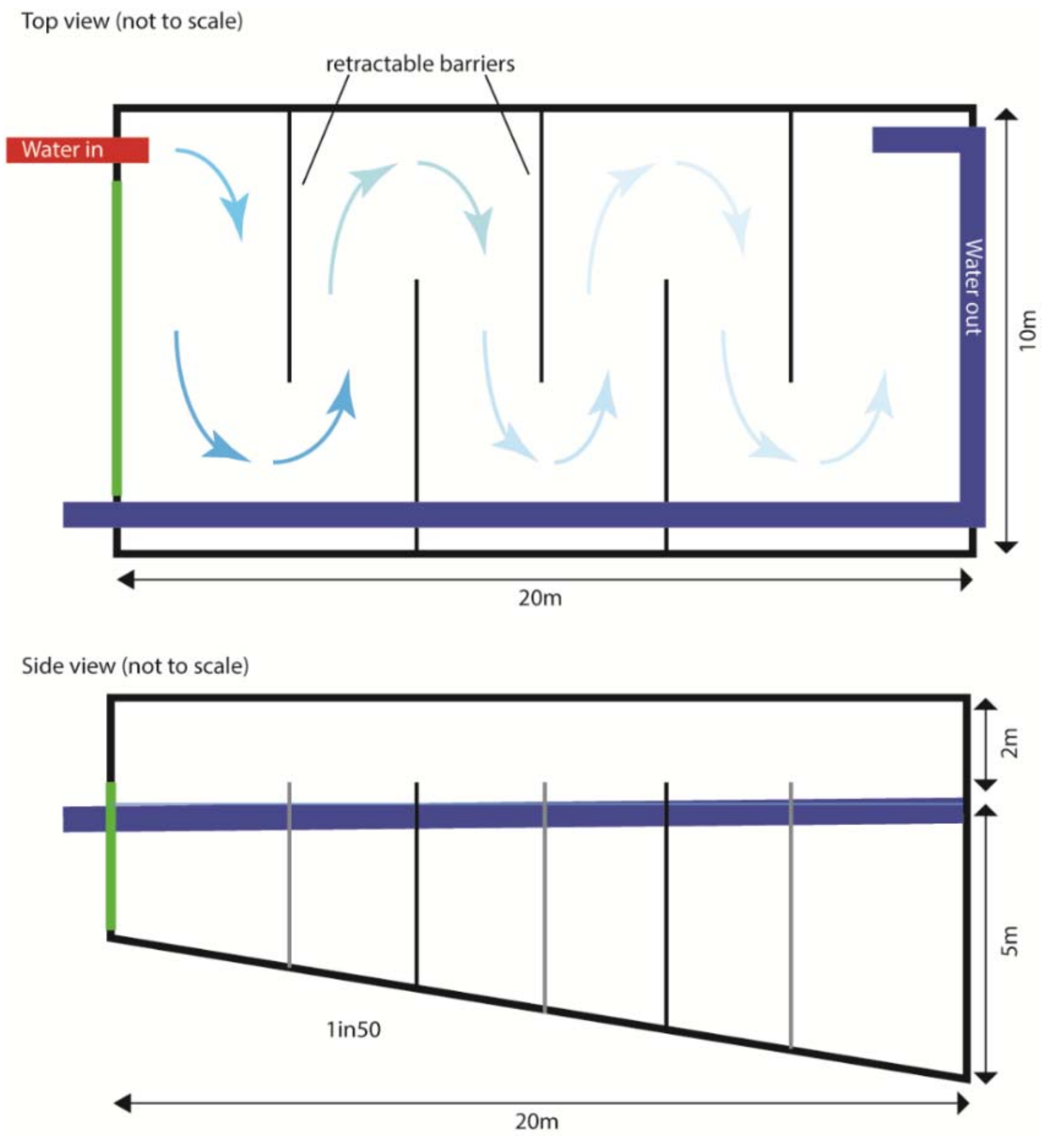

Figure 1-60: Sketch of clarification basin at L-150, Panel 1.

When the mine is extended to new production areas (panels 3, 2 and 0) towards the east and west separate settling basins will be installed to capture contact water from these areas. Within these basins a number of retractable walls will slow down the velocity of the water and all coarse particles will settle out. Clarified water is collected at the end of the basin by a weir with a pump box. It is expected that most of the particles settle in the first 1 to 2 “chambers” of the basin with finer particles settling in the chambers downstream. Barriers within the sump are removed to allow emptying of the fines, as is the main barrier at the front of the basin. Once pumped out and the barriers removed, the basin can be cleaned out using a small wheel loader or a load and haul machine.

Most of the fines are generated by drilling and washed out through stopes and muck piles, and can as such be characterized as ore. After being mucked out from the settling basin, the fines will be loaded on the conveyor belt and enter the ore handling cycle. It is important to avoid unnecessary turbidity; all fines and cuttings from production drilling should be collected and cleared away after each working day. If necessary, drainage holes into empty stopes below can be drilled to re-direct excess drill water. Ditches should also be established along all haulage routes.

To ensure that no hydrocarbons reach into the environment, equipment to soak up any spill form diesel or oil needs to be located nearby and have easy access in case of an accident.

Cleared water is collected at the far end of the basin and pumped up towards the surface in 160 mm (PE100) diameter pipes. These pump sumps will be concrete basins which will be equipped with two pumps, one operating and one standby. Pumps should be controlled using level monitoring and a simple float. Once at the surface the water will be fed into the water cleaning cycle at the processing plant and ultimately discharged into Repparfjorden.

Water handling requirements are estimated from Rana Gruber's own experience in dewatering, associated equipment for Nussir is summarized in Table 5-30.

A sewage tank / septic system will be installed outside the canteen for sewage disposal. The septic tank sludge will be removed by a vacuum truck at regular intervals.

Table 1-30: Specification and Bill of Quantities for Nussir Waste Water Handling

Item	Pcs / meter
Bilge pump (J405 HD 400V)	6 pcs
160 mm PE100 SDR 17 (12m)	2 500 m
Muff El. SDR 17 160	250 pcs
Valve (butterfly) DN150	6 pcs
Krave 160 mm SDR17 Lang DN150	8 pcs
Flange Galv PN10 DN15 0	6 pcs
Packer / Seals DN150	6 pcs
Equipment for welding of el. Muffs	1 pcs

At Ulveryggen a similar sump will be installed to clean waste water. However, most of the waste water can flow naturally through the existing ditches in the Ulveryggen tunnel. Only waste water from the extraction levels will have to be pumped to the main Ulveryggen transportation drive. The bill of quantities for the Ulveryggen waste water handling is summarized in Table 5-31.

Table 1-31: Specification and Bill of Quantities for Waste Water Handling at Ulveryggen

Item	Units / meter
Bilge pump (J405 HD 400V)	2
160 mm PE100 SDR 17 (12m)	2 000
Muff El. SDR 17 160	200 pcs
Valve (butterfly) DN150	2 pcs
Krave 160 mm SDR17 Lang DN150	4 pcs
Flange Galv PN10 DN15 0	2 pcs
Packer / Seals DN150	2 pcs

1.4.2.2 Fresh Water

Fresh water for use in the canteen, workshop and the drilling operations will be delivered through pipes (160mm PE100) from the main entrance of the mines. From the lowest point in the mine water will pass through several pumping stations to the active mining areas. An estimated bill of quantities for Nussir Panel 1 and Ulveryggen is summarized in Table 5-32 and Table 5-33. The Panel 1 water supply and pump station will be duplicated at each of the other panels for costing purposes. More detailed engineering and design will be completed in future studies.

If 14 bar water pressure is achieved at the main level L-150 at Nussir, no pumping will be needed to the first 2 drill levels. From there a pump will deliver water further up in the mine through pipes (110 mm PE100).

Table 1-32: Pumps and Pipes Bill of Quantities for Nussir Fresh Water, Panel 1

Item	Pcs / meter
Pump; Caprari HVU50-5N+60	1
160 mm PE100 SDR17 (12m)	3 000 m
Tapping sleeves M/Kniv 160/2''	30 pcs
Reduction valve	2 pcs
Muff El. SDR 17 160	250 pcs
T-pipe Red. Pe 160x110	3 pcs
Valve (Butterfly) DN150	3 pcs
Collar 160mm SDR17 Lang Dn150	18 pcs
Flange Galv PN10 DN150	18 pcs
Packer / Seals	18 pcs
110mm PE100 SDR17 (12m)	3000 m
Tapping sleeves M/KNIV 110/2''	30 pcs

Table 1-33: Ulveryggen Fresh Water Handling Bill of QuantitiesItem	**Pcs / meter**
Pump; Caprari HVU50-5N+60	2
160 mm PE100 SDR17 (12m)	2 400 m
Tapping sleeves M/Kniv 160/2''	24 pcs
Muff El. SDR 17 160	250 pcs
T-pipe Red. Pe 160x110	5 pcs
Valve (Butterfly) DN150	3 pcs
Collar 160mm SDR17 Lang Dn150	18 pcs
Flange Galv PN10 DN150	18 pcs
Packer / Seals	18 pcs
110mm PE100 SDR17 (12m)	2000 m
Tapping sleeves M/KNIV 110/2''	20 pcs

1.5 Discussion - Mine Design and Production Schedule

Previous studies of the Nussir Project indicated that production from the Nussir (and Ulveryggen) deposits could start within the first project year. The previous studies were not completed to the same level of design and engineering analysis as a PFS, and as such many of the productivity, design and scheduling constraints defined in this study were not defined to the same level of detail. This is not a criticism of previous work but a realistic view of how the level of detail and additional cost estimating and engineering identifies constraints and limitations to the mining and process plant. This issue is coupled with the time required to construct and commission the process plant, which is expected in Year 3 (2020) of the PFS plan. Any production before the commission of the process plant will have to be stockpiled, with the risk of oxidising the initial mine production whilst it awaits processing.

The ore release schedule and the production rates were designed to reflect the timing of the availability of the processing plant and the time required to develop the mine access tunnels, install and commission infrastructure and develop the two levels required to start mine production. Developing the extraction and drill levels requires time, and the mining method and design require that the levels are developed to their full extents before production can commence.

The mine development and production schedules completed for this study were focussed on maximising resource extraction and were not optimised to maximise ore production from any given area or combination of areas that comprise the Nussir Project. Further engineering productivity and efficiency analyses, industrial engineering, scheduling combined with mine design, resource modelling and simulation are required to be complete in the period between the effective date of this PFS and detailed production planning after the completion of a positive feasibility study.

Further industrial engineering analyses will also identify optimal equipment specifications, capacities, and fleet size numbers. There have been recent advances in engine and power train technologies which may make electric or hybrid mobile plants feasible, which will impact on the mine ventilation requirements. The increasing application and adaptation of automation and autonomous vehicle operation will reduce both the workforce underground and the total number of workers in the mine.

The next study of the Nussir Project is proposed to be a feasibility study. The issues discussed above will all be investigated and re-examined during that study. The mine design and scheduling will be based on updated Mineral Resources, geotechnical and hydrogeological sampling and testwork and other sub-studies which will increase the confidence in the design basis used in the feasibility analyses. This study will further refine both the engineering inputs to the design but also the mine CAPEX and OPEX for the project.

Additional drilling and resource definition will upgrade the Inferred Resources to Indicated or Measured categories which will also inform and affect the strategic planning for developing and mining the Nussir and Ulveryggen deposits.

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Oversettelse av sammendrag fra Vedlegg 7b om gruvedrift

Nussir ASA, 10. mai 2017

Tilnærming og metode

Topografien, strøklengden og dybden av Nussir-malmen gjør leteboring med tilstrekkelig tetthet (for å sikre at alle de foreslåtte gruveområdene er til minst en indikert grad av tillit) umulig for Nussir å ha fullført mellom Preliminær økonomisk vurdering (PEA), fullført i 2014, og ønsket om å fullføre PFS i 2016. Leseren er derfor advart at selv om mineralressursene har blitt generert til en standard som er i samsvar med internasjonale rapporteringsstandarder, har gruveplanen og tidsplanen inkludert Inferred materiale som også er inkludert i Base Case prosjektets kontantstrøm.

Fordelingen av de utvannede ressursene i Nussir orebody er **18,4 Mt @ 1,03% Cu, 9,68 g / t Ag og 0,11 g / t Au**, med indikerte ressurser som omfatter ca. 48% av Nussir-gruveplanen.

Ulveryggen-malmen ble inkludert i gruveplanen med produksjon mellom år 1 og 7 i tidsplanen. Ulveryggen-malmen er tilgjengelig nesten helt fra starten av prosjektet. Delene av Ulveryggen malmen som inngår i gruveplanen utgjør 2,5 Mt @ 0,94% Cu.

Lokale myndigheter, regler og tillatelser har pålagt begrensninger på overflateforstyrrelser som gjør dagbruddsdrift av Nussir-malmen uaktuelt nå. Som et resultat, ble dagbruddsdrift ikke vurdert i denne studien.

Ulveryggen-forekomsten består av flere mindre malmkropper, og en felles parameter for disse malmkroppene er at de blir mindre med økende dybde.

Geoteknisk analyse

Bergegenskapstestene av Nussir og Ulveryggen malmene indikerer at hengvegg, liggvegg og malm er relativt sterke og intakte steinmasser. Regionale spenningsmålinger gjennomført både på stedet og i Finnmarkregionen har fastslått at hovedspenning er omtrent 20 MPa og er justert omtrent parallelt med strøk for Nussir-malmen. Hoved-spenningen er mindre enn om lag en tredjedel av UCS for Nussir-bergarter, og bare ca. 10% for Ulveryggen-prøvene som er testet. Ytterligere arbeid bør utføres for å vurdere bergmassen, særlig testing som tilfredsstiller ISRM Standard, men de første indikasjonene er at den intakte steinstyrken er mye høyere enn in-situ spenninger.

Spenninger som følge av gruvedrift i nærheten av de foreslåtte gruveområdene vil bli modellert som en del av Feasibility studien for å vurdere risikoen for gruvedrift og gruvestabilitet basert på gruvegeometri, stollsekvensen og uttakshastigheter.

Gruvedesign for Nussir

Gruvedriftmetoden for Nussir-malmen er valgt basert på en komparativ studie, skivepallbrytning (Sub-level Open Stoping), som det er lett tilpasset til varierende bredder og helning av strosser (Stope) og mineralisering. I tillegg er utstyret bevist og kan brukes til automatisering og fjernoperasjon.

Å finne en stabil, men effektiv strosse-størrelse, er viktig når du implementerer skivepallbrytning. Bredden på strossene bestemmes av tykkelsen av malmen, som i dette tilfellet varierer mellom ca. 3 til 5 m. Der malmen er smalere enn 3 m vil strossene ikke være økonomisk levedyktig å utvinne, derfor er hovedparameter for dimensjonering av strosser deres horisontale lengde langs strøk og vertikale høyder. Den vertikale høyden er også begrenset på grunn av maksimal borlengde og ønsket om å minimere antall bormeter. Vertikale og horisontale pilarer innføres mellom strossene for å forbedre stabilitet. Innføring av punktpilarer kan vurderes når det er nødvendig for å unngå at det er behov for kabelbolting av hengveggen.

Etter empirisk analyse og 2D numerisk modellering av de geotekniske aspektene ved gruven design kan følgende observasjoner gjøres:

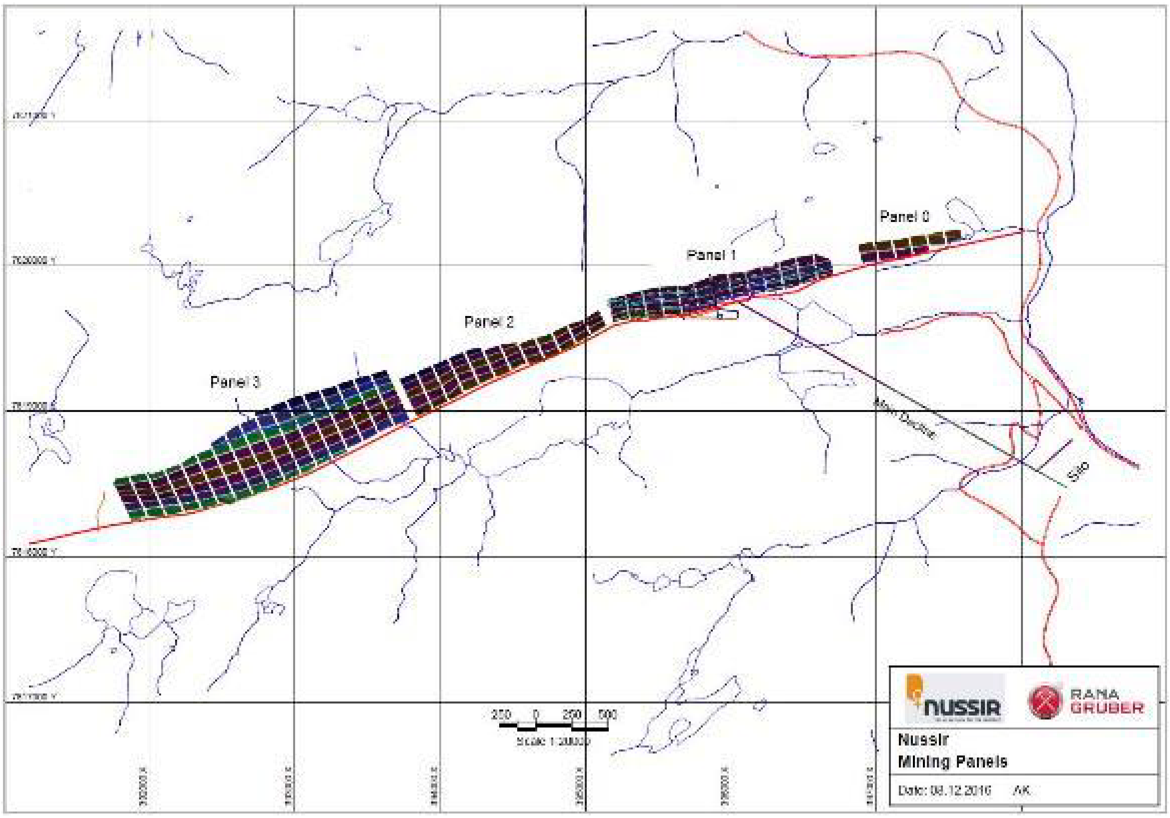
- Flertallet av testene gjort på forekomst, ligg- og hengvegg viser verdier som varierte mellom rimelig (fair) til veldig god, som indikerer at skivepall og varianter av denne metoden er mulige, i hvert fall for panel 1-området;
- Den nåværende strosse-geometrien på 90 m høy og 100 m lang, langs strøk, kan være ustabil basert på en kort analyse ved hjelp av «Mathews Stability Graph»-metoden for stoppdesign og plotting av produksjonen Isoprobability konturer. Stoppestabiliteten er ikke spesielt følsom for tykkelsen på forekomsten;
- Bruken av vertikal (sill) (og horisontal (rib)) pilarer kan være en kilde til langvarig ustabilitet i gruven. Vertikal-støtte-stabilitet var ikke analysert; og
- Tensile svikt på strosskrone (stope crown) og i hengveggen er et vanlig problem for en skive pall metoden. Installere egnet forsterkning, for eksempel kabelbolter, i strosskrone og hangingwall ville forbedre stabiliteten på stoppene på lengre sikt.

Som en del av feasibility-studien anbefales det at følgende gjennomføres (for både Nussir og Ulveryggen):

- 3D modellering for å ta hensyn til hovedspenningen (vinkelrett på modellplanet), spenningsfordelingen i Vertikale søyler (i tillegg til sillstolper) og den globale stabiliteten langs stoplengden;
- Alle fremtidige borekjerner logges og registreres ved hjelp av Q loggingssystemet introdusert i 2016;
- Borkjerner i hengvegg, liggvegg og i forekomsten er kartlagt med samme Q-system for å øke den geotekniske databasen og gi orienterte strukturdata for å informere stabilitetsgrafen metode;
- En optimalisering av strossedimensjonene og dermed delnivåintervallet utføres som balanserer stopp stabilitet med antall gruvenivåer som kreves innenfor hver av panelene;
- Gruvedesignet, strosse-dimensjoner og søyleposisjoner og dimensjoner er bekreftet med ytterligere 2D og 3D modellering;
- Fremtidig 2D- og 3D-modellering skal fylles ut med hensyn til gruvefølgen og passende modellfaser for å reflektere utvinningsfaser av stoppene;
- En 2D-analyse av et vanlig utvalg av stopper og søyler ble fullført basert på stoppdesignet til Panel 1. Et design beregnet på å maksimere malmekstraksjonen bør også analyseres, som kan ha variabel Stopp streik lengder og ribbe søyle bredder;
- Offsetavstanden fra fotvoksen til Nussir forekomsten til den permanente gruveinfrastrukturen (passerer, Utvinningsstasjoner og ramper) bestemmes ved hjelp av en stressmodelløvelse;

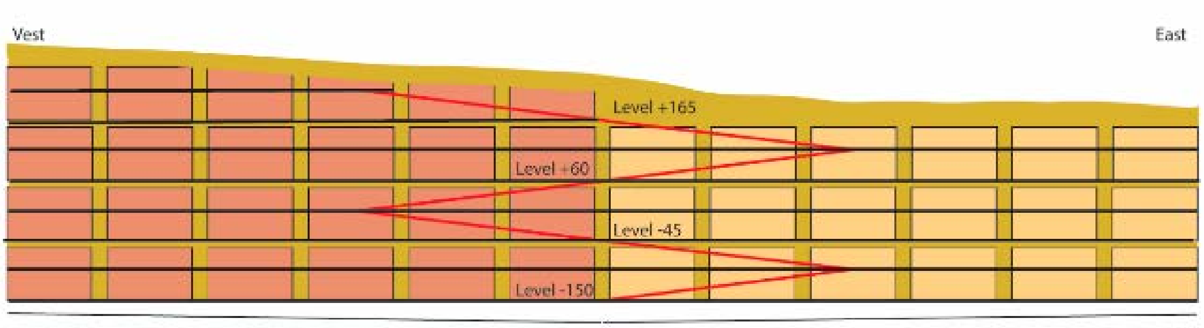
- Stabilitetsgrafer bør oppdateres regelmessig for å sikre at mine- og stoppdesign er konsistente med den siste geomekaniske forståelsen av innskuddet;
- En probabilistisk tilnærming til modellering og geomekanisk utforming av panelene utenfor det innledende 5 år gruveperioden er tilpasset og brukes til å informere min planlegging og design; og,
- Størrelsen og de relative plasseringene av nøkkelbasert infrastruktur (verksteder, kantine, pumpestasjoner, og sumper) er bekreftet gjennom modellering.

Nussir-forekomsten ble deretter delt inn i 4 paneler nummerert 0 til 3 fra øst til vest (Figur ES-8). Disse panelene ble så videre delt inn i nivåer og stopper basert på undernivåintervallet og strossestørrelse definert av foreløpig elastisk og lineær elastisk 2D-bergmekanikkmodellering og empirisk design. Den primære gruveadgangen, en tunnel, ble designet fra Øyen-prosessanlegget til midtpunktet av panel 1. Panel 1 ble deretter utformet til et detaljnivå som var i samsvar med en PFS. De resterende panelene var ikke utformet, men faktorer og utviklingslengder ble skalert fra stoplayoutene og ved hjelp av Panel 1 design som grunnlag.



Figur ES-8: Illustrasjon av plasseringen av Nussirs gruve-paneler og deres stoller (Panel 0, 1,2 og 3).

Gruven vil bli utviklet fra øst til vest fra og med Panel 1 i øst. Hovedinngangen til gruven vil ligge på industriområdet nær fjorden, som også huser prosessanlegget, kontoret bygninger og kaserner. En hovednedgang på ca. 2,6 km lang med en maksimal gradient på -1: 10 vil føre til hovednivået (L-150) som slutter i sentrum av gruvepanelet 1. Et skjematisk diagram som illustrerer oppsettet av panel 1 presenteres i figur ES-9.

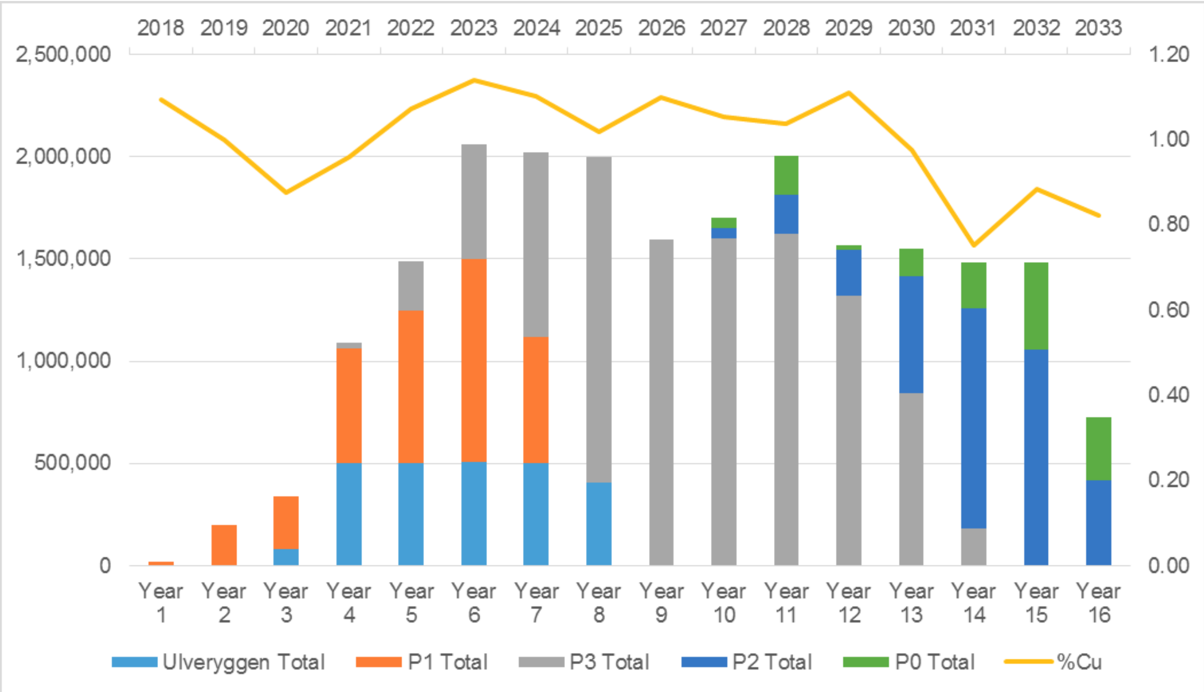


Figur ES-9: Skjematisk diagram av panel 1 som en projeksjon sett fra fotvognen (ikke i skala)

Gruveplan (Mining Schedule)

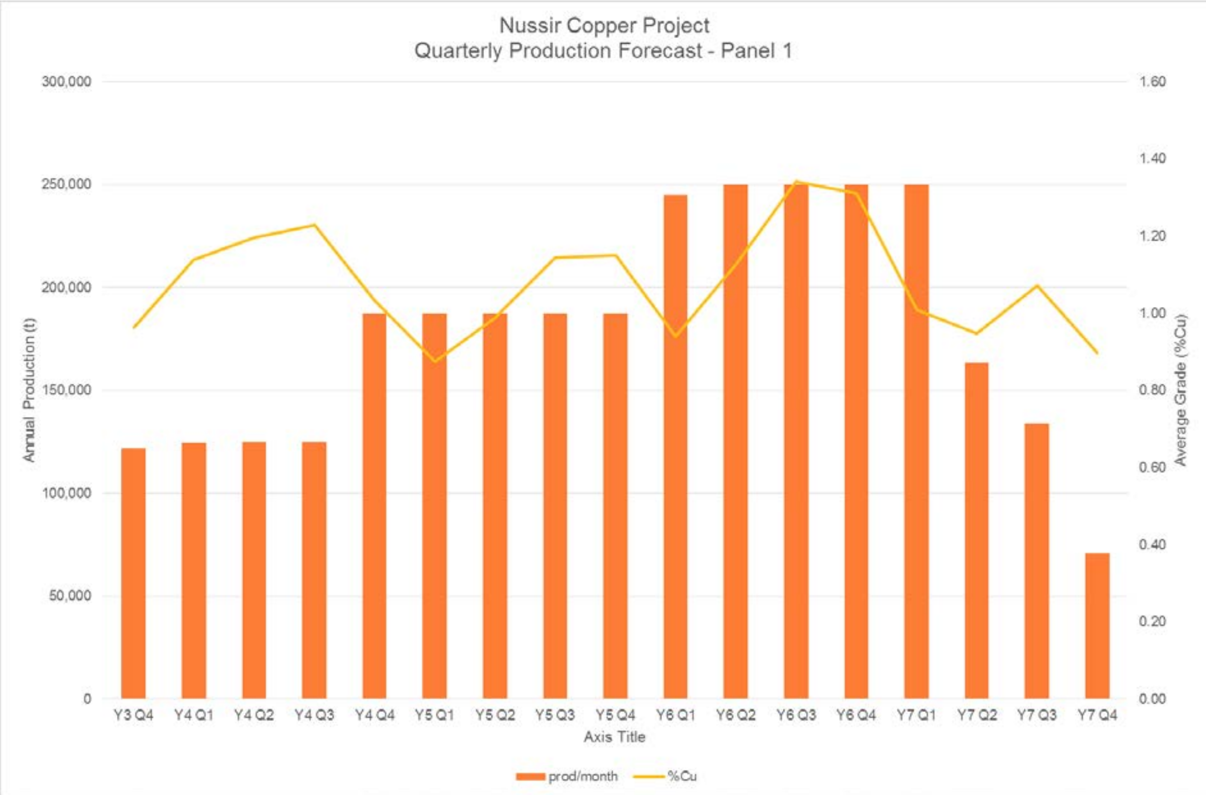
Gruveplanen er basert på produksjon som starter så snart som det første borenivået og dets tilknyttede stoller er utviklet. Gjennom hele gruvas levetid vil det være minst 2 strosser i drift samtidig.

En detaljert gruveplan (produksjonsplan), sammen med Ulveryggens produksjonsplan, presenteres i figur ES-10.



Figur ES-10: Produksjonsprognose basert på paneler i drift.

Den forventede Cu-innholdet i malmen som skal tas ut fra stollene i Panel 1 (Nussir malmen) varierer fra 0,804% til 1,551% Cu (Figur ES-11).



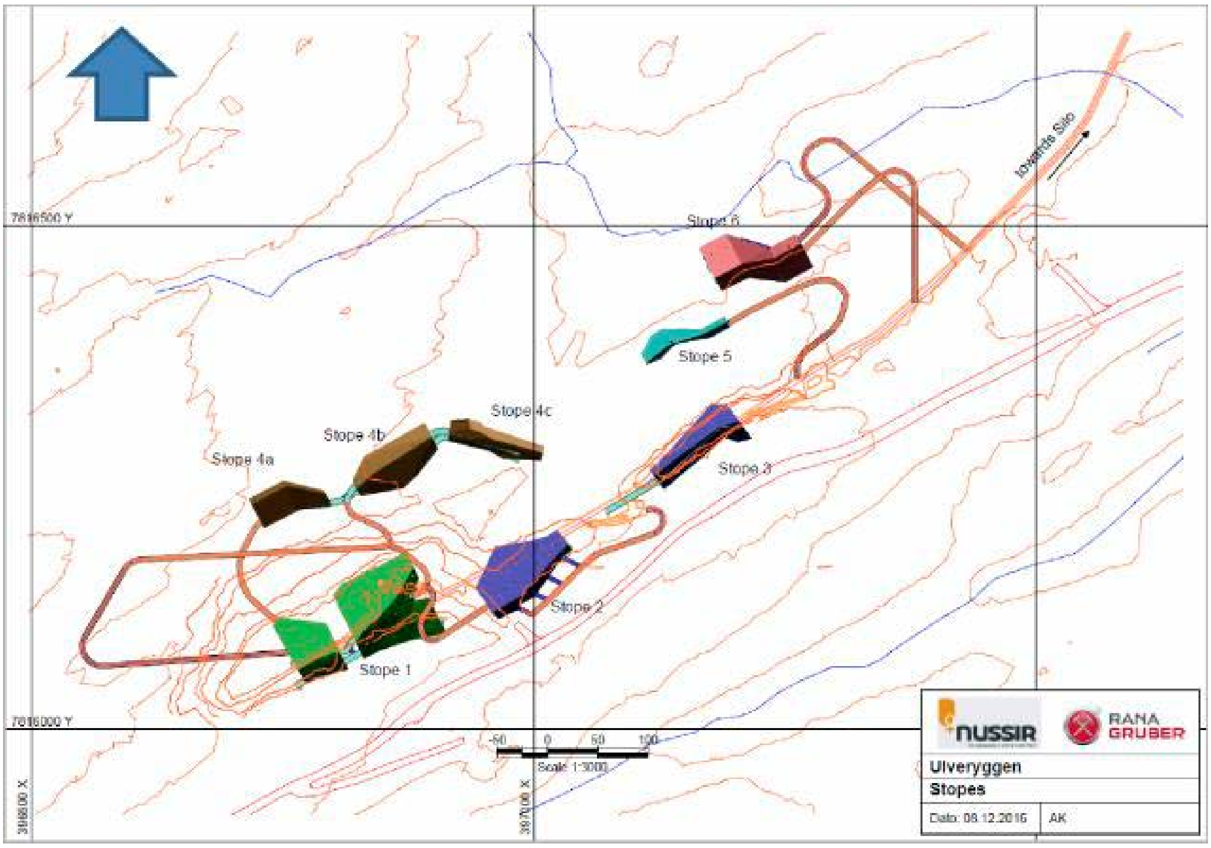
Figur ES-11: Forventet kvartalsvis kobbergrad, Nussir Panel 1

Gruvedesign for Ulveryggen

Ulveryggen sin mineralisering varierer i både størrelse, form og gehalt i forhold til Nussir. Mens Nussir er flere kilometer lang, kontinuerlig og ganske smal, består Ulveryggen forekomsten av flere distinkte kortere tykke kropper som følger strøket. En vanlig egenskap for kroppene er at de bli mindre ved økende dybde. Det har tidligere blitt drevet gruvedrift på Ulveryggen, ved flere mindre dagbrudd. Det finnes en 2,4 km lang tunell under Ulveryggen. Det eksisterer to malmsjakter og en ventilasjonsjakt som ble brukt til transport av malm til prosessanlegget på Øyen under åpning av dagbruddene på 1970-tallet. Disse underjordiske arbeidene vil bli brukt i nye gruveaktiviteter på Ulveryggen.

Underjordsdrift på Ulveryggen er planlagt å være ved Skivepallbrytning med 6 strosser. Stross-dimensjonene vil være ca 90 m høy x 100 m lange, men lengden på stollene vil endres for å imøtekomme variasjonen i bredden av malmkroppen. Strossene vil bli utviklet med et ekstraksjons- / borenivå og et sekundært borenivå 60 m over utvinningsnivået. Utmating vil bli utviklet fra en transportkjøring på utvinningsnivået i boreanlegget rett under strossen.

En generell utforming av foreslåtte Ulveryggen strosser og utvikling er presentert i Figur ES-12.



Figur ES-12: Ulveryggen stoller (Nummerert 1 til 6 fra vest til øst)

Ventilasjon

Primærventilasjon til Nussir-gruvepanel 1 vil bli levert av en ventilasjonsjakt på 3,5 m diameter. Vifter med lyddempere vil bli montert på toppen av sjakten. Lyddemperen reduserer støynivået til viftene til 69 db. Viftene vil bli utstyrt med en elektrisk oppvarming for å varme luften om vinteren. Avgassluften vil følg rampen til det øverste produksjonsnivået. Derfra vil en eksosjakt bli konstruert som kobles til overflaten. Sekundær ventilasjon består av vifter på hver ekstraksjon og bornivå og fleksible ventilasjonskanaler som leverer frisk luft til de aktive produksjonsområdene. Under utviklingsfasen, spesielt når hovedtunnelen er konstruert, vil frisk luft bli levert fra hovedinngangen til gruven. Luft vil bli presset av en vifte til utgangen av tilgangsstasjonen gjennom fleksible kanaler med avtrekksluft som følger hellingen til inngangen til gruven.

Hydrogeologi - gruvevann

Hydrogeologisk testing har forbedret forståelsen av hydrogeologiske forhold i nærheten av Nussir prosjektet, som bekrefter at vulkanittene i Nussir-gruppen har lav permeabilitet, og at grunnvannet er nært til bakken i prosjektområdet.

De generelle egenskapene til det hydrogeologiske regimet er relativt godt begrenset:

- Ikke-konsoliderte overflatiske innskudd er fraværende over høyere bakken i Prosjektområdet;
- Grunnvann ligger nær bakken over prosjektområdet, lokalt utladning i nord og sør til innsjøer og bekker og i bredere grad som strømmer nord-øst mot Repparfjord;
- De gamle klippene i Repparfjord-vinduet har lav permeabilitet med strøm dominert av sekundær bruddporøsitet; Grunnvannstrømmen vil trolig bli dominert av geologiske

strukturer. Gitt forståelse av deformasjonshistorien til området, er disse strukturene i stor grad sannsynlig å fremme grunnvannstrømmen i nordøst mot Repparfjord; og

- Gitt høye nedbørsmengder, tynn / mangelaktig overfladisk dekning og lav gjennomtrengelighet, grunnvannsgjennfylling (recharge) anses ikke som sannsynlig å legge en begrensning på innstrømning av gruvearbeidene: innstrømningen vil bli begrenset av hydrauliske egenskaper av verten bergarter. På dette stadiet av gruveplanleggingen anses det som passende å vurdere grunnvannsinnstrømning basert på en antagelse om et konstant nært vannbord.

For tiden er det begrenset hydrogeologisk informasjon for området umiddelbart rundt gruvedriften. Foreløpige beregninger av innstrømning til Nussir-gruvearbeidet basert på estimert grunnlags permeabilitet fra testing i Nussir-gruppen indikerer at for hele gruvearbeidet kan innstrømningene variere mellom 40 m3 / time og 190 m3 / time, med en sannsynlig verdi basert på eksisterende karakterisering på rundt 100 m3 / time.

Det anbefales for tiden at Nussir-gruvearbeidene dehydreres passivt med vann som kommer inn i gruva blir samlet inn og pumpet ut til overflate gjennom rørarbeid installert enten i avtak eller sjakter.

Foreløpige geokjemiske studier har konkludert med at drenering fra Nussir-gruven vil være nøytral med lave kobberkonsentrasjoner, og kan styres hensiktsmessig gjennom god administrasjonspraksis og hensiktsmessig nedleggelse. Det er for tiden planlagt at i løpet av driften blir utslipp fra Nussir og utslipp fra Ulveryggen, hvis de overskrider strømmen, vil bli sendt til prosessanlegget og vil ikke gi utslipp til omgivelsene. Ved nedleggelse kan en trenge å innlemme tiltak for å håndtere kvaliteten på utslipp til grunnvannet, men disse vil også bli styrt av videre studier av grunnvannskvalitet og gruvegeokjemi under feasibility studiet.