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Assessment of undergorund mining of Nussir copper deposit

With special emphasize on cut-off and mining method

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Abstract

The narrow steeply dipping strata bound sediment hosted copper deposit, Nussir, located in Kvalsund community, Finmark, Norway, have been subject to an assessment of underground mining. Resources classified as Indicated, within a 0,9%Cu cut-off limit have been identified as suitable for sublevel open stoping mining method, accessed by 590m long tunnel from fjord, a 2000m haulage tunnel in footwall progressing westward along strike and two individual ramps separated by 1280m along strike. Stopes are dimensioned 40m long and 3m wide on average with 62m and 102m height, with two and three sublevel drifts respectively. The stoping layout's stability rely on 5m rib pillars, 8m sill pillars and yielding pillars in stope centre, calculated from geotechnical data and empirical design methods. The retreating mining method implies narrow ore mining practices, utilizing small mechanized equipment, developing drifts along strike in two directions from ramp. Stopes are planned produced by up-and-downward drilling of 20m holes carefully planned within the narrow mineralisation, blasted carefully to avoid dilution from stope walls. Dilution causing head grade drop and obtaining of available treatment capacity, must be expected and dealt with. Cable bolting of the on average 60^{o} dipping hangwall from drifts, is necessary to avoid caving. Dillution, estimated as dependent on mineralisation thickness, have been added to the stope resources obtained by wireframe evaluation of block model in DataMine. A planned production schedule of development tunnels and stoping, producing 3Mt in 6 years time, is the basis for a financial model with 245million NOK in capital expenditures and 177NOK/tonne in mining costs. The planned mining operation gives a Net Present value of 32million NOK. Project value is considered highly sensitive to changes in copper price, mining costs and copper grade.

Sammendrag

Den smale, steilstående og lagbundne kobberforekomsten Nussir, i Kvalsund kommune, i Finnmark, har vært gjenstand for en vurdering av underjordsdrift. Ressurser klassifisert som indikert, innenfor en cut-off grense på 0,9% Cu er valgt ut for skivepallbrytning. Adkomsten sikres med en 590m lang tunnel fra fjorden i øst, en 2000m lang transportort i liggveggen som går vestover langs strøket og to ramper med avstand 1280m langs strøket. Strossene er dimensjonert med en lengde på 40m og bredde på 3m i gjennomsnitt. Høyden er 62m og 102m, med henholdsvis to og tre ortnivåer. Stabiliteten til strossene sikres med 5m ribbepilarer, 8m horisontalpilarer og ettergivende pilarer i sentrum av hver strosse. Pilarene er beregnet ut fra geotekniske data og empiriske designmetoder. Den smale malmen og valgt brytningsmetode forutsetter at det tas i bruk metoder for smalmalmsbrytning; bruk av små, smale maskiner, der orter drives langs strøket i to retninger fra rampen. Strossene er planlagt produsert med langhullsboring både oppover og nedover med 20m hullengde. Hullene må være nøyaktig boret innenfor den smale mineraliseringen og sprengt forsiktig for å unngå gråbergsinnblanding inn i strossa fra veggene. Gråbergsinnblanding vil forårsake fortynning av produserte gehalter og ta opp tilgjengelig produksjonskapasitet. Dette må forventes og håndteres. Hengveggen som gjennomsnittlig faller 60°, må kabelboltes systematisk fra orter, for å redusere innrasing. Gråbergsinnblanding, estimert som avhengig av mineraliseringens tykkelse, har blitt lagt til tonnasjen og verdiene i hver strosse, ved hjelp av wireframeevaluering av blokkmodellen i DataMine. Den planlagte produksjonsplanen med oppfaring og strossing som produserer totalt 3 Mt i løpet av 6 år, er grunnlaget for en økonomisk modell med 245 millioner NOK i kapitalkostnader og 177NOK/tonn i driftskostnader. Den planlagte gruvedriften gir en netto nåverdi av 32million NOK. Prosjektsverdi er ansett som svært følsom for endringer i kobberpris, driftskostnader og kobbergehalt.

Forord

Denne diplomoppgaven er avlagt ved NTNU masterstudiet i teknologi, som det så fint heter etter at de byttet ut det klingende, men noe utdaterte begrepet *sivilingeniørstudie*. Sivilist er jeg da fortsatt, men *bergmand* er vel egentlig mitt nye liv og virke, ettersom oppgaven du nå holder, er avlagt ved institutt for geologi og bergteknikk.
Glad i stein? Ja det kan du trygt si, men helst en digital en, som jeg kan snurre, løfte, og endre farge på med et tastetrykk! En stein som kan måles og veies, flyttes og dreies, på et blunk. Forvaltning er stikkordet, og tegning er verktøyet. Alle steiner kan tegnes, gråstein og gull, kalkstein og kull. Favoritten er derimot sedimentære kobberforekomster, som jeg liker å farge lyseblå, som fargen på babyklær eller himmelen en vakker sommerdag. Tegnede steiner gir uante muligheter! Et bilde av det ukjente, en form, et volum, et innhold med varierende kobbergehalter som hjelper oss å bestemme hva vi skal gjøre med fjellet under oss. Forvaltning av mineralske ressurser, på en bærekraftig måte, der man estimerer verdier, ikke med silkehansker og pinsett, men med Krig! Ja, Krige var hans navn. Mannen fra Afrika som statistisk beregnet hvor mye gull som lå i Witwatersrand til alles begeistring. Ja følelsene er sterke blant dem som har mineralressurser tett på kroppen. Naboer klager sin store nød, gruveslusker feller gledestårer når godene kan høstes etter strevet og geologene sperrer opp øynene når nye kjerneprøver kommer rykende ferske opp fra undergrunnen. For det er dem som veileder min tegning av steinene. Steinene som jeg planlegger å grave ut med bor, emusjonsprengstoff, last og bær maskiner og lastebiler som farter rundt inni fjellet på jakt etter malm. Steiner med mineraler over cut-off grensen, slik at det gir profitt å grave dem ut. Gjennom strosser og orter, som jeg tegner inn i fjellet på min datamaskin!

Det har vært en ære å jobbe med Nussir forekomstens gruvedesign og bli tatt inn i varmen av CEO Øystein Rushfeldt, Geolog Magne Martinsen, Gruveingeniørene Per Smeberg og Ola Eriksen som har kommet med ideer og tanker. Også stor takk til smalmalmsspesialisten Pekka Läppalainen og Jouni Hansen-Haug , sistnevnte som stolt viste meg rundt i Zinkgruvan’s smale orter. Ikke minst takk til ekspertene på NTNU, Siv Krane Rodahl og Ida Berg som har deltatt i gode faglige gruvediskusjoner. Og selvsagt veilederen min, Steinar Løve Ellefmo, som annenhver uke dette året har fulgt opp arbeidet.

Audun Mortveit Sletten   Trondheim 18.6.2012

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1. Introduction

1.1 About Nussir

Information below is taken from the NTNU report "Resource evaluation of Nussir copper deposit"

Nussir I is a large late-exploration stage copper deposit with silver, gold, palladium and platinum as by-products, and covers the so far partially explored part of the Nussir ore deposit for which mining rights have been secured. The deposit is located in the north of Norway in the community of Kvalsund, south of Hammerfest.

Early phase exploration drilling of Nussir began in 1985 as geologist Kjell Nilsen, then working for A/S Sydvaranger, by curiosity visited the area some years before. Copper mining was done from the nearby Ulveryggen deposit by Folldal Verk A/S from 1972 to 1978, leaving some useful infrastructure for future mining operations.
Scoping study of 2009, describe the Nussir Cu-Au-Ag orebody as of sedimentary origin with folding and redistribution of the metals. Copper mineralisation is indicated in the Nussir I deposit to occur in a single structure with a known strike length of over a 9 km along a steeply-dipping dolomite horizon. Mineralisation consists mainly of chalcopyrite and bornite, with silver and gold in a horizon of varying widths from 2 to 5 m grading 1 to 3% copper, with 18,4g/t silver and 0,13g/t gold on average (Golder, 2009).

The topography of the area is a fjord with a narrow strip of land on its bank before rising to higher ground that is used for reindeer grazing. Environmental concerns limit mining operations to underground only.

1.2 Current status

Current mineral right's owner, junior prospecting company Nussir ASA, with CEO Øystein Rushfeldt, have been developing the prospect since 2005. Scoping study was carried out in 2009 and the resource estimation was updated in may 2012, by 6 additional sample intersections drilled in 2011. Permission to extract minerals are granted from the Norwegian directorate of mineral resources, application for tailings disposal in the fjord and impact assessment report is approved by local authorities, waiting for final approving from the climate and pollution agency. Initial study of mineral processing was carried out by SGS Minerals Services in 2011. Conceptual mine design proposals have been conducted by Nussir mining engineers based on available geological knowledge and reasonable assumptions about rock mechanical conditions.

1.3 Mining – a modifying factor

A mine design is an assessment of the technical and economical feasibility of extracting a mineral resource. In mineral resource classification rules like the JORC code or the NI-43-101, mining is also defined as one of the modifying factors considered when defining a resource as a reserve, illustrated in Figure 1. Other modifying factors are metallurgical, economic, environmental, marketing, legal, governmental and social factors. All these factors must be assessed and sorted out in a feasibility study before investors are willing to finance a mine start. The mining factor, mentioned above, is one of the most comprehensive factors, where geological factors, such as knowledge of the geology of the deposit, must be sufficient so that it is predictable and verifiable. Extraction and mine plans must be based on ore models, with statistical and variography to ensure the ore is sampled properly. Quantification of geotechnical risk—basically, managing the geological faults, joints, and ground fractures so the mine does not collapse (JORC, 2004). The mining factor is essentially a consideration of the technical risk.

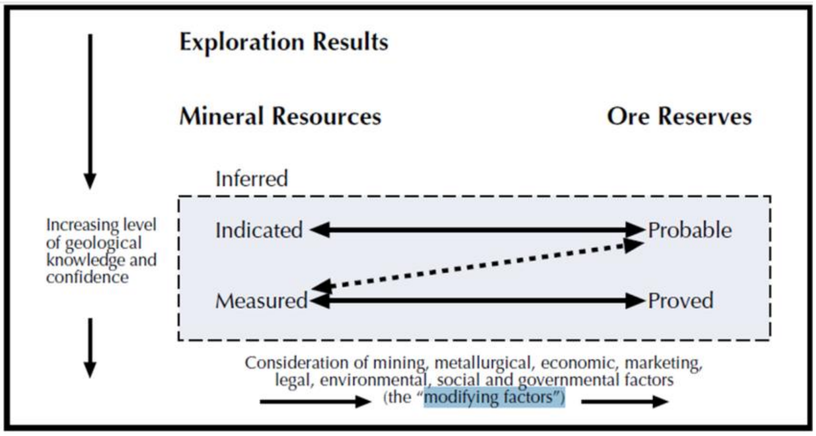

Figure 1 General relationship between exploration results, mineral resources and ore reserves

1.4 Description of this assignment

The candidate is expected to do an assessment of suitable mining methods for the Nussir copper deposit and suggest and build a conceptual mine design with the mining software Datamine. The need for equipment along with costs and product prices must be assessed briefly and special emphasis should be put on cut-off and annual production rates.

Geometric and qualimetric model from the project work, along with any new data, should be used in the construction of the mine design. Assumptions must be made where data is not provided and several scenarios should be presented to assess possible outcomes. Uncertainties must be clearly stated. Qualimetric model must be updated with dillution skin.

Representatives from the Nussir mining team will in an initial phase act as co-supervisors. Their contributions must clearly be stated in the thesis.

2. Previous mine design assessments

The technical and economical feasibility of mining Nussir deposit have previously been assessed by current owner Nussir ASA, previous owner ASPRO and external parts (see Table 1).

Table 1 List of previous reports evaluating mining of Nussir deposit

Report	Date	Authors	Comment
ASPRO 2191	30.4.1991	Husum, O; Gvein, Ø.	Economic evaluation
ASPRO 2194	23.5.1991	Iversen, E	Assesment of drift head grade
ASPRO 2197	05.6.1991	Hansen, Tord	Equipment needs and operating costs
SINTEF STF36 F91054	12.6.1991	Ludvigsen, Erik et. Al.	Feasibility study
Golder 09514950033.500	5.10.2009	Golder associates	Scoping study with mine design
Nussir PFS Draft	2012	Smeberg, P; Eriksen, O	Proposed mine design

Early evaluations of Nussir mining in 1990 where based on 45 drillholes distributed over the whole strike length, giving very low sample density. (Ludvigsen, et al., 1991) assessed mining of the thicker parts in the west by means of top down double bench stoping progressing from surface and 100m downward along the dip. Stopes where planned to be 30m long, 40m high. With cut-off 0,5% Cu, 30% dilution, 70% recovery, Ludvigsen assessed the economic feasibility of mining 1,3Mt at 1,28% Cu, 26 g/t Ag and 0,11 g/t Au. With an, at the time, copper price of 13kr/kg, annual production of 150 000tpa and operating costs of 166kr/t, the report stated the project as uneconomic.

Later evaluation by Golder 2009, proposed mining of eastern parts by similar method as proposed by (Ludvigsen, 1991), referred to as Sub-level retreat, with 40x40m large stopes illustrated in Figure 2. Open pit mining of the western part (not a current option due regulations) would commence simultaneously as the underground operation, producing 600 000tpa in total. Ore development costs were estimated to be as high as 42$/tonne due to the many drill drifts required for uphole drilling only. Recovery was estimated to be 70%, and dilution estimates surprisingly only 5%. With estimated ore production costs of 20$/tonne, waste development costs of 28$/t, copper price of 4300$/t and an average copper grade of

1,33%, Golder concluded the project to be uneconomic. A financial sensitivity analysis from the same report, stated that positive NPV could be achieved at a 5000$/tonne copper price.

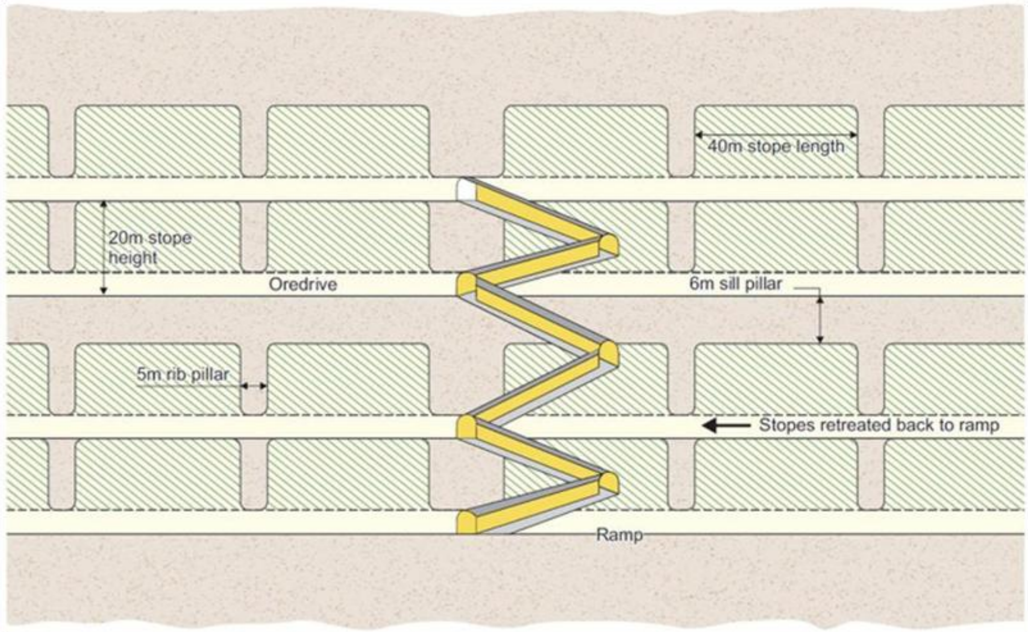

Figure 2 Sub-level retreat (SLR) mining method proposed for exploiting Nussir's eastern part. *Golder 2009*

The previous assessments of mine design presented above, share more or less the same mining method where short sublevel interval cause excessive ore development cost due to the necessity of many drill drifts. More recent mine design proposals carried out by Nussir’s own mining engineers Per Smeberg and Ola Eriksen, solve the excessive drill drift cost, by suggesting up and downward drilling and 50m sub-level interval. Their mine design proposed in the PFS Draft of January 2012, suggests up to 300m high stopes, supported by evenly distributed small pillars, instead of rib and sill pillars suggested in the previous reports. This reduced footwall development tunnels to one haulage level, ramp and some ore passes, making waste rock development costs as low as 50,81NOK/t. Mine operating costs where estimated to be 150NOK/t for 3,5million tonne total production.

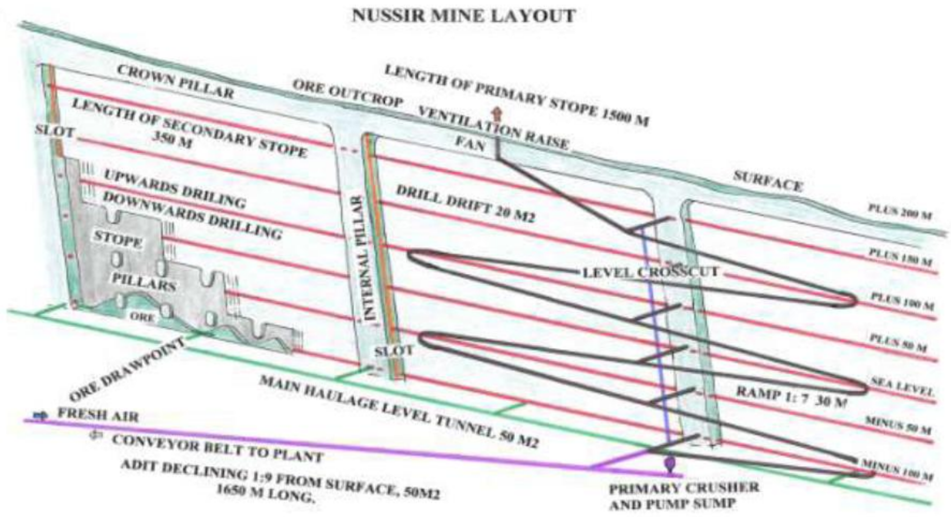

Figure 3 Mine design layout, SLOS with 50m subl.interval. and 300x350m stopes. **(Smeberg, et al., 2011)**

3. Geology

3.1 The Nussir deposit

Information below is taken from the NTNU report "Resource evaluation of Nussir copper deposit"

The Nussir deposit, named after the nearby Nussir mountain in Kvalsund community, lies within a stratigraphic sequence in the Komagfjord-Repparfjord Precambrian window. Deposit is indicated as a plate shaped dolomite horizon of 2-5m thickness, dipping 50-70 degrees to the north-west extending 9km along strike and proven to a depth of 500m below surface. It belongs to the upper parts of the stratigraphic Saltvatn group within the Stangvatn formation of Paleoproterozoic age (Pharaoh, et al., 1983). Deposition of the Saltvatn group is related to a continental rift basin, with a thick unit of conglomerate deposition followed by a transgressive event, depositing a thin layer of shale and then the dolomite. Mineralising event occurred over a long period of time, when circulating saline fluids leaching copper from the oxidized conglomerates, bringing them upward in the basin as copper chloride or copper sulphate complexes. As these copper bearing fluids entered the reduced dolomite-shale horizon, copper precipitated along the red-ox boundary, with a noticeable mineral zonation from the boundary and upward into the dolomite-shale. Highest copper grades are found near the footwall of the mineralised dolomite, and just below this zone there are high grades of Pt, Pd and Au(Sandstad, 2008). Dominating copper bearing minerals are chalcopyrite ($CuFeS_{2)}$, bornite (Cu_5FeS_4) and chalcocite (Cu_2S). The nature of this mineralisation, where copper is bound to a horizontal deposited strata, makes Nussir I a very regular and uniform copper deposit.

Today, some 2000 million years after the horizontal deposition of sedimentary rocks, one can see the stratigraphy dipping north-west, with the lower units of Stangvatn formation lying south near the mouth of Reppardfjord river and the upper parts in the north near Fiskevatn. Stratigraphy which Nussir I deposit is part of, have been tilted to this steep dipping position because of large scale folding during the SwecoKarelian and the Caledonian orogeny.

3.2 Geometric model

The geometric resource model, being the basis for the conceptual mine design in this assignment, is a 3D visualisation of geological information. It is based on a combination of hard sample data from 94 diamond drill core intersections, 20 percussion drill hole intersections, 10 surface chip samples and the soft data; surface outcrop mapping, structural data and magnetometry interpretations. Being a stratabound sediment hosted copper deposit (ref chapter 3.1), the geometric model is modelled as being one single plate shaped body, dipping 60-70 degrees in the eastern and central part and 40-55 degrees in the west, where deposits strike is bending in a big folding system. The outer limit of the geometric model is defined by drillhole samples with Cu grade above 0,3%, which is the geologic cut-off limit. Thickness is modelled by the length of each Cu >0,3% intersection. Minimum thickness for model is 1m and the average thickness in areas covered by drillholes is 3,2m.

Author of this report, constructed a geometric model in the project report *Resource evaluation of Nussir copper deposit*, using LeapFrog implicit modelling software. In march 2012, this model was updated with sample results from 6 new bore holes drilled in 2011 and sent to competent person Adam Wheeler, who makes the official resource estimations for Nussir ASA. Wheeler used this geometric model as a reference, when updating his existing geometric wireframe model in Datamine software. Wheelers Datamine model is based on the same sample data and interpreted data as the LeapFrog model. Difference is that Datamine model is constructed by wire frame triangulation between interpreted vertical sections, making model more rugged. LeapFrog model looks smoother as triangulations are finite approximations of surfaces with infinite detail. The Datamine constructed wireframe model is however chosen for further mine design evaluation, as it allows stopes to be evaluated by the block model which is constructed within the geometric Datamine model.

3.3 Block model

The block model is a quantitative estimation of mineral grades within the geometric model, based on geostatistics. Block orientation correspond to Nussir structures, dipping 55 degrees towards azimuth 340 for eastern and central parts and dipping 40 degrees towards azimuth 70 in the western part. Block size is 5m in x direction (down dip), 5m along y direction (along strike), with z direction being block thickness (Wheeler, 2011). Each block provide a range of estimated parameters for a given location in space. Areas which are covered by a grid of at least 250m x 250m (along-strike x down-dip) drilling were demarcated as indicated resources. All other modelled resources were allocated as inferred. No areas have sufficient sampling density to be demarcated as measured.

Grade estimations are based on composites created across each indentified intersection, therefore making variable composite length as deposit thickness varies. A variogram model was defined against the basis of a copper grade experimental variogram, which is essentially isotropic within the plane of the mineralised zones. Wheeler identified from the experimental variogram, an overall structure with a range of about 1000m with 2/3 of the variability being reached by approximately 500m. Revealing shorter range variations, require denser drilling. Copper grades were interpolated by three different ways for comparative purposes, ordinary kriging, inverse-distance weighting and nearest-neighbour. The Kriging estimator was chosen for its smoothening nature, avoiding the extremities of the nearest-neighbour and inverse-distance weighting.

Table 2 Reported indicated and inferred resources of Nussir copper deposit. **(Wheeler, 2012)**

Region	Indicated Resources								Inferred Resources							
	Tonnes Mt	Cu %	Ag g/t	Au g/t	Pd ppb	Pt ppb	Cu Eq %	Width m	Tonnes Mt	Cu %	Ag g/t	Au g/t	Pd ppb	Pt ppb	Cu Eq %	Width m
West	1.95	1.10	20.4	0.08	6.2	0.9	1.34	5.0	13.21	1.15	23.2	0.10	4.2	5.4	1.43	4.1
Central									8.17	1.18	9.7	0.12	10.0	13.3	1.36	
East	4.87	1.11	12.5	0.17	63.1	88.6	1.40	2.5	2.67	0.97	10.3	0.13	51.3	72.5	1.21	1.9
TOTAL	**6.82**	**1.11**	**14.7**	**0.14**	**46.8**	**63.5**	**1.38**	**3.0**	**24.05**	**1.14**	**17.2**	**0.11**	**11.4**	**15.5**	**1.38**	**3.3**

Notes **. Cut-off grade = 0.3% Cu**
. Minimum thickness = 1m
. Cu Eq equivalent grade calculated using prices of Cu $4,300/t, Ag $12/oz, Au $900/oz, Pd $200/oz and Pt $950/oz

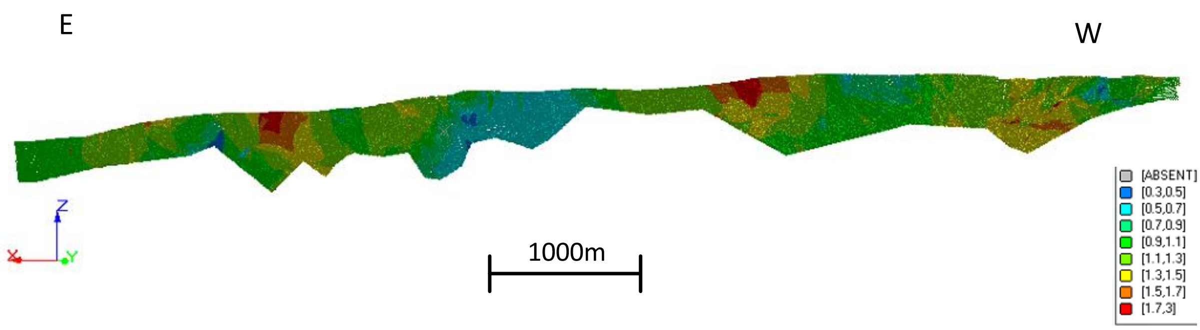

Figure 4 Block model of Nussir showing Kriging estimated copper grades. **(Wheeler, 2012)**

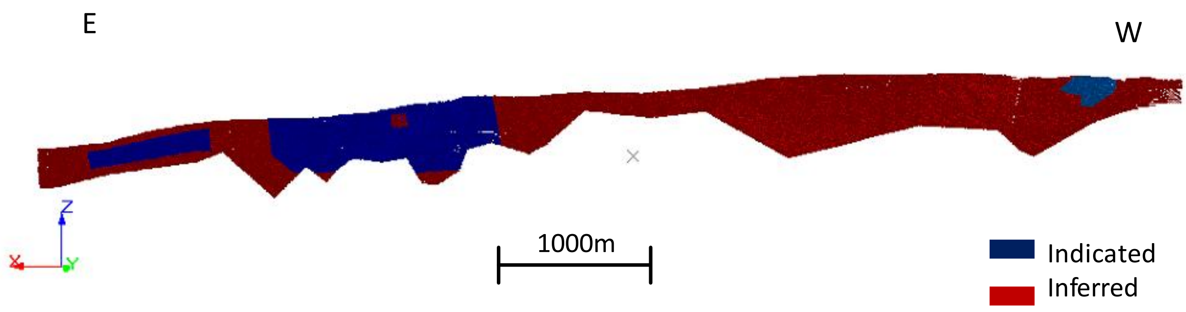

Figure 5 Block model of Nussir divided in spatially sampled inferred resources (red) and indicated resources (blue) being those within a grid of sample intersections not more than 250x250m. **(Wheeler, 2012)**

4. Geotechnical considerations

The following chapter consider relevant theory for underground mine excavations and how stability can be assessed prior to mine start. All relevant geotechnical data for the Nussir area is presented in sub ch. 4.2 and the calculation methods for this report is presented in ch.4.3.

4.1 Background

4.1.1 Empirical Design Methods

Mine structures, whether it is a pillar or an opening, is influenced by numerous blocks of intact rock, which can be individually assessed for their properties on a laboratory scale. The challenge arise, when trying to assess a structure in a mine wide scale, where the rock mass behaviour is difficult, or impossible to determine solely on rock block properties. Instead of a design relying solely on a deterministic approach, empirical methods can be implemented to assess stability of structures by the use of past practices to predict future behaviour based upon factors most critical towards the design (Pakalnis, 1998).

Design methods can be categorized as being analytical, observational or empirical, but they can also be combined as steps in the assessment of a design. In rock mechanics, analytical design methods are based upon an estimate of the constitutive behaviour of a rock mass, such as a failure criteria, which may be Hoek-Brown criterion or Mohr-Coulomb criteria. Once a constitutive rock mass behaviour is given, it becomes part of the design method and is not varied. Empirical methods on the other hand, combine more input data, providing an estimate of the constitutive properties of a rock mass. Based on the collection and analysis of case histories, the input data, such as joint orientation, orientation of opening, strength and spacing, are given weights reflecting the relative influence on stability. Numerical codes, analytical tools and observational approaches still have a great value as tools to the overall process, which will incorporate an empirical component towards the design. A failure criteria can be empirically derived based on weighted constitutive properties of the rock mass and a measure of the underground geometry (Pakalnis, 1998).

Empirical methods have gained acceptance since the mid 80’s, largely because of their predictive capabilities, needed for mine design evaluation. The stability graph method for open stope design, proposed by Mathews et al (1981) and subsequently modified by Potvin

(1988) and Nickson (1992), is specifically chosen for the evaluation of Nussir mine design. The method assesses the likelihood of major instability or caving of the excavation surfaces forming the open stope, using the Stability Graph which compares the Stability Number (N) of the rockmass in which the surface is excavated and the hydraulic radius (HS) of the surface. The stability number N is given by:

$$N = Q * A * B * C \qquad \textbf{4.1}$$

Where:
Q Factor - Also called the Q-value, expressing quality of rock, explained in ch.4.1.2.

A Factor - This value is designed to accout for the influence of high stresses reducing the rock mass stability. The A value is determined by the ratio of the unconfined compressive strength of the intact rock divided by the maximum induced stress parallel to the opening surface. The A factor is set to 1.0 if the intact rock strength is ten (10) or more times the induced stress indicating that high stress is not a problem. The A factor is set to 0.1 if the rock strength is two (2) times the induced stress or less indicating that high stresses significantly reduce the opening stability.

B Factor - This value looks at the influence of the orientation of discontinuities with respect to the surface analysed. This factor states that joints oriented at 90° to a surface are not a problem to stability and a value of 1.0 is given to the value of B. Discontinuities dipping within 20° to the surface are the least stable representing structure which can topple within the stope. A value of B=0.2 is given for this condition.

C Factor - This value considers the orientation of the surface being analysed. A value of eight (s) is assigned for the design of vertical walls and a value of two (2) is given for horizontal backs. This factor reflects the inherently more stable nature of a vertical wall compared to a horizontal back.

The Hydraulic Radius (HR) = excavation surface area/excavation surface circumference.

The different stability numbers calculated from the available data and the geometry of the openings surfaces (HR), can be plotted in the modified stability graph, for comparison with case historic data and empirical failure criteria's. Mathews stability graph initially proposed in the 80's have been extended with use of a significantly increased database of mining case histories. (Mawdesley, et al., 2001) extended Mathews stability graph with a database containing 400 case histories, giving them the possibility to perform logistic regression to delineate and optimize placement of stability zones statistically. Isoprobability contours have been generated for all stability outcomes (Figure 6), represented by a percentage likelihood of stability. The advantage of the logistic regression lies in its ability to minimize the uncertainties reflected in the method through the use of maximum likelihood estimates. The risks associated with use of the Mathews method can now be quantified and the true statistical significance of the stability zones understood.

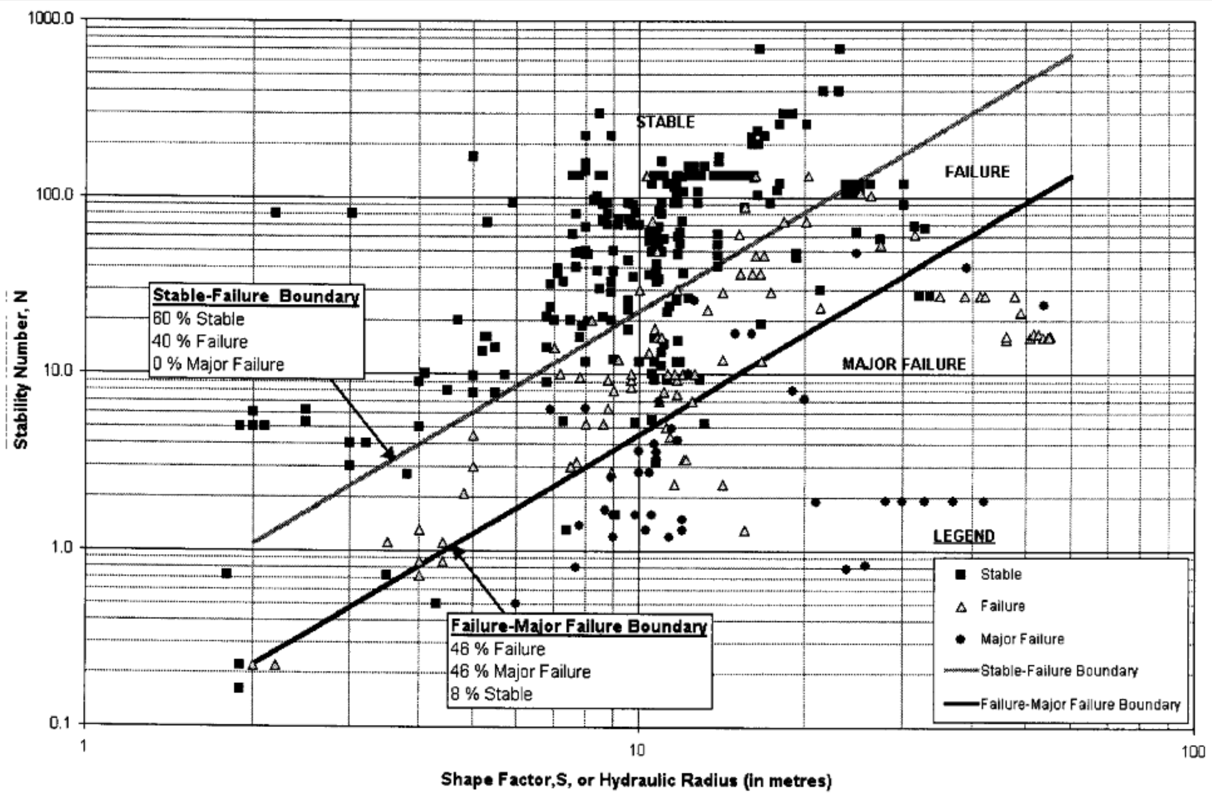

Figure 6 Extended Mathews' stability graph based on logistic regression

4.1.2 Q-system

The Q-system for rock mass classification developed by Barton, Lien, and Lunde (1974), expresses the quality of the rock mass in the so-called *Q-value*, on which are based design and support recommendations for underground excavations. The *Q-value* varies on a logarithmic scale from 0,001 to 1000, where everything above 10 is regarded as good rock mass quality (Barton, et al., 1974). Q is calculated by:

$$Q = \frac{RQD}{J_n} x \frac{J_a}{J_r} x \frac{J_w}{SRF} \qquad \textbf{4.2}$$

Where,

RQD = Rock Quality Designation;

Jn = Joint set number;

Ja = Joint alteration;

Jr = joint roughness;

Jw = Joint water number; and,

SRF = Stress reduction factor.

The first quotient appearing in equation 4.2, *(RQD/Jn)* represents the overall structure of the rock mass, and it happens to be a crude measure of the relative block size
The second quotient *(J,./Ja)* represents the roughness and degree of alteration of the joint walls or filling materials.
The third quotation *(J~/SRF)* consists of two stress parameters. The parameter J~ is a measure of water pressure, which has an adverse effect on the shear strength of joints due to a reduction in effective normal stress. Water may in addition cause softening and possible outwash in the case of clay filled joints. The parameter *SRF* is a measure of: (1) loosening load in the case of excavation through shear zones and clay bearing rock, (2) rock stress in competent rock, (3) squeezing or swelling loads in plastic incompetent rock. The quotient *(J~/SRF)* is a complicated empirical factor describing the "active stresses" (Barton, et al., 1974).

4.2 Geotechnical data

4.2.1 Drill ability

A drill ability analysis was carried out by (Jóhannsson, 2001) at Sintef, on behalf of ASPRO, to determine relevant rock parameters for production and development drilling in the Nussir deposit. Core sample material from footwall, mineralisation and hangwall were laboratory tested to determine the Drilling rate index (DRI) and Cutter life index (CLI), according to NTNU, project report 13A-98 DRILLABILITY Test methods. A differential thermal analysis was also done, to quantify quartz and pyrite content. Relevant results from Sintef laboratory testing is obtained in Table 3.

Table 3 Results from Sinfef laboratory analysis and calculated indexes relevant for drillability of Nussir rocks.

Results from analysis:	**Hangwall**	**Footwall**	**Mineralisation**
Embrittlement number (2-4mm)	18,3	17,5	14,9
Embrittlement number (11,2-16mm) calculated	38,7	37,9	35,4
Sievers J-value (pair. Foliation)	31,2	41,1	42,4
Abration value cutter steel (AVS)	5	4,5	1
Quartz % (DTA)	25	29	18,5
Pyrite % (DTA)	<0,05	0,2	0,5
Calculated indexes	**Hangwall**	**Footwall**	**Mineralisation**
Drilling rate index (DRI)	46	47	44
Cutter life index (CLI)	28	32,4	58,5

The indexes can be categorized by using a classification based on past experience from Norwegian tunnelling and a record of over 2000 samples tested in laboratory (Bruland, 1998). From the classification in

Table 4, the drill ability is categorized as medium for all Nussir rocks. The Cutter life index is categorized as high for hangwall and footwall and very high for the mineralisation. DRI values will provide valuable input data for net penetration rate and consequently drill rig performance capacities in the mine design.

Table 4 Category intervals for drillability indexes. **(Bruland, 1998)**

Category	DRI	BWI	CLI
Extremely low	- 25	- 10	< 5
Very low	26 - 32	11 - 20	5.0 – 5.9
Low	**33 - 42**	**21 - 30**	**6.0 – 7.9**
Medium	**43 - 57**	**31 - 44**	**8.0 – 14.9**
High	**58 - 69**	**45 - 55**	**15.0 - 34**
Very high	70 - 82	56 - 69	35 - 74
Extremely high	82 -	70 -	≥ 75

4.2.2 Q values

The parameter for rock quality, Q-value after (Barton, et al., 1974), was registered in 2009 by (Golder, 2009), based on core material from 6 diamond drill holes. Samples were half core, as the other half had been chemically analysed, which means that data should be treated with caution. Also the third quotation of the Q-value *(J~/SRF)*, must be treated with caution, as there are in fact no actual stress measurements underlying these calculations. (Golder, 2009) set the quotation to be 1/20 as 20MPa has been used as a qualified guessing of the horizontal stresses, based on regional geotechnical experience.

Only Q measurements from drill holes intersecting the eastern part of Nussir have been assessed in this report, presented in Table 5 below.

Table 5 Selected Q values from samples intersecting eastern area of Nussir

Borehole		NUS-DD-08-027		NUS-DD-08-023		NUS-DD-08-011		NUS-DD-08-016	
	Av. Q	Depth [m]	Q	Depth [m]	Q	Depth [m]	Q	Depth [m]	Q
Hangwall	43,55	93,5	70,0	118,4	160,7	65,0	19,3	395,1	35,9
		95,0	4,1	124,0	43,3	69,2	20,5	401,9	40,2
		96,0	45,2	125,5	33,6	93,6	37,7		
		99,1	48,5	127,4	0,0	113,1	19,3		
		100,8	75,0						
Mineralisation	79,65	101,5	14,1	128,0	166,7	134,5	30,0	405,2	147,6
		102,5	125,0			135,4	48,1		
						142,1	26,1		
Footwall	36,23	103,3	54,6	129,2	70,5	144,4	18,6	407,3	0,0
				132,5	93,3				
				134,0	10,1				
				136,6	61,2				
				141,5	0,0				
				142,0	17,8				
Depth of mineral. [m.a.s.l]		-23		46		157		83	

Registered Q values indicate good rock conditions, with consistency in values. Mineralisation may have a higher average Q value as a result of fewer registrations. Upper and lower extremity filtering have been applied prior to stability calculations, by removing the highest and lowest Q value registered for the rock category.

4.2.3 Rock strength

Selected core sections from 4 different bore holes where tested for their physical properties, two bore holes where tested by (Myrvang, 2009) and two other boreholes tested by (Hagen, et al., 2012). Three of the tested boreholes are located quite close to each other, just east of the central part of Nussir (see Figure 7). The last borehole, NUS-DD-08-033 is located just east of the big fold in the west. Test results from the 4 boreholes are listed in table Table 6 below.

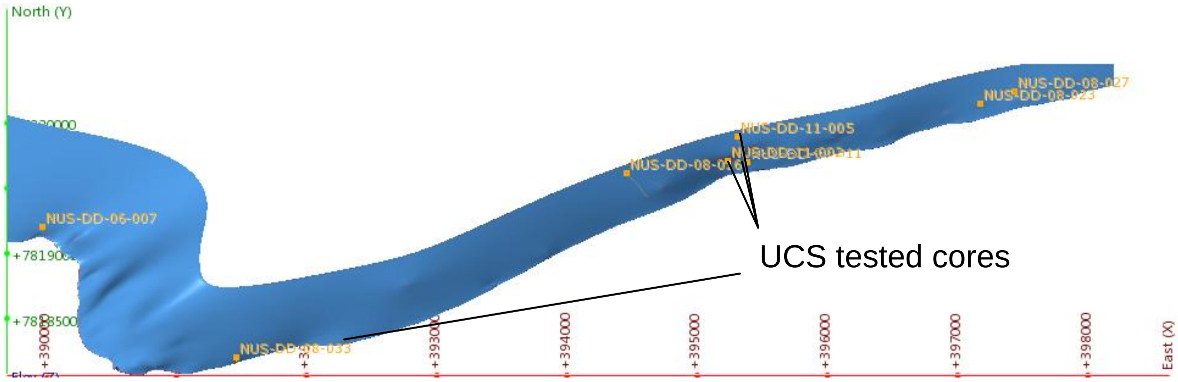

Figure 7 Deposit seen from above, Q-value tested and UCS tested cores highlighted. *LeapFrog*

Table 6 Laboratory test results from core samples. *Sigma H 2009 and Sintef 2012*

Category	Borehole	Depth	Lithology	UCS	Tensile str	E-Module	Poissons ratio	Velocity	Spec. weight
		m		Mpa	Mpa	Gpa		m/s	kg/m3
hangwall	NUS-DD-08-033 *	47,0 -48,0	SST	122***				5384	2660
	NUS-DD-08-011 *	93,6 -95,0	SST	125***	7***				
	NUS-DD-08-011 *	135,4 -142,1	SST	92				5383	2740
	NUS-DD-11-002	142,0 -143,0	CLY	69,1		68,7	0,2	5422,1	2734
	NUS-DD-11-005	309,0 -310,0	CLY	67,7		69,5	0,18	5233	2715
mineralisation	NUS-DD-08-033 *	51,5 -53,5	DOL	102***	6***				2730
	NUS-DD-11-002	147,0 -148,0	DOL	120,8		60	0,24	5853,4	2698
	NUS-DD-11-005	316,0 -317,0	SST	39,2**		45,7	0,27	5560,9	2667
footwall	NUS-DD-11-002	152,0 -153,0	CLY	99		74,8	0,19	5667,4	2675
	NUS-DD-11-005	322,0 -323,0	SST	175,8		77,3	0,19	5732,1	2657

* From Lab tests carried out by Sigma H 2009, rest is from Sintef lab test 2012
** Unreliable test result, comprise of only one core
*** Based on point load index

Results from the uniaxial compressive strength test indicate strong rock in both hangwall, mineralisation and footwall. Sintef 2012 measurements must be treated with caution, (Hagen, et al., 2012) report that the ISRM test standard requiring 5 test cores for each measurement, could not be achieved due insufficient core material received at the lab. The low extremity UCS result, 39,2Mpa for NUS-DD-11-005 mineralisation is based solely on one core and should therefore be excluded from any strength estimates used in further calculations.

4.2.4 Stress estimates

The following text describe professor in rock mechanics, Arne Myrvang's expressions regarding regional stress conditions in Finmark. Taken from (Myrvang, 2009).

‘‘ No in-situ rock stress measurements have been carried out near Nussir. However, regionally in Fnnmark, horizontal stresses in the order of 20 MPa have been measured by overcoring in the Sydvaranger iron ore mine, Kirkenes, the Stjernøy nephelin syenite mine, Altafjord, and the Biddjovagge Cu, Au mine, Kautokeino. In the Sydvaranger case, high horizontal stresses were measured at less than 50 m depth, and caused heavy spalling in the roof of the main access ramp to a planned underground mine.

Another indicator of high horizontal stress has been up to 15 mm off-sets of vertical boreholes in road-cuts in the Laksefjord and Porsangerfjord areas, i.e. not far from Nussir. Typically, in most cases the maximum horizontal stress seems to be oriented N-S to NW-SE. However, in some of the locations high horizontal stresses have been measured also in the E-W direction. Based on this, at Nussir, one must expect horizontal stresses considerably higher than the vertical stress due to gravity even at moderate depths. This may result in stress induced roof spalling in different types of drifts, and may also affect the stability of the hanging wall of stopes. High horizontal stress normal to the strike of the orebody could result in shorter stope lengths and increased waste dilution, and also spalling in drift along the strike. On the other hand, high horizontal stress along the strike could favour longer stopes and less chance of waste dilution.’’

Based on personal communication with Arne Myrvang 8th of may 2012, discussing Nussir stress consitions, I establish the following assumptions for further use in stope calculations:

Table 7 Assumptions for stresses acting on Nussir deposit

Component	**Notation**			**Unit**
Vertical stress	σv	=	$\rho g h$	*Mpa*
Density rock	ρ	=	2770	*kg/m3*
Gravity constant	g	=	9,81	*m/s2*
Max. Horizontal stress	σH	=	$2 \times \sigma v + 10$	*Mpa*
Direction of max. horizontal stress			160	*Azimut*
Min.horizontal stress	σh	=	$0{,}66 \times \sigma H$	*MPa*
Direction of min. horizontal stress			70	*Azimuth*

4.3 Stope design

In order to maintain a high level of safety in the mine, stopes must be dimensioned in accordance with the actual conditions, at the same time keeping in mind that stope design has a high influence on mining efficiency, recovery and costs. With the available data and knowledge regarding rock mass and stress conditions, having a high level of uncertainty, one can only suggest a conceptual stope design.

As the stope width will be given by ore thickness, varying between 2-5m, the main parameters to determine is the maximum stope height one can allow before a horizontal sill pillar have to be left, dividing the ore vertically. And the maximum stope length along strike one can allow before a rib pillar or internal pillar is left for support. Stope height should be maximized, reducing drift meters per tonne, but sill pillars must eventually divide the 300m high stoping layout, reducing the chances for stope instability, hangwall breakings and dilution. Ore flow downwards in the stope could also be a problem if footwall boundary is irregular. Stope length and the number of rib pillars along strike influences mining efficiency and stoping cost due to the 20m long slot hole required for blasting behind each rib pillar. Upward and downward drilling from drifts should be utilized for efficiency, and optimal hole length is set to be 20m along dip, making 17,66m vertically. Drill drifts are 4,5m high. Stopes depth will be dimensioned to utilize drilling from two drifts. Shallower stopes can be higher, fitting three drifts. Vertically. Dimensions and stress conditions for the two stope sizes are listed in Table 8.

Table 8 Inputs for stope stability calculations: Chosen dimensions, mineralisation UCS average and estimated stresses acting in the centre of the stope, calculated from estimations in chapter 4.2.4.

	Dimensions			***Stresses***		
	Vertical height	62	m	Vertical	7,1	Mpa
-80 stopes	Dip height	72	m	Hor.-strike	16,0	Mpa
				Hor.-dip	24,3	Mpa
	Vertical height	102	m	Vertical	4,4	Mpa
-10 stopes	Dip height	118	m	Hor.-strike	12,4	Mpa
				Hor.-dip	18,8	Mpa
	Span (rib pillar)	5	m			
	Length	40	m	UCS	120	Mpa
	Dip	60	^{o}			

The stresses used in further calculations are basically qualified guessing and should be backed up by real measurements. One should also note that all Q-values in the following Table 9 are calculated with an assumed stress reduction factor (SRF) of 20Mpa (Myrvang, 2012). The B-factor assigned each evaluated stope surface in Table 9 expresses the joint orientation in respect to each surface. Determined by (Golder, 2009) who performed geotechnical core logging for Nussir.

Table 9 Stability number inputs. Q from **Table 5** , A calculated by stress and UCS relation, B and C taken from Golder 2009, HR calculated from stope dimensions. (a) lower -80 stope, (b) higher -10 stope

(a)	***Stability numbers lower stope (-80 to -18 level***							
	Q						**N**	
	Low	High	A	B	C	HR	Low	High
Back	26,09	147,62	0,45	0,8	1	1,40	9,4	53,1
Vertical end	26,09	147,62	0,45	0,5	8	1,44	47,0	265,7
Hangwall	19,29	75	0,7	0,4	4,5	12,8	24,3	94,5
Footwall	10,1	70,45	0,7	0,4	8	12,8	22,6	157,8

(b)	***Stability numbers higher stope (-10 to +92 level)***							
	Q						**N**	
	Low	High	A	B	C	HR	Low	High
Back	26,09	147,62	0,59	0,8	1	1,40	12,3	69,7
Vertical end	26,09	147,62	0,59	0,5	8	1,46	61,6	348,4
Hangwall	19,29	75	0,95	0,4	4,5	14,9	33,0	128,3
Footwall	10,1	70,45	0,95	0,4	8	14,9	30,7	214,2

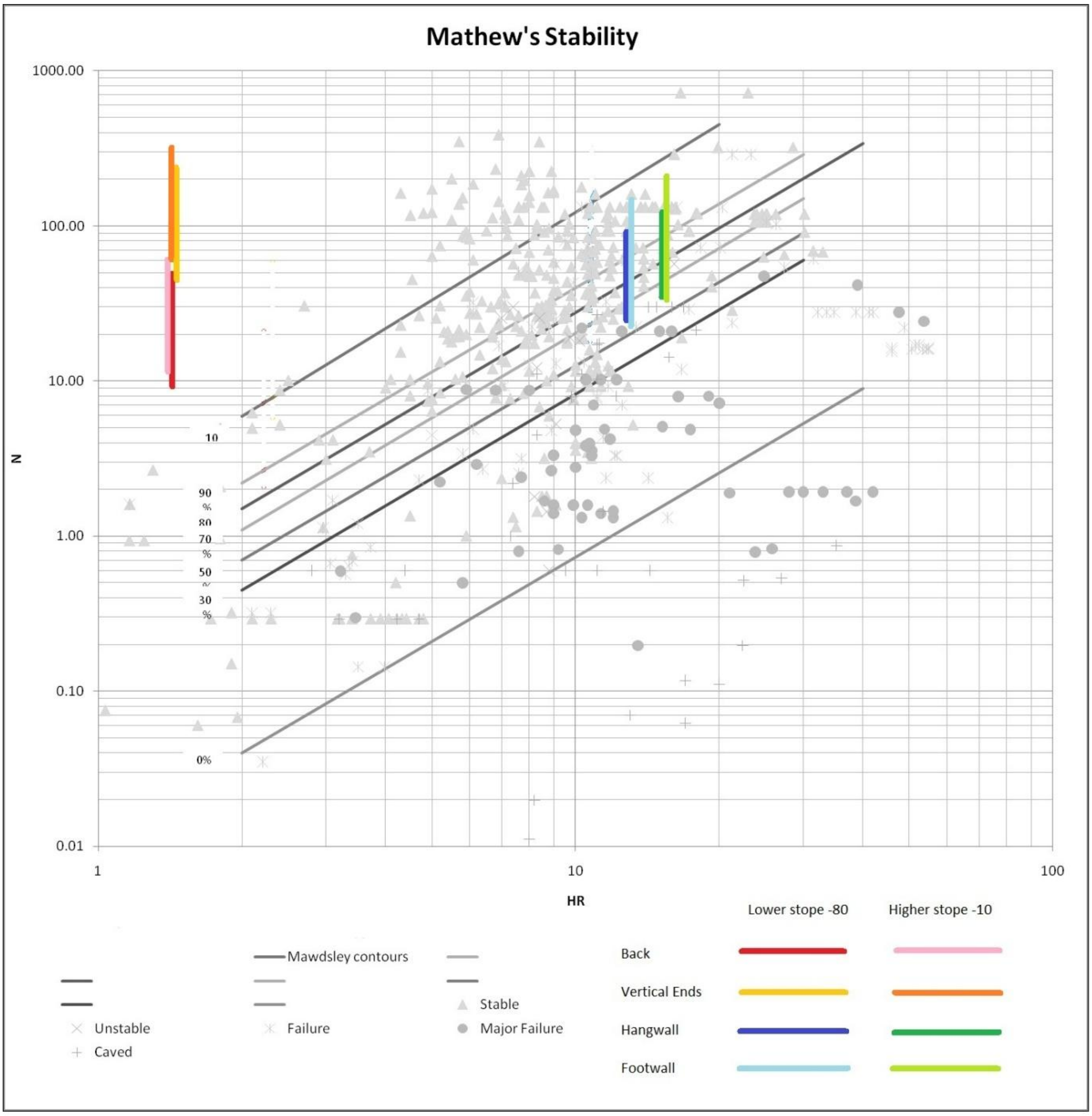

Figure 8 Stability chart for two different Nussir stopes using Mathews method and Mawdsley et.al.2001 Stability contours. Coloured lines represent the stability number span due to variability in the Q-value

From Figure 8 we identify the vertical ends and the back as being stable, because the surfaces have a very short open span. The hangwall and footwall is identified as being stable for higher q-values, implying that the relatively large stopes require rock mass to be very good and additional cable bolting in weak areas to secure stability during production. Both hangwall and footwall surfaces from the lower and the higher stope lie below the 100% stability regression line. With the variability in Q values, (Mawdesley, et al., 2001) stability contours and the stability number calculations in Table 9 one can state at this point that the likelihood for stope stability may be anything between 58% and 98%. The likelihood for stability in stopes should ideally be 90% (Lappalainen, 2012). To achieve this, additional yielding pillars should be left in the centre of each stope, to reduce the open span (Lappalainen, 2012). Cable bolting of hangwall should be carried out additionally when needed.

4.4 Dimensioning pillars

A pillar should be dimensioned to carry hangwall load and stress exceeded just above the pillar. The load carried by each pillar, $\sigma_{p,}$ depends on the pillar size, strength of rock comprising the pillar, pillar intactness and an appropriate safety factor, reflecting a stability criteria.

A first approach to pillar dimensioning is to determine the required pillar area in relation to the excavated area. Tributary method analysis is used to calculate the necessary pillar dimension to withstand the overlying pressure exceeded by the rock above (Myrvang, 2001). The method is common for horizontal room and pillar mining, and the same approach is used in this case, introducing a 60° dip. The method considers pillar rock strength $\sigma_{p,}$ set to be equal to the uniaxial compressive strength (σ_c), the stress component normal to the pillar cross section (σ_t), the area of the pillar cross section (A_p) and the area of overlying rock carried by the pillar analysed (A_t). The stress component (σ_t),will be a component of both vertical and horisontal stresses $\perp(\sigma_V + \sigma_H)$, in this case illustrated in Figure 10 below. The pillar cross section area/tributary area ratio, A_p/A_t can be calculated by formula:

$$\frac{A_p}{A_t} = \frac{\sigma_t \cdot f}{\sigma_c} \qquad \textbf{f = safety factor} \qquad \textbf{4.3}$$

$$\sigma_t = \sigma_H \sin(\alpha) + \sigma_V cos(\alpha) \qquad \boldsymbol{\alpha} \textbf{ = deposit dip} \qquad \textbf{4.4}$$

For tunnels and excavations where personnel stay unprotected, safety factor should be 2, but for retreated mining stopes, where no personnel need to stay, a safety factor of 1 is sufficient. With the stress assumptions from chapter 4.2.4, where both horizontal and vertical stresses increase as a function of depth (z), the A_p/A_t have to be calculated for a variety of depths. Results are presented in the figure below:

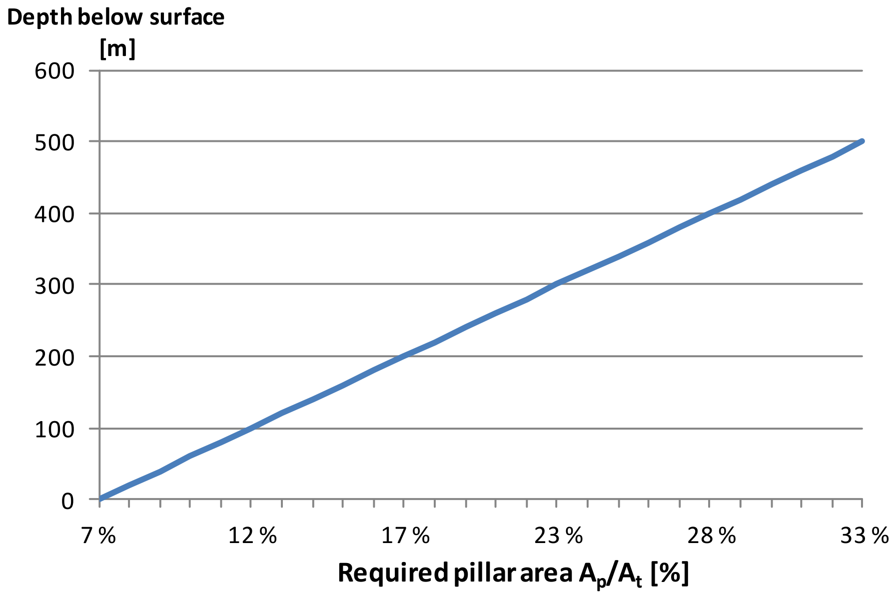

Figure 9 Results from tributary area method. Given UCS=120Mpa and the stress estimates from 4.2.4.

The tributary area method results indicate that the necessary pillar cross section area for a pillar located at the deepest parts of the suggested Nussir mine, must be 23% of the deposit surface area. At shallower levels, fewer, or smaller pillars could be sufficient, according to results in Figure 9, but in our case, we have increased the stope height for shallower levels rather than decreasing pillar sizes. Pillar size for the main supporting pillars between the stopes, will be 5m thick rib pillars and 8m high sill pillars.

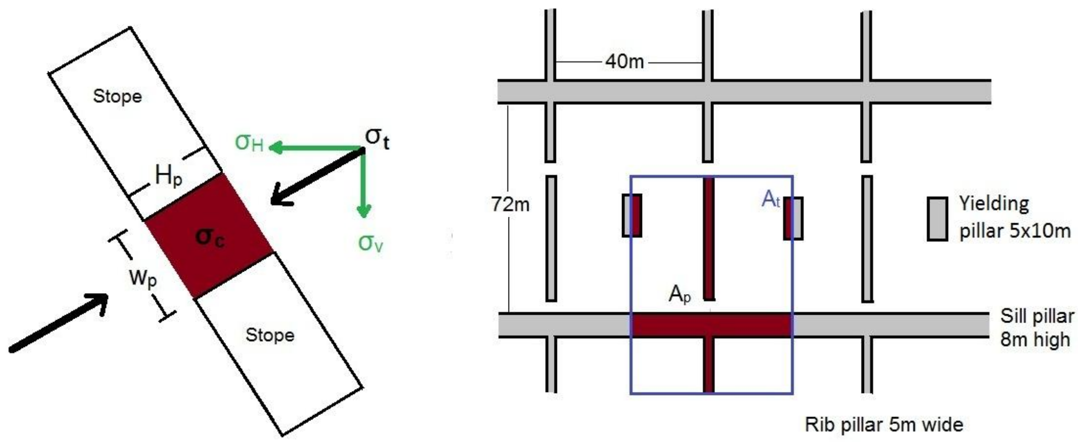

Figure 10 Illustration of tributary area method for calculating necessary pillar area A_p carrying the area A_t with the load σ_t caused by stresses.

In addition to the overall rib and sill pillar support, a large internal rib pillar should be left between each stoping section A,B,C and D for global stability of mine. Suggested pillar width is 10m, but this could easily be increased if necessary. Also additional yielding pillars should be left in the centre of each stope with 5m width and 10 m height. In the highest stopes with 118m dip height, two of these pillars will be needed. The yielding pillars will be left below drift in the downhole stoping sequence. By the term yielding, it means that the pillar is planned to yield over after time, due to stresses, stille being able to carry some load (Lappalainen, 2012). There will be some 380 tonnes ore loss per yielding pillar including some 15 tonnes of crushed material trapped on top of each pillar.

Ore loss in pillar design for 62m high stopes, illustrated in Figure 10 is calculated by Ap/At relation to be 20,1% including 15 tonnes crushed ore trapped on yielding pillar.

4.5 Stability of mine

The empirical Mathew's method, modified by (Mawdesley, et al., 2001), is a well regarded calculation method, but input data used in this report, must be treated with caution. The proposed stope layout, with given dimensions in ch.4.3 p.21, are therefore empirically estimated to be stable with a probability range of 58-98%, rather than stating a fixed probability number. This probability range is used to reflect the variability of registered Q-values, rather than Q-value average. The Q-values also introduce uncertainties in itself, relying on a guestimated stress reduction factor of 20MPa. Values area also a subject of an individual geologists work, in this case (Golder, 2009).

The mine is highly dependent on achieving stability. Even though stopes are retreated with no personnel staying within the open stopes, 90% of stopes are recommended to be stable after excavation (Lappalainen, 2012). The recommendation is considered viable, coming from an experienced mining engineer with narrow mining experience, and logical in the sense that stope failure incur ore loss and dilution, but the likelihood that 10% of stopes fail, is a risk we can defend by cost savings of utilizing bigger stopes.

The largest uncertainty for mine stability in this report is the in-situ stresses acting upon the mine design. Both direction and magnitude of the principal stress, used in calculations are guestimates. Direction assumed normal to strike, is the optimum direction when opening stopes along strike. The consequence of principal stress oriented parallel to stopes, can be derived from the Mathews method stability number (N) equation. A factor will decrease when stope parallel stresses are higher, giving a lower stability number for the stope surface. The magnitude of horizontal stresses have high influence on mine stability as it determines the strain exceeded on stope walls and pillars. Assumed values of horizontal stress, are believed to be realistic, with a slight probability of being exaggerated. The assumption of $\sigma_H=2\sigma_V +10$ (Mpa) is anyhow backed up by regional stress conditions in Finmark and recommendations from a rock mechanical professor with Finmark experience (Myrvang, 2012).

Pillar dimensions and stope open spans, are identified as the most important mine parameters for stability. Pillars carry the concetrated pressure from surrounding rock, when stopes are opened, and stope surfaces have a certain span of unsupported area between pillars. Pillar dimensioning by tributary area method, is regarded as suitable for the overall pillar area dimensioning at this early stage of mine planning, before numerical analyses provide foundation for optimal pillar dimensioning. Results should however be treated with caution: The load in which each pillar is able to carry σ_p, will normally be lower than the rock strength comprising the pillar σ_c, due to pillar spalling and fractures. It is therefore realistic to assume that reality demand larger pillar area, creating lower ore recovery.

Stability of hangwall and footwall in stopes, cannot be achieved without additional support of yielding pillars and cable bolting, according to Mathew's method. Whether the proposed additional support is sufficient, is difficult to quantify without numerical analysis. Stopes are anyhow chosen to be dimensioned this way, given the stability uncertainties, as it allows more cost efficient mining, than smaller stopes.

The other option is of course to reduce stope size to 40m long, 45m high separated by rib and sill pillars, providing a hangwall hydraulic radius of 11,3, plotting within the 70% stability contour for the lower Q-value scenario. Consequence is 33% more drill drifts required, yielding 8% increased mining production cost.

5. Dillution and cut-off

5.1 Economic definition of ore

Economic definition of ore as that definition which maximize the net present value of a mining operation. Present value dependent on time, the resource and a set of variables in which describes the way in which the operation is to be conducted (Lane, 1988). Purpose of an economic model is to provide a means for calculating the effect of changes in certain variables. Economic model components are:

Mineralised material; Also called the mining component, concerning the development needed to access the interior of a mineralised body. For underground mines, it includes Development, raising and cross-cutting. Costs are incurred per tonne of mineralised material made accessible and capacity is the maximum rate at which the needed development can be carried out.

Ore; can also be called the treatment component, concerning with the extraction and treatment of that parts of a mineralised body defined to be ore. For underground mines, it includes Stoping, Hauling, grinding and separation. Costs are incurred per tonne of ore extracted and the capacity is the maximum of throughput of ore that the installation can handle.

Mineral; can also be called the marketing component, involving smelting refining and selling. Costs are incurred per unit of mineral and the capacity is a limit of the output of mineral.

Cash flow arising from one unit of mineralised material is:

C = Marketing income – Treatment cost – Mining cost – Fixed costs

$$C = (p - k)xyg_h - xh - m - f\tau \qquad 5.1$$

Where; p = price per unit of mineral

k = marketing cost/unit unit mineral output (smelting, refinement and selling)

y = yield of mineral in the treatment process in percentage (process recovery)

g_h= average grade of the ore as a mineral, ore ratio (head grade)

x = amount of mineralised material classified as ore (tonnage in the unit)

h = treatment cost per unit throughput (operating and processing cost/tonne)

m = Development cost per unit throughput (capital cost per tonne)

f = time costs per year (fixed costs like electricity, administration etc.)

τ = time taken to work through one unit of mineralised material.

5.2 Economic Cut-off

Economic cut-off ($g_{cut\text{-}off}$) is applied to mineralised material to define ore, thus being the mineral ore ratio (Lane, 1988). It should not be mixed up with the geologic cut-off, which is applied to overall rock mass to define mineralised material. The average grade of ore within the cut-off boundary (g_h) is that grade, fulfilling a certain profit criteria or the grade in which the ore pays for itself, hence the break even grade given by equation:

$$g_b = \frac{h}{(p-k)y} \qquad \textbf{5.2}$$

A cut-off grade is a complex number depending on economic variables, but also on the nature of the mineralisation. It dictates the selection of mining method, by defining the location and tonnage of ore. A high cut-off grade, may require more selective mining method, affecting mining cost and capacity. If high capacity method is chosen instead, mining costs per tonne may decrease, allowing a lower cut-off (Hall, 2003). Change in raw material prices, costs and technology may also affect the cut-off for production in a long term or short term. It is tempting to mine the high grade stopes at once, gaining quick money, but saving them for periods of low raw material prices, may be the crucial step to avoid mine closure. Production rate and cut-off decided by the mine management, will affect the projects value, usually calculated as the net present value. (Hall,2003) illustrates the value-cut-off-production rate relation in the three dimensional graph illustrated in Figure 11, and how both cut-off and production rate must be optimized together to climb the hill of value.

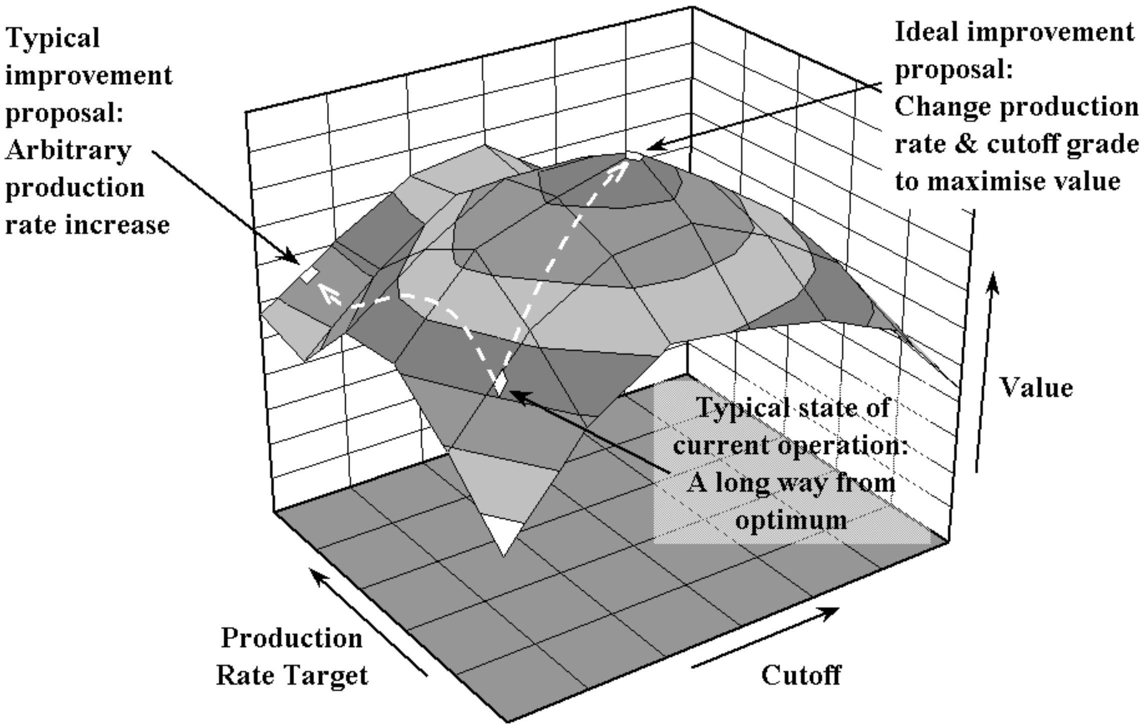

Figure 11 Finding and climbing the hill of value. Illustrating the relation between project value, cut-off and capacity. **(Hall, 2003)**

5.3 Dillution's effect on grade

Dillution influences the cut-off in narrow vein mining because you intended or unintended mine material with less or no minerals at all. Mining material below the economic cut-off limit can be critical for a marginal project, because treatment cost factor xh (production stoping and processing) increase by the dilution percentage. The income component $(p - k)xyg_h$ is not directly affected by dilution, as the tonnage x increase by the same factor as head-grade g_h decreases. Problem occur when treatment is limiting, meaning that process plant or mining equipment has a limited capacity. The outcome in this case will be the same annual production as for no dilution, producing less minerals, requiring more years to produce the reserves. This can be illustrated by an example: Reserves for one stope is estimated to be 1000 tonnes at 1,4%Cu with thickness of 2m. Stope and drift must be widened 0,3m on each side giving an intended dilution of 23%. Hangwall and footwall dilution skin contain 0,1%Cu. Hangwall caving and drilling deviation cause an unintended dilution of 7%. The outcome will then be 1300tonnes with head grade g_h=1,107%Cu.

5.4 Dilution in narrow mining

Dilution in narrow vein mining is a serious concern that may cause dramatic head grade drop for the mined ore. Dilution can easily occur because the ore itself is so thin, that excavation may intended or unintended include country rock from hangwall or footwall. The intended dilution, which we can calculate in advance, may be drifts dimensioned for machine width in a part of the ore where thickness is less than drift width. This dilution can be reduced by utilizing special narrow mining equipment and carefully plan drifts to follow ore boundary (Finkel, et al., 1993). The unintended dilution, may be caused by long hole drill holes deviating into the country rock or hangwall caving in the stopes. Longhole deviation can be reduced by using modern longhole rigs, analysing drill cuttings to verify that drilling actually takes place in the ore. However, the safest way to reduce risk of drill hole deviation, is reducing sublevel interval, making shorter drill holes. Dilution from hangwall caving can be reduced by cable bolting the hangwall from each drill drift or leaving more pillars. For high grade ores, the additional cost of cable bolting can easily be defended by the value of each stope, but for low grade ores, leaving pillars may be more economic. This question is discussed in more detail with examples from the Viscaria mine (ch. 6.2.3 p.43).

When evaluating Nussir mine design, sublevel interval pops up in several occasions as a key parameter for stability, cost, production rate, dilution and turnover. Higher sublevel intervals, meaning fewer drill drifts, will cause a lower operating cost, as stoping is a cheaper way to extract ore than drifting, but drill holes would have to be longer, thus more dilution caused by deviation. More drill drifts will also allow denser cable bolting of hangwall, reducing the unsupported span.

5.5 Nussir dilution estimates

Dillution must be included in the calculation of probable reserves, because every past experience with narrow ore mining point to the fact that dilution will occur (ch.6.2 p.37). What we know about the Nussir deposit from drilling is that the nature of the mineralisation, make mineral grades vary along ore thickness. A general trend is identified, being increased Cu-grade in the middle of the mineralised horizon, with decresing Cu-grade toward footwall and hangwall. High Au and Ag grades are normally located in the middle or just below the middle of the mineralisation and high Pd and Pt grades usually occur along footwall, just below the high Cu-grades (Sletten, 2011). Thickness of the minable area covered in this report, varies between 1,43-6,2 with an average of 2,9m. Mineralisation regularity have been described as very smooth and straight, being a sediment horizon (Sletten, 2011, ch. 2). Wireframe model of indicated resources also present deposit surface as smooth. Even if sampling density is only 150-250m, implying that deposit may be bulky between holes, I choose to consider current model as realistic for ore boundary in the following estimations.

With the current knowledge and data, I choose thickness as the resource variable affecting dilution, given that sublevel interval and mining equipment remain the same for all stopes. The relation will be given by the curve and equation in Figure 12 below.

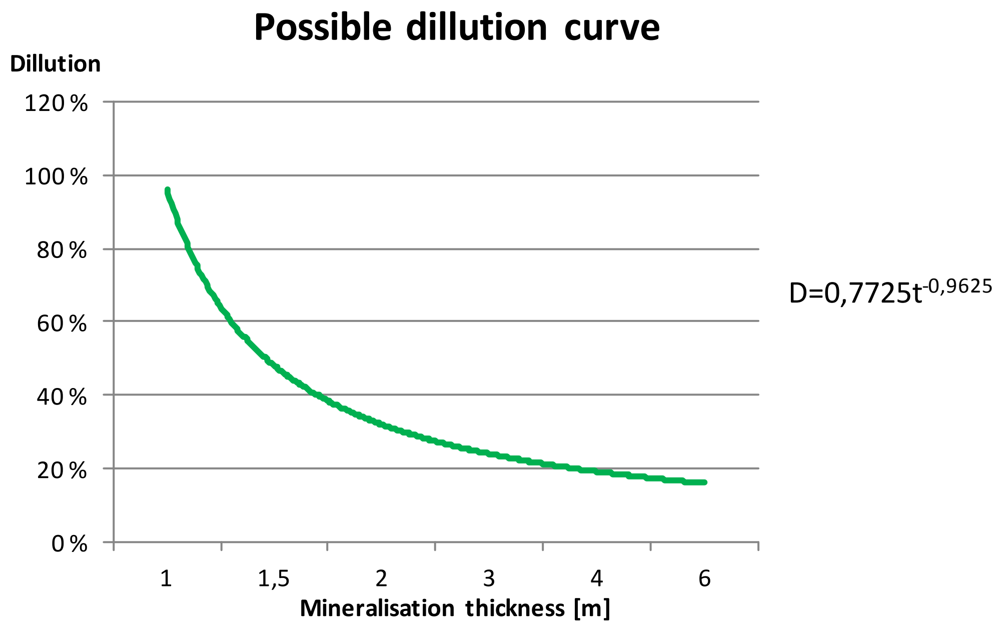

Figure 12 Imaginary relation between mineralisation thickness and dilution, given that sub-level interval and mining equipment remain the same for all stopes.

Dillution equation:

$$D = 0.7725t^{-0.9625}, \qquad t = \text{thickness} \qquad 5.3$$

Calculated resources in each stope, will be added dilution. This means that the tonnage per stope will increase by the dilution percent from equation

$$C=(p-k)xyg_h - xh - m - f\tau \qquad 5.1:$$

$$Tonnage = Tonnage_{\mathrm{mod}el}(100\% + D) \qquad 5.4$$

Head grade will be given by:

$$g_h = \frac{g_{\mathrm{mod}el} + Dg_{dillution}}{100\% + D} \qquad 5.5$$

Dilution grade will be set to be 0,1%Cu, 1ppmAg and 0,02ppm Au, based on a simple drill sample observation.

6. Mining methods

6.1 Underground Mining methods

6.1.1 Sublevel open stoping

Sublevel open stoping (SLOS) is used for mining mineral deposits with; steep dip where the footwall inclination exceeds the angle of repose; stable rock in both hanging wall and footwall; competent ore and host rock; and regular ore boundaries. SLOS recovers the ore in large open stopes, which are normally backfilled to enable recovery of pillars.
The orebody is divided into separate stopes, between which ore sections are set aside for pillars to support the roof and the hanging wall. Pillars are normally shaped as vertical beams, across the orebody. Horizontal sections of ore are also left as crown pillars (Hartman, 1987).

Miners want the largest possible stopes, to obtain the highest mining efficiency, subject to the stability of the rock mass. This limits their design dimensions.

Sublevel drifts are located within the orebody, between the main levels, for longhole drilling of blast patterns. The drill pattern accurately specifies where the blastholes are collared, and the depth and angle of each hole. Drawpoints are located below the stope to enable safe mucking by LHD machines, which may tip into an adjacent orepass, or into trucks or rail cars.

The trough-shaped stope bottom is typical, with loading drifts at regular intervals. Nowadays, the loading level can be integrated with the undercut, and mucking out performed by a remote control LHD working in the open stope. This will reduce the amount of drift development in waste rock. Sublevel stoping requires a straightforward shape of stopes and ore boundaries, within which only ore is drilled. In larger orebodies, modules of ore may be mined along strike, as primary and secondary stopes.

6.1.2 Sublevel up-hole benching

Sublevel benching is a variant of sublevel stoping, where ore is extraxted bench by bench rather than stope by stope. Benching comprise of uphole drilling starting at the end of the drift, retreating backwards to the ramp access. Ore must be loaded from the drift by remote controlled loading as oppose to sublevel stoping where ore blasted from multiple drifts fall into draw points at stope bottom. Benching can progress top down or bottom up. Top down does not require special arrangements for leaving pillars. Bottom up bench stoping require sill pillars between every stope as a working platform for the next level (Lappalainen, 2012). It is possible to do double benching also, stoping from two drifts simultaneously by up-hole drilling (see Figure 2 on page 6).

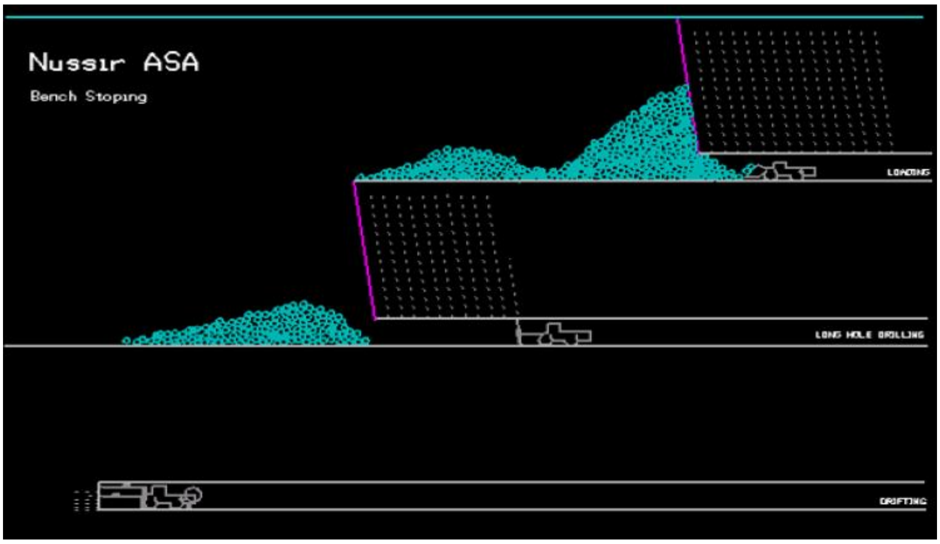

Figure 13 Illustration of top-down bench stoping production cycle

6.1.3 Cut and fill

The higly selective underground mining method cut and fill is commonly applied for steep, narrow, high grade mineral deposits with undulating boundaries. It is normally an overhand mining method in which horizontal slices of ore are excavated in the stope and replaced with waste as fill. The filling is part of the mining cycle as oppose to back filling after stope is completely excavated (Hartman, 1987). Method can be applied in weak rock conditions due to the fill support, and pillars may not be necessary at all, implying very high recovery.

6.1.4 Narrow vein mining

Special methods and variants of the methods described above, can be used to mine steeply dipping narrow ore bodies. The term vein is commonly used in addition, because narrow deposit are usually associated with a vein mineralisation, but as for the case of Nussir a narrow mineralisation can also be a thin sedimentary horizon. Definitions vary, but (Finkel, et al., 1993) define narrow ore bodies to be less than 4m thick. Prior to the development of mechanized mining in the 80's, steep narrow deposits, where mined by labour intensive methods with handheld equipment, if mineralisation was rich or labour was cheap. Methods relied mainly on timber sets to support the walls during mining, but the technique of backfilling started developing when material handling became more efficient, also known as the cut-and fill method. In high labour cost countries, mining engineers saw the need for improved capacity and efficiency to economically mine steep narrow ore bodies, but the dimension of available equipment where so large that drifts and stopes would have to be made wider than the mineralisation (Finkel, et al., 1993).

Modern mining methods for steep narrow ore bodies, aim to mine selective, cost efficient and safely, by utilizing small mechanized equipment. Stability is either achieved by back fill or pillars. Retreating methods allow stopes to deform after extraction, as long as the overall stability of the mine is maintained. Including cut and fill described in 6.1.3, there are a variety of possible mining methods, unique to each deposit.

6.2 Narrow mining case studies

Mining is a practical discipline, theories can only be used as principles and guidelines. Every deposit is unique and so is the mine producing from it. Three different case studies will therefore be presented, to gain background knowledge about narrow ore mining: An extensive research project from Zinkgruvan, a field visit to Zinkgruvan and experiences from Viscaria.

6.2.1 Zinkgruvan research project

An extensive research project was carried out in the 80's by Finkel, Olson and Thorshag, to demonstrate by full scale trials at Zinkgruvan, the technical and economical feasibility of applying mechanised sublevel stoping and cut and fill for mining steeply dipping narrow ore bodies. Previous mining practices of narrow ore bodies, where labour intensive, with low production and excessive dilution, reducing value of the product. Together with partners Atlas Copco, Sandvik Rock Tools, Boliden Mineral, Nitro Nobel and SveDeFo, test mining was carried out on a selected block in Nygruvan, of 50m height and 150m length. The area is characterised by distinct layering in both strike and dip directions with distinct ore boundaries.

The zink, copper, lead, silver mine in the southern part of the Bergslagen region of central Sweden, can be looked upon as a reference mine for studying mining of steeply dipping narrow vein ore bodies. The Zinc-rich ores at Zinkgruvan, consist of sphalerite and galena, occurring as stratified, calciferous, leptite impregnations. Grades vary from 6-10% Zn, 1,5-5% Pb and about 45g/tonne Ag. Ore body occur in a 5-25m thick stratified zone in the upper part of a meta-volcanic sedimentary group. The geometry of the ore body is a 5m long tabular body, known to a depth of 1500m (Lundin) with a thickness varying between 0,5 and 10m. Strike is east-west, dipping 70-75^{o} north, and the orebody is intersected by a north-northeast subvertical fault system dividing mining area into two blocks, the Knalla mine in the west and Nygruven to the east.

In rock mechanical terms, the area chosen for test mining, is dominated by good rock qualities. Hangwall consist of a homogeneous gneiss-leptite of good strength. The ore is compact and of continuous and regular mineralisation. The ore block above and to the side of the test area, where mined out. The stress state is therefore assumed to be representative of that of a typical mining block. The maximum principal stress is horizontal, oriented perpendicular to strike, and of relatively high magnitude (Finkel, et al., 1993).

Table 10 Factors for the test mining stopes of Nygruvan in the 80's, Zinkgruvan (Finkel, et al., 1993)

Mining factor	Values
Mining method	Sub-level stoping and cut & fill
Sublevel intervals	13m
Principal stress	Horizontal, high magnitude
Direction of principal stress	Perpendicular to strike
Jointing	Low
Stope height	50
Stope length	150
Depth below surface bottom level	496m

Mechanisation and thereby choice of equipment was vital for the feasibility of mining the selected stopes. The small drill rig, Tracker 526, only 1,1m wide and 1,85m high had to be developed by Atlas Copco for the purpose of narrow mining. Equipped with a 38mm drill bit, one could drill holes, charged with ANFO, small enough to avoid damages to rock walls caused by blasting and thereby reducing dilution. The rig was later rebuilt for 38mm longhole drilling, stoping by upward drilling. Planning was essential for accurate longhole drilling, achieved by continuous mapping of ore contours. A geologist recorded the roof, and drillers recorded the face after each blast. Each row of holes consisted of 3 blast holes with 0,6-1m spacing, burden set to 0,8m and contour holes placed 0,1m inside ore boundary to avoid damage to the waste rock in the hangwall and footwall. Normally drilling was stopped just before breakthrough in the level 13m above. A few holes were drilled trough to check accuracy. Hole deviation registered afterwards, proved that only 2 of 50 holes drilled where not straight enough for a flashlight to be seen.

Charging and blasting was also carefully planned. 3 to 4 rows were blasted at one time, using a 3g/m detonation cord, which had the effect of reducing ANFO detonation velocity, lowering the explosion pressure inside the borehole. A relatively high charge density of 4kg/m^3 was recorded, partly due to the bottom slice which was drilled and blasted with 51mm holes, while retaining the same pattern as for 38mm holes. The walls in the mined out stopes where smooth and fragmentation excellent.

Loading in drifts and from draw points at the main haulage level, was carried out by Toro 150 E LHD's. With a width of 1,4m, 0,3m wider than the drill rig, it became the limiting factor for drift width. If a more narrow LHD had been available, even lower dilution could be achieved.

The results after mining the selected stope by both sublevel stoping and cut and fill, where satisfying and very close to expectations. Results from the actual report are presented below:

Table 11 Sublevel stoping, comparison of expected results and actual outcome. (Finkel, et al., 1993)

Item	*unit*	*Expected result*	*Actual utcome*
Ore recovery	ton	28 600	29.730
Advance	m/mansh	1.4	0.7
Productivity	ton/mansh	22.2	25.4
Mining cost	SEK/ton	208	214
Waste dilution, tot	percent	23	25
Ore losses	percent	25	10

Table 12 Cut and fill, comparison of expected results and actual outcome. (Finkel, et al., 1993)

Item	*unit*	*Expected outcome*	*Actual rresult*
Productivity	ton/mansh	11.5	10.5
Mining cost	SEK/ton	329	316
Waste dilution	percent	10	31
Ore losses	percent	10	4

As seen in the tables above, sublevel stoping proved to be more economically feasible than cut and fill, as mining cost is registered to be 208 SEK/tonne as oppose to 329 SEK/tonne and productivity is more than twice as high. Waste dilution is registered to be higher for cut and fill, but ore loss is lower. Important remarks regarding dilution is that development drifting contain 45% waste, while long hole stoping only gave 5% waste rock, making total dilution 25% for sublevel stoping.

6.2.2 Field visit to Zinkgruvan

Author of this report made a field visit to Zinkgruvan, currently operated by Lundin Mining on the 15th of March 2012, to gain experience from their current narrow mining operation of the Cecilia orebody, discussing technical solutions with mine planning chief Jouni Hansen-Haug.

Total production of 1 million tonne ore at Zinkgruvan was reached in 2011, with aims of even higher production for 2012, as their new ramp access allow higher capacity and flexible logistic for the mine (Lundin). Current mining operations are divided in three separate, steeply dipping ores: The thick Burkland ore representing over 50% of planned production, the large plate shaped ore Nygruvan, and the small narrow Cecilia orebody 3 km south of Nygruvan (Figure 14), targeted for investigation due to its similarities with Nussir.

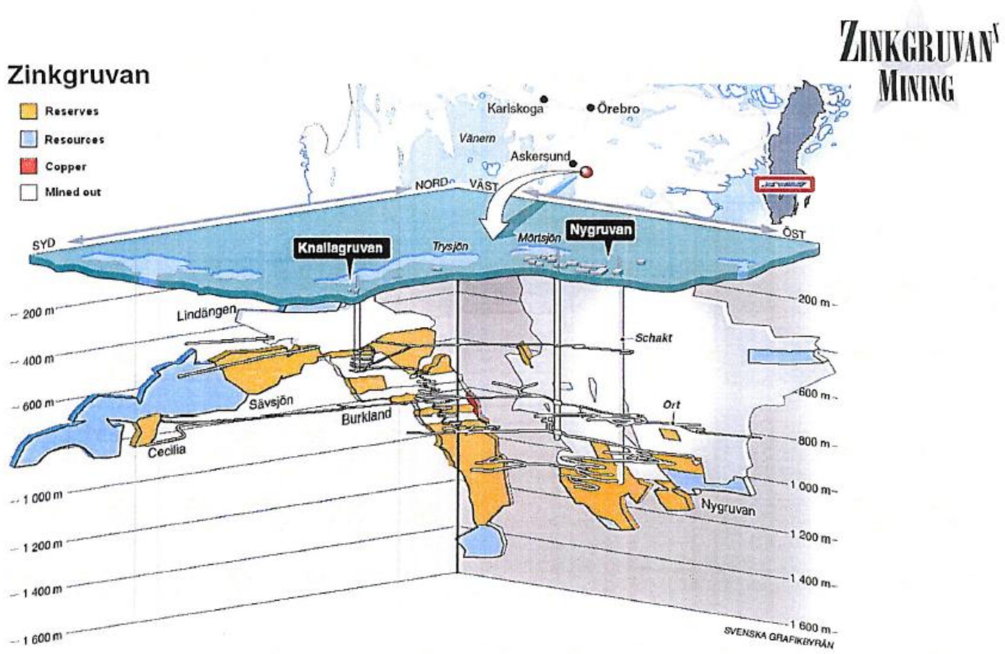

Figure 14 Overview of Zinkgruvan resources and reserves. Current narrow mining operation is in the yellow reserves named Cecilia. Narrow test mining in the 80's happened at level 496 of Nygruvan. *Zinkgruvan Mining*

The reserves of Cecilia ore, make up a tabular plate shape of 350 height, 360m length along strike, dipping 70^{o} and thickness varying between 3 and 5m. Mining of Cecilia recently started at level 650m and will advance upward by single lift benching, referred to as Avoca mining, involving bottom up mining, with lateral drill drift development along strike from a ramp to both directions until the cut-off boundary (Figure 15). Drill drift interval (sublevel interval), is 17m exluding drift height of 4m, to assure accurate up-hole drilling and the possibility to check deviation by drilling into drift above. Stopes of 40m length, 17m height, are blasted in one go, by 16 blast hole rows, 2,5 m apart, leaving rib pillars of 5m width between each stope. A relatively high mining cost is due to the robins slot hole necessary for blasting the next stope after a rib pillar. Ore from both drift development and stoping is loaded and transported to an ore pass by CAT and TORO LHD's.

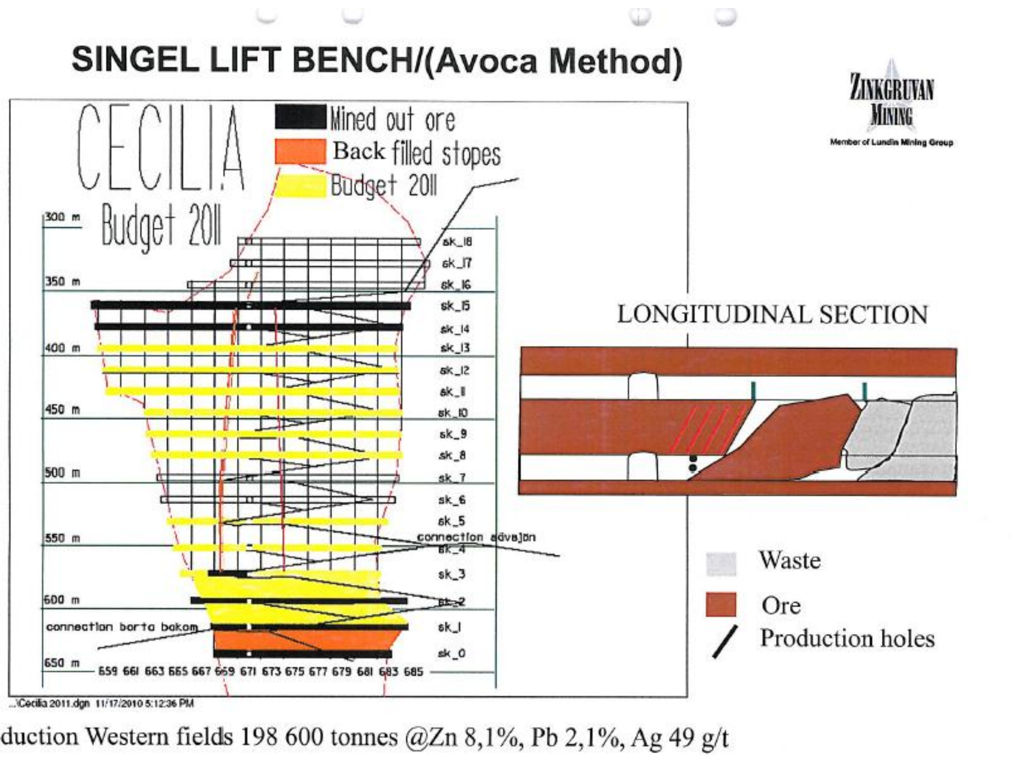

Figure 15 Mine layout for Cecilia ore body at Zinkgruvan. Ore thickness is 3-5m, 17m sublevel interval. *Zinkgruvan Mining*

Geotechnical conditions are dominated by strong (UCS=200Mpa), but very brittle rocks (Hansen-Haug, 2012). Stress measurement from the deeper parts of the Burkland ore are reported to be σ_V=0,028Z, σ_h=0,047Z and the maximum stress component σ_H=0,068Z (all given in MPa), oriented in a E-W direction. For the Cecilia ore, this means a maximum stress component being roughly 44,2Mpa oriented horizontally normal to strike. The high stresses combined with the brittle nature of the rock, may cause serious collapse in the mine, in case of earthquakes, heavy blast vibrations or pillar collapse (Hansen-Haug, 2012). Seismometers are therefore installed throughout the mine for surveillance and stopes are backfilled with a mixture of cement and waste rock.

Being a steeply dipping (70°), narrow ore deposit (3-5m), ore boundary control at Cecilia, is vital for keeping dilution low. Otherwise will occupy limited haulage capacity, which is already associated with long distances and high costs. . Zinkgruvan have established a good practice for mining, where drill drifts are carefully planned in accordance with mineralisation. For each blast section, geologist sketch the hangwall and footwall lithology, to identify whether drift excavation has gone too far into country rock or if mineralisation still remain in the drift walls. From this, dilution and ore loss is registered, and the next blast is adjusted to match ore boundary. Drifts will then vary in width, to match ore thickness exactly. Luckily, ore thickness is the limiting factor for drift width, not machine width, which was the case in Olson and Thorsag's narrow mining trials in Nygruvan. When drifts are excavated to the end of economic mineralisation at several levels, accurate up-hole drilling can be done based on data collected from drifts.

6.2.3 Viscaria mine

A mining operation worth mentioning, when discussing narrow mining, is the Viscaria copper and gold mine in northern Sweden, operating between 1983 and 1997. The sediment hosted deposit occur as a copper rich graphite and copper rich limestone with thickness varying between 5 and 30m with an average width of 10m (Mäkinen, et al., 1987). The dip also varies between 80° and 90° in the northern area, and between 70° and 80° in the souther area, where dip declines to 45° below 200m depth. Ground conditions vary throughout the mine for different lithologic units. Strongest are the greenstones with measured compressive strength (UCS) of 225MPa. The graphite schist often occurring along hangwall have a measured compressive strength of 100MPa. In 1987 the production was reported to be 1,3Mt at 2,5% Cu with 250 employees (Viscaria,1987)

Viscaria deposit was primarily mined by tree variants of sublevel stoping, where sublevel intervals are adjusted according to dip, thickness and ground conditions (Viscaria, 1987). As seen in Figure 16A, near vertical parts of deposit being 10m thick was efficiently exploited by 30m longholes both upward and downward. When dip is below 60° illustrated in (C), one needed twice as many drill drifts for cable bolting purposes. (Mäkinen and Paganus) predicted that a hangwall collapse in stopes dipping less than 60° with 2 - 3,5%Cu would cause greater economic loss than the additional cost of an extra drift-level and/or cable bolting. They also calculated backfilling to be profitable in the high, near vertical stopes (Figure 16A), if grade was above 4%Cu.

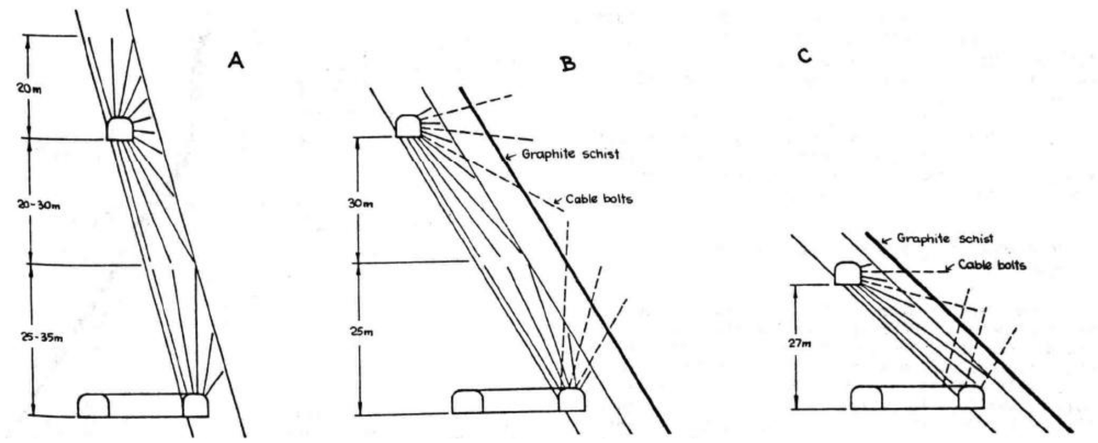

Figure 16 Sublevel stoping methods applied to 10m thick Viscaria deposit in the 80's. Sublevel interval is consequently decreased for declining ore dip, seen in (B) and (C). Unfavorable ground conditions caused by graphite schist and low dip require additional cable bolting for support of hangwall (Mäkinen, et al., 1987).

An other method described in a paper from 1997 by (Marttala) is called the top slicing method, applied to mine parts of Viscaria deposit less than 4m wide. Method is similar to up-hole single benching with top-down progression. Sill pillars are left between every sublevel and rib pillars are left along strike, drilling 25m upward from each drift level. Relatively high production capacity is achieved, by 5m wide drifts, with loading drifts every 50m to the hangwall side of drifts, allowing space for a 10t remote controlled LHD and 30t highway trucks into the drift. Drift dilution is in the order of 25-50%, but (Marttala, 1997) express that drift face where some times blasted in two operations to separate ore from waste. Yet, only practically doable in the case of near vertical ore boundaries and plenty of time to do the extra job.

6.3 Choosing mining method

Initial evaluation of possible mining methods for Nussir copper deposit, is not necessary for this study, but rather a detailed assessment of mine design parameters for the selected method, and establishment of known narrow mining practices. Previous assessment of Nussir mining methods, carried out by (Golder, 2009), identified the more likely and favourable mining methods, based on empirical selection methods. From UBC mining method selection, cut and fill came out as the most favoured option, followed by sublevel stoping and shrinkage stoping. Cut and fill method is proven favourable because of the underlying parameters of the selection method; geometry, thickness, plunge and rock mass quality, but it is not favourable when considering the value of mineralisation in relation to method costs. With the current level of copper price being 6600$/t and the average copper equivalent grade (Cu grade+ Ag and Au grade in price relation to Cu), being 1,45%Cu for the targeted areas, each tonne of ore has an approximate value of 400NOK in sales revenue. We can see from a first glance that ore value is not high enough to defend cut and fill, which is a high operational cost method. Shrinkage stoping, is not considered as a possible method due to hazardous working conditions occuring when the working platform in the stope is a pile of crushed ore.

Top-down bench stoping was considered for Nussir by (Smeberg, et al., 2011) for its ability to open up large stopes without the risk of blocked production due to hangwall breakings, but ramp access to the top requires high investments before mining can start. Method might be more suitable when advancing in depth. Bottom up double bench stoping, leaving sill pillars every 50m vertically have also been considered during the thesis, for its ability to mine safely in small stopes being 40m long, 44m high. However, this method require 12 drill drifts, drilling upward only, increasing mining costs.

The large 300x350m stope layout with evenly distributed small pillars, suggested by (Smeberg, et al., 2011), was considered as an alternative for it's low development and drift costs, but later rejected after studying narrow mining practices. The 300m high stope design, without any sill pillars in between, involve high risk and low flexibility.

Sublevel stoping is hereby the favourable mining method for Nussir deposit, as it can easily be adapted to the narrow steep geometry, with lower operational costs and higher capacity compared to cut and fill and bench stoping. Progression should be bottom up, with 40m sub-level intervals, drilling 20m updip from bottom drift and upward and downward from higher drifts.

7. Nussir mine design

7.1 Defining minable areas

The absolute outer extent of the Nussir mineralisation is the geometric model defined as a plate shape wireframe modelled from bore hole intersections with geological cut off of 0,3% Cu. Within this interpreted body of mineralised material, indicated resources are defined to be areas covered by sampling in a grid with no more than 250x250m (Wheeler, 2011). Indicated resource is the minimum classification criteria when considering extraction and defining reserves (Figure 1, p.3) (JORC, 2004).

Besides, classification criteria, there are modifying factors affecting the definition of minable areas for this mine design assessment. Environmental and legal considerations, limit any road access or other development on the mountain plateau along the mineralisation outcrop. By this access must be from east along the fjord, limiting mining of the indicated resources further west at this stage. By this, minable areas are limited to the areas marked by grey in Figure 17.

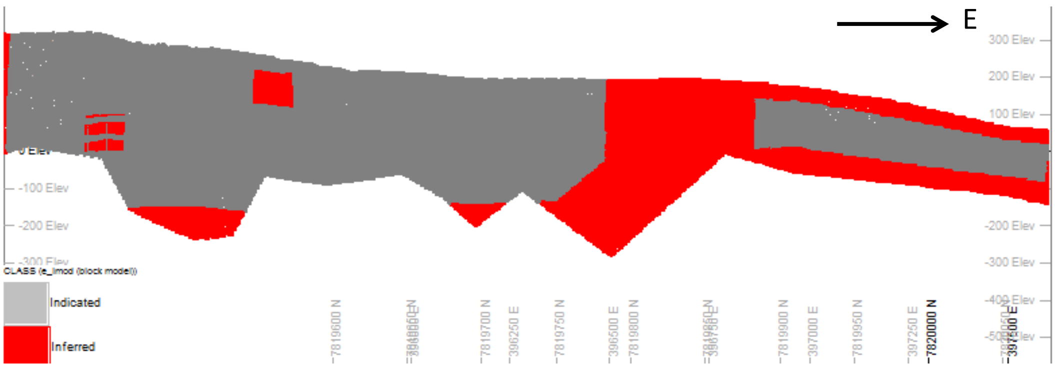

Figure 17 Indicated resources in east (grey). Inferred are (red). Block model seen from south.

The third, criteria for selecting minable areas is the economic cut-off. This requires a cash flow analysis based on mine design costs, which I do not know until the design is made and evaluated for a chosen part of the mineralisation. An initial 0,9% Cu cut-off is therefore chosen to define areas for mine design layout. This layout may need a change, based on a new cut-off arising from mine design's incurred cash flow. The size, location, grade distribution and cut-off grade curve for the minable areas is presented in the Figure 18 and Table 13 below:

Table 13 Resources in minable areas from block model. (Wheeler, 2012)

TONNES [t]	AG [ppm]	AU [ppm]	PD [ppb]	PT [ppb]	CU [%]	THICK [m]	Cu_equiv [%]
3 500 266	14,36	0,13	54,95	82,04	1,21	3,27	1,45

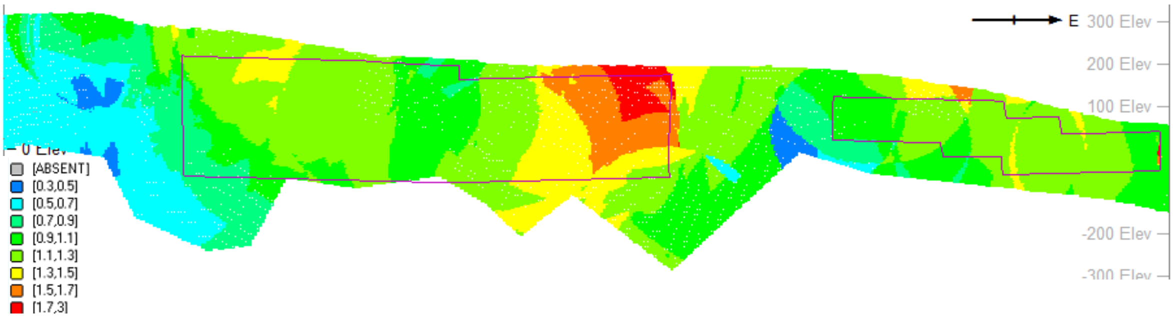

Figure 18 Minable areas chosen from Adam Wheelers block model, defined as indicated resources > 0,9% Cu

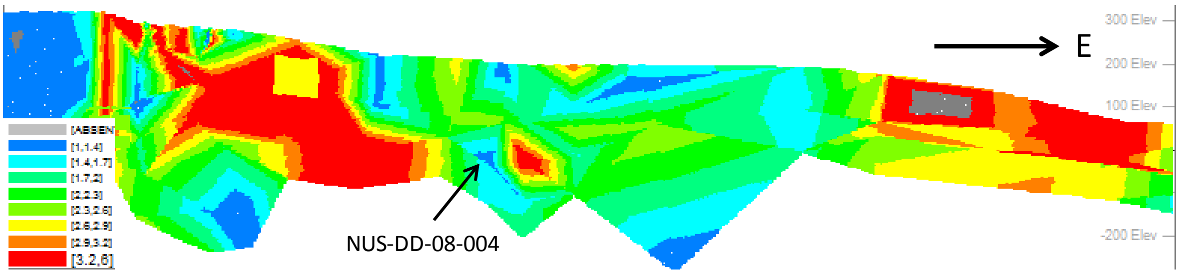

Figure 19 Thickness estimation. Error introduced by marked borehole not fully intersecting deposit. (Wheeler, 2012)

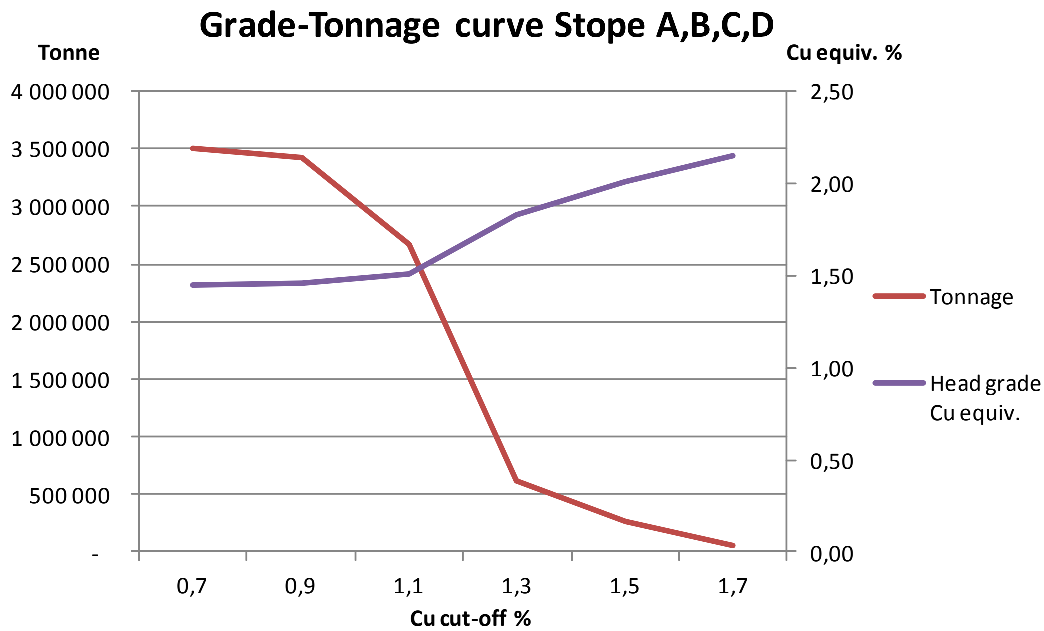

Figure 20 Grade- tonnage curve minable areas. Cu equiv. is Cu + Au and Ag grades in price relation to Cu

7.2 Modelling stopes

Given the stope stability considerations in 4.3 p.21, the pillar size considerations in 4.4 p.25, the mining method selected in 6.3 p.45 and the minable area chosen above, a stoping layout have been made. Initially, drill drifts string lines were constructed in Datamine with 40m sublevel interval and trimmed to and 1:40 incline from planned ramp location. Parallel strings where constructed between drill drifts dividing up-and-downward stoping sequences. All strings where then projected on to geometric model hangwall and footwall in vertical view, and then trimmed to rib pillar outlines in horizontal view. Each hangwall and footwall pair of strings, 40m long, where then connected, making closed stoping outlines on top of each other in dip direction. Wireframe stopes where generated by linking outlines to make a closed wireframe volume. Yielding pillars in stope centre, 5m wide, 10m high, where created by extruding the pillar outline trought the stope, separating the rib pillar volume from the stope volume. This step did not work entirely according to plan, as the excavated stope wireframes where no longer closed volumes, which may have introduced some minor errors in the block model stope evaluation.

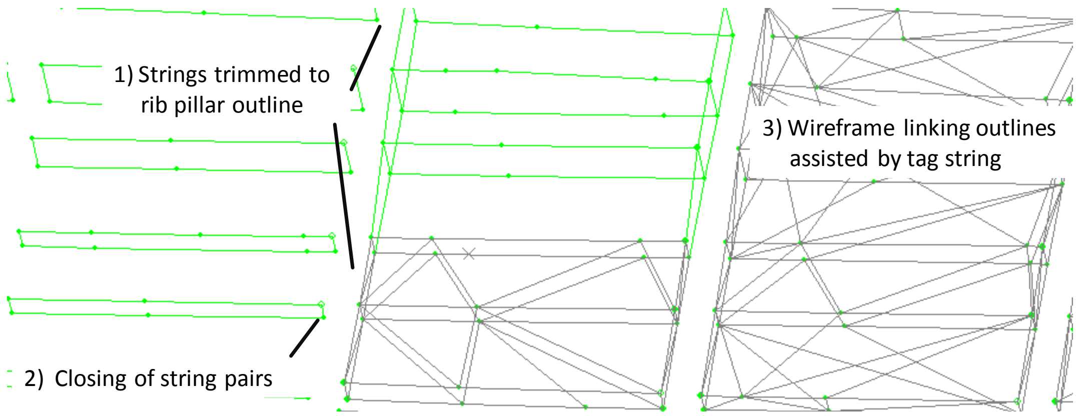

Figure 21 Stope wireframing workprogress

Four main stoping sections where created, A and B in the east and C and D located further west. Sections A and B lie on each side of a planned ramp, with a 10m wide internal pillar separating them. Similar case for sections C and D. Each section is divided into 40m long stopes separated by 5m rib pillars. In section A and B, stopes are divided vertically by a 8m high sill pillar. For section C and D the case is similar, but with two sill pillars, making three stoping levels vertically. Stope and pillar design is illustrated in Figure 22.

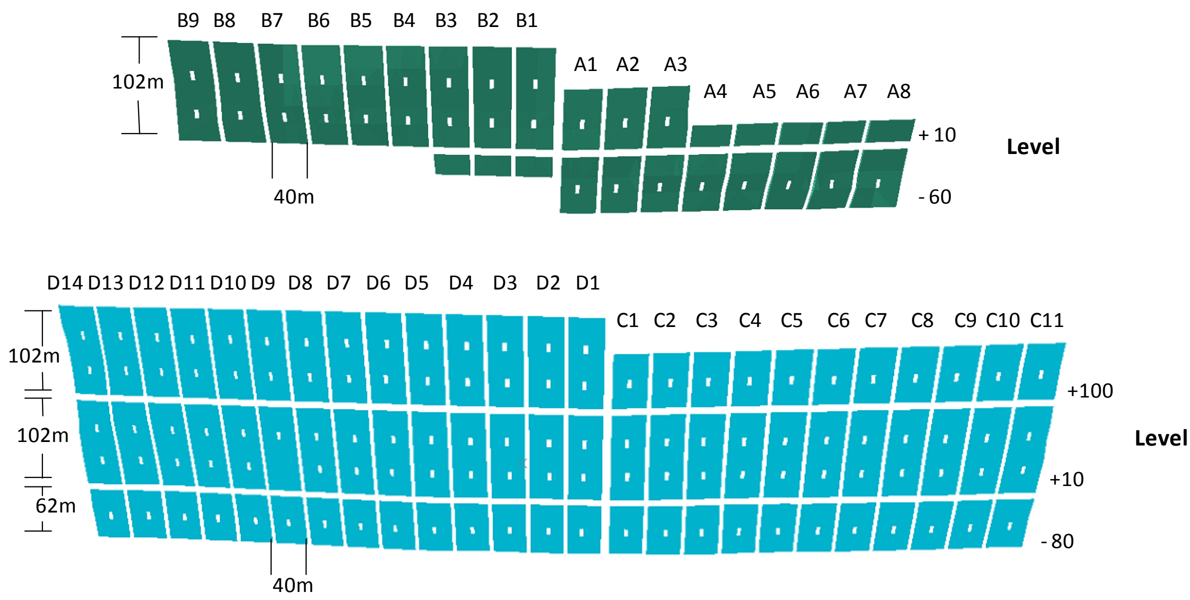

Figure 22 Nussir stope layout of the minable areas from Figure 18 seen from south. Stope ID comprise of stope letter, number and level at which stopes are loaded. Rib pillar width is 5m, Sill pillar height is 8m.

7.3 Mine descriptions

7.3.1 Access tunnel

Stoping will start at +10 level between stoping section A and B. This allows quick access from the main road along the fjord, where topography is low. Access tunnel will be 8,5m wide and 6m high, face area 50m^2, giving sufficient space for conveyor belt, air ducts in the roof, water pipes and large vehicles. Two alternative portal locations have been discussed in Nussir: From the little valley of Dypelv, 1km northwest of Øyen industrial area, requiring a 590m access tunnel, or a 2000m long tunnel from flat area in front of a steep rock face formed by a previous quarry operation – about 60m above sea level in the vicinity of the processing Plant at Øyen. Pros and cons for the longer tunnel is that conveyor belt will be little affected by snow and ice during winter if underground all the way, but development cost become high and access to mine will be granted later. Final decision depend on location of processing plant, but the shortest alternative will be used in this report because it's reducing capital costs.

7.3.2 Main haulage level

From the access tunnel, a 50m^2 haulage tunnel will be driven at -10 level with a 1:40 incline westward along the mineralisation footwall to the western end of stoping section B, length is 410m. Haulage tunnel need only be 135m eastward from the access point with 30m^2 face area, as this way is approaching a dead end, in terms of current block model. Placed 30m away from the mineralisation, and reinforced by shotcrete and bolting, stability will be granted during and after production of the stopes. Draw points every 45m along strike, provide loading access into each stope at +10 level.

Some time before stopes A and B are mined out, haulage tunnel will progress westward along strike, declining 1:22, reaching the eastern end of stope C after 385m. From here, the 1100m long haulage tunell for stope C and D will be made at level -10. The combined access and haulage tunnel is the largest vein of the Nussir mine, containing extended conveyor belt to stope C and D, ventilation and access. In addition, it provides perfect location for probe drilling prior to stoping.

7.3.3 Drifts

The long deposit geometry require drill drifts to be very long, saving ramp developments along strike. Drill drifts are drilled and blasted with 40m vertical spacing, from the ramp towards east and west, allowing two faces to progress on the same level. Drift dimensions will vary by deposit thickness and drift function. Drifts at stope bottom, without haulage tunnels, will be 5m wide and 4,5m high allowing space for haulage trucks. Sub-level drifts will have minimum 4,5m width allowing space for horizontal drilling of cable bolt holes, or dimensioned to deposit thickness if this is thicker, illustrated in Figure 23. Loading drifts will be established every 100m in footwall for truck loading. These will also be used for probe drilling to prove grades and boundaries. Drifts should anyhow be placed along hangwall contact with angled wall, otherwise stresses are likely to concentrate in uneven areas of stope hangwall, causing failure and consequently increasing risk of potential caving (Lappalainen, 2012). Sublevel interval is given by the stability considerations in (ch.4.3 p.21) and the dilution considerations in (ch.5.5 p.32). Sub-level interval may change after dilution and stability is registered. Drift key numbers are given in Table 14.

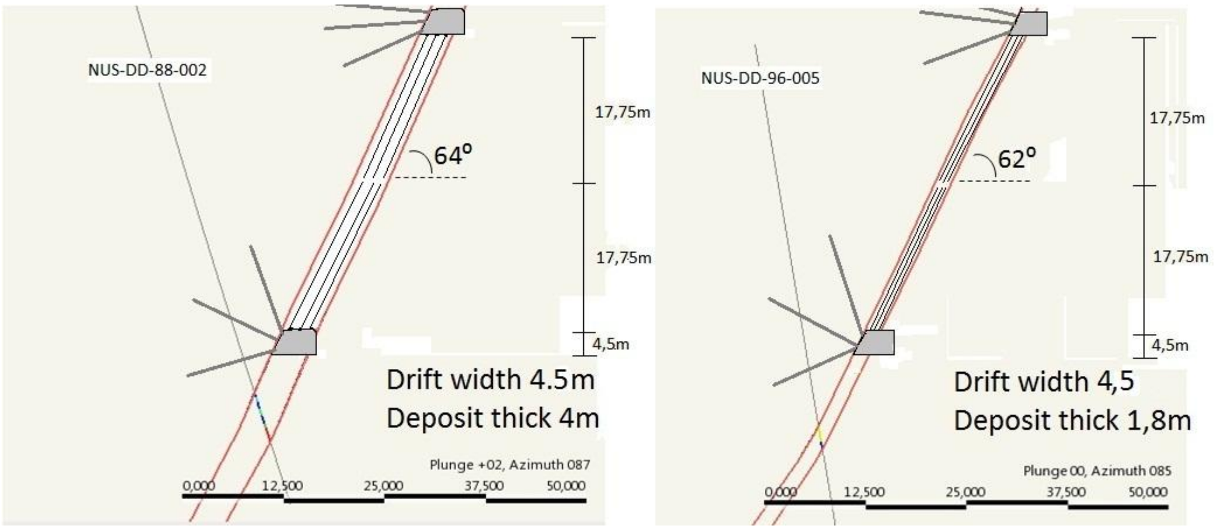

Figure 23 Drill drift illustration in wide stopes to the left and narrow stopes to the right. Cable bolting of hangwall (thick grey lines) and production long holes illustrated.

Table 14 Mine design drift dimensions and calculated key numbers

Drift dimensions		
	Width	**Height**
Drill drifts	4 m	4,5 m
Combined drift and loading tunnel	5 m	4,5 m
Drift key numbers		
	Stope A & B	**Stope C & D**
Total drift length	2612 m	8684 m
Driftmeter/undillluted ore tonne	0,0036 m	0,0049 m
Undillluted ore production from drift	16 %	17 %

The driftmeter per undiluted ore tonne and undiluted ore production from drifts is slightly higher for stoping section C&D, even though stopes are larger, with less drifts per height along dip than stoping section A&B. The cause is due to very narrow stopes in the eastern end of stope C.

7.3.4 Ramps

A ramp ascending with a gradient of 1:7 will be driven from the Haulage level to level 90 and down to level minus 60 in stope A and B. Ramp at C and D stopes will be driven from haulage level at minus 10 level to 180 level and down to minus 80 level, illustrated in Figure 24. Ramps will give access to the various mine levels. The cross section of the ramp will be 30 m^2. From the highest level in the mine a ventilation raise will be mined to the surface. The raise will also form an emergency escape way. The ramps will be established along the strike with a distance of 1300m. Each ramp will give access to two stopes on each side of the ramp. Ramp in CD stopes noes not have equally the same amount of stopes on each side, because stoping section C, will be extended eastward by 135m when resources are better sampled from more drilling.

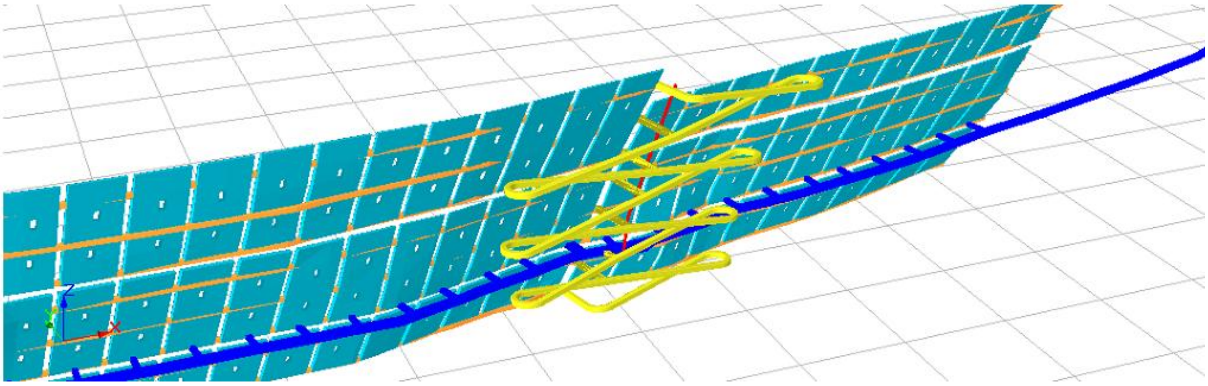

Figure 24 Ramp DC 1:7 gradient, 30m2 face area, providing access to drill drifts

7.3.5 Slots

Slot holes will be driven upward and downward from drill drift to establish a free face for blasting after each rib pillar. Slot holes of 20m length will be excavated by raise boring machine. Reaming holes surrounding the slot hole will be drilled and blasted first to open up the stope without damaging pillars, before the rest of the stope is blastet.

7.3.6 Stoping

The 100 meter high stopes will be exploited from 3 drill drifts every 40 meter vertically drilling 20 m long holes upward from the bottom level and both upward and downward from the sub-levels. Ore from A and B stopes at 10 level and C and D stopes at minus 10 level will be loaded from haulage tunnel. Stopes above and below the haulage level, separated by sill pillars, will be loaded in stope bottom trough the combined drill and loading drift. Drill holes will require careful positioning and high quality workmanship. The stopes will have an average width of 3 m. A cut-off practice, assesing both thickness and grade, will determine if stopes are to be mined, or if mining practices have to be adapted.The drilling pattern and the drill hole positions will be planned and set out by the mine. The drill will be equipped with an on-site sampling and analysing system rendering information direct to the driller. Pumped slurry explosives will be used for blasting. When the drilling of one round is completed, the drill will move to the opposite end of the stope for a new round, while the first round is charged and blasted.

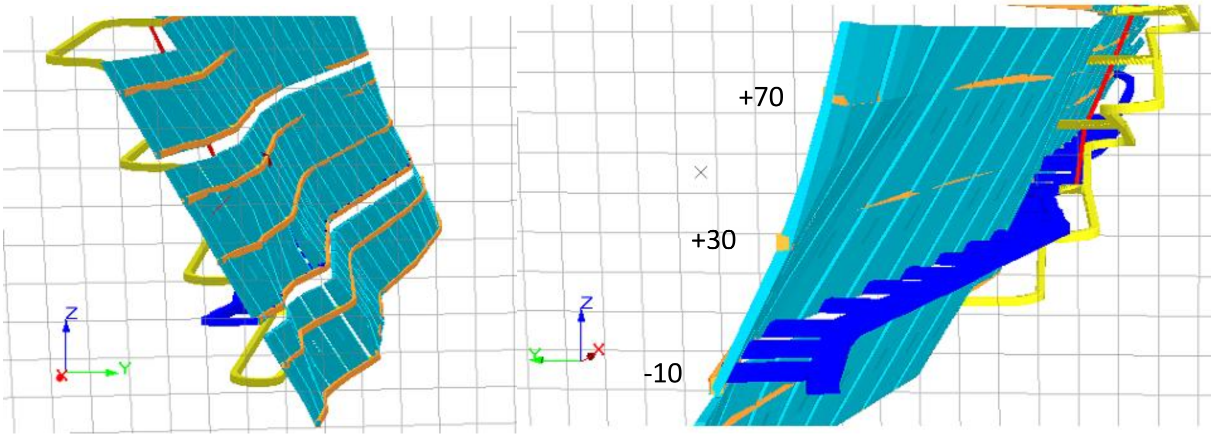

Figure 25 Cross section of stopes DC with development tunnels in footwall and drill drives following ore boundary.

7.3.7 Ore pass and waste pass

Separate ore pass and waste pass will be developed between haulage level and drill drifts above, located next to the ramp. Face are will be circular with $7m^2$ cross section. At ore pass bottom, ore will flow directly into the crusher chamber by a regulating mechanism. Waste pass will end in a designated area along the haulage tunnel. Cross cuts from drill drifts to ore pass will be 6,5m high, allowing space for truck tipping. An option of several ore passes, every 200m along strike was considered an option as it allows direct LHD loading and tipping without haulage trucks. The option was rejected due to higher development costs, and loading drifts for truck hauling in drifts was favoured.

7.3.8 Cable bolting

The low cost stoping strategy with 40m sublevel interval and 102m high stopes, will require artificial support of hangwall, as the likelihood for instability is estimated empirically to be high (ch.4.3 p.21). Experience from other similar deposits in Scandinavia, suggests that cable bolting should be a part of the stoping routine. At Nussir, it is suggested to bolt in a fan every 2 meters along drill drifts. Cable bolting of hangwall will be done for every drill drift, except the bottom drifts where hang wall is more stable. Cable bolt design suggested is 3-5 bolts per fan, 15.2 mm diameter, fully cement grouted cable bolts (without pretensioning) in ring shape towards hangwall, each giving strength of 50 ton (see Figure 26) . Cable bolt length can vary from 7 to 12m, and each fan consists of 30m at average bolting (Läppalainen, 2012).

Cable bolt holes normally are 51 mm dia, drilled by long holing. The same holes can be partly used for sludge sampling for ore boundary determination. Hole grouting is done by a proper cement pump. Cable bolting will be mechanized, done by a specially constructed cable bolt rig from a contractor.

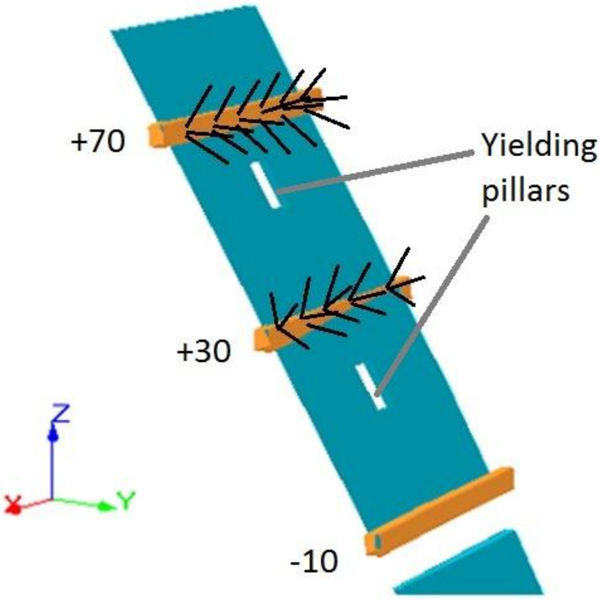

Figure 26 Stope 102m high vertically, 40m long and 3m thick. Proposed cable bolting of hangwall illustrated

7.4 Mechanized equipment for narrow mining

Mechanized equipment and mining technology is a requirement for successful mining of narrow ore bodies like Nussir. Below is a short description on some of the suggested equipment, with emphasis on the capacity vs, space requirement. Equipment investments will not be considered in this report, as Nussir plan to hire contractors to do most of the excavations underground. Equipment size and drift size relations are illustrated in Figure 27.

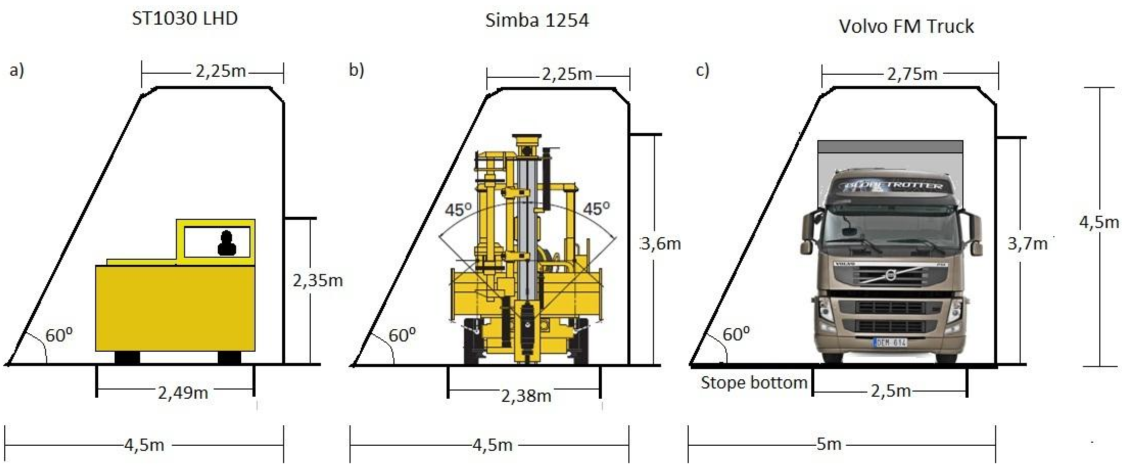

Figure 27 Suggested equipment placed in cross sections. a) LHD in drill drift, b)production rig in drill drift, c) haulage truck in combined drill and loading drift.

7.4.1 Production drill rig

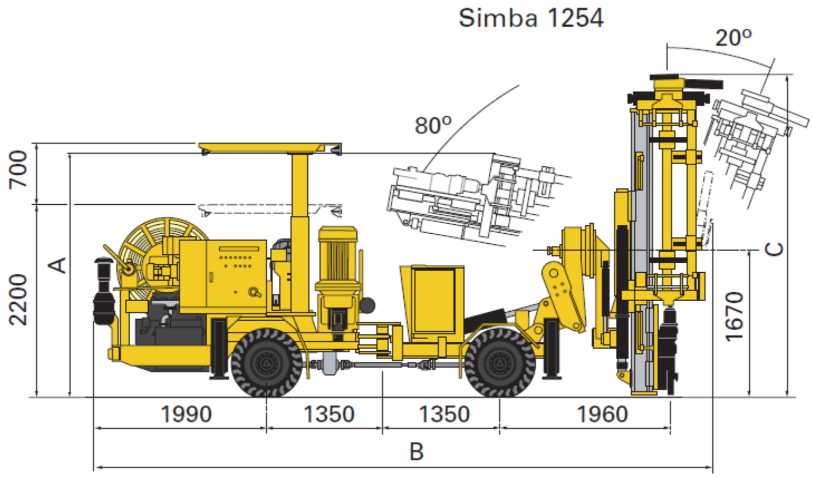

Figure 28 Suggested longhole production drill rig, Simba 1254 from Atlas Copco

An electrical powered long hole drill rig, Simba 1254, from Atlas Copco is suggested for stoping. Dimensions are 2,38m width, 2,81m tramming height and 3,6m height when drilling, which will fit in the 4,5m wide and 4,5m high drifts. It is able to drill long holes both upwards, downwards and to the sides, which allows it to drill cable bolt holes. The rig will be fitted with a COP 1838ME rock drill, capable of drilling 51-64mm holes. Hole diameter depend on the rock conditions and operators skill to drill straight holes. Larger holes will generally be straighter, but lead to higher charging density, causing harmful blasting which may lead to dilution. Suggested hole diameter will be 51mm. From the drill rate index and net penetration diagram in (ch.4.2.1 p.17), drill rig performance is calculated in Table 15.

Table 15 Production drill rig performance and theoretical capacity

Inputs		**Unit**
Drilling rate index Nussir mineralisation rocks	44	
Net penetration rate for 51mm holes	208	cm/min
Moving and hole positioning time per hole	3	min
Number of holes per fan	3	holes
Number of fans per 40m stope length	16	holes
Hole length	20	m
Calculations		
Drillhole length per 40m stope	960	m
Time to drill one stope upward	10,1	Eh
Time to drill one stope upward and downwards	20,2	Eh
Rig drilling capacity drilling	**658,7**	Tonne/Eh

Considering the extra time needed for identifying ore boundary prior to stoping, cable bolting and tramming between stopes, in-situ capacity will be reduced. Two production drill rigs will therefore be needed, providing the flexibility needed to maintain desired production.

The suggested production drill rig will not be capable of horizontal drift drilling, unless reconfigured to do so. Horizontal drifts and development tunnels will therefore be drilled by common tunnel rigs.

7.4.2 Bolting rigs

Specially configured rigs will take care of tunnel reinforcements such as bolting and shotcreting as a part of the tunnel excavation cycle. Cable bolting of hangwall in stopes, may also be mechanized by a specially configured cable bolting rig, available on the marked, if such machines can be fitted into the small drill drifts. Cable bolts can alternatively be mounted in holes drilled by the production rig, manually (Lappalainen, 2012).

7.4.3 Loader

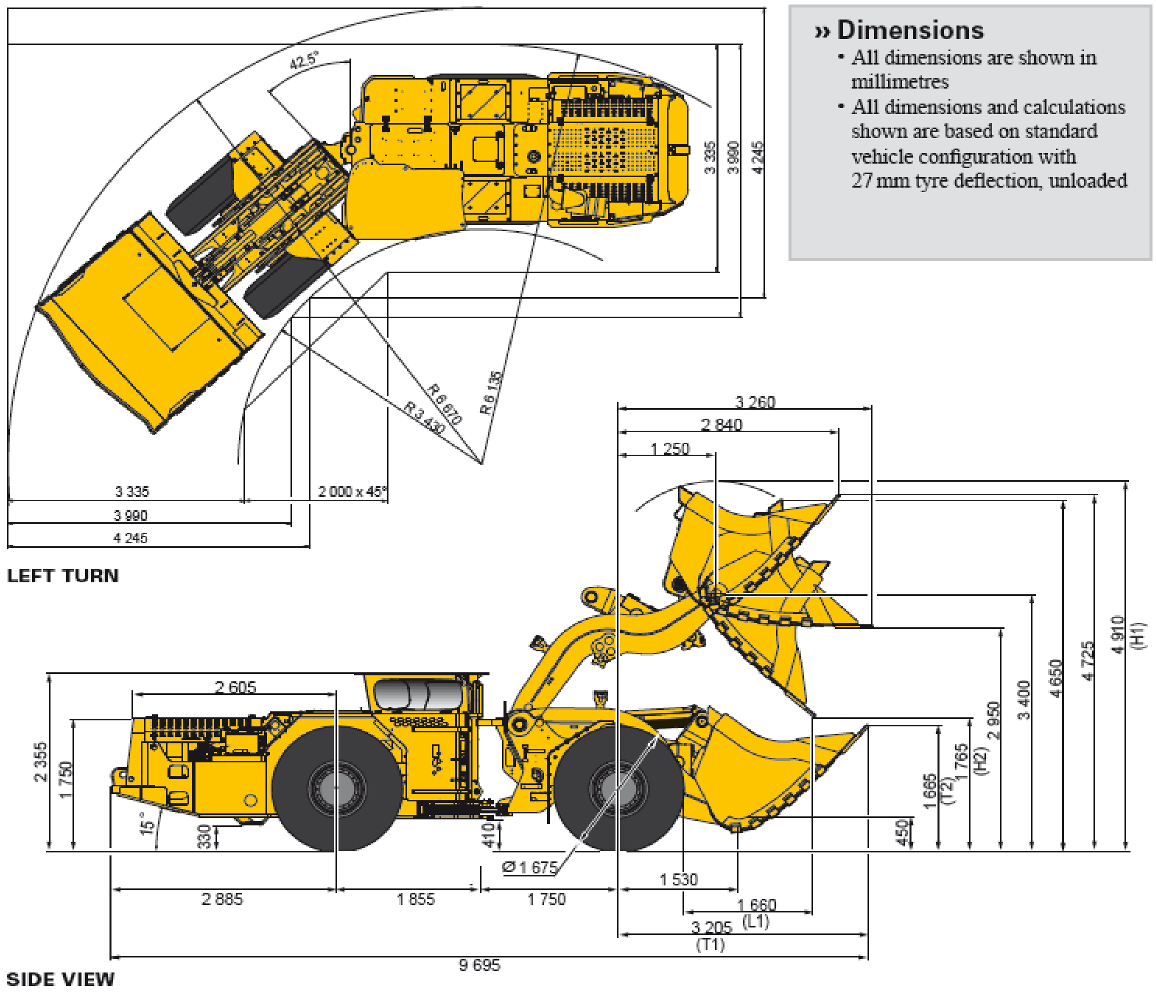

Figure 29 Suggested LHD for narrow drifts, model ST1030 from Atlas Copco. **(Copco)**

The suggested LHD machine for underground operations in Nussir is an Atlas Copco Scooptram ST1030 with a maximum width of 2.49m, equiped with the largest bucket size option of 5m^3. It is chosen for it’s tramming capacity of 10t and narrow construction, allowing narrow drifts yet high production.

The machine provides a safe working enviroment for the driver, protected by the steel roof, allowing good overview when driving backwards and forewards. Blasted ore from drifting will be driven all the way back to the ore pass next to the ramp. Blasted ore from stoping will be loaded from the bottom drift, using the remote control option when working under unsupported areas, and loaded onto haulage trucks. Calculated cycle times and capacities are given in Table 16 below:

Table 16 LHD Scooptram ST1030 performance and theoretical capacity

Speed in drift	10 km/h	Loading time mucking	20 sek
Bucket capacity	10 t	Dumping and turning	30 sek

	Stoping	*Drifting*
Cycle time	61,25 sek	185 sek
Av. LHD driving distance	25 m	300 m
Loading capacity	**587,8 t/Eh**	**194,6 t/Eh**

The mine should have a sufficient number of LHD’s for handling both drifting and stoping operations simultaneously. The ST1030 LHD will be the smallest machine, preferred for loading in drill drifts because of width. Two of these machines would be suitable. One or two larger LHD’s should be used for loading ore from draw points, where more space is available. These larger machines would also be used in the excavation of development tunnels prior to production stoping.

7.4.4 Haulage truck

Haulage trucks have to be chosen wisely for the suggested mine design. The Maximum haulage distance from stope to ore pass will be 600m. Suggested truck option is either highway tip trucks, e.g. Volvo FM or mine trucks, e.g. Atlas Copco MT431B. Both trucks would be able to carry around 30tonnes. Difference is that highway trucks are 2,5m wide, by 3,7m high, while mine trucks are generally 2,7m high and 3m wide. Highway trucks are also faster and lighter and come with modern, low emission engines. The price of a mine truck is generally five times more than the highway truck, but maintenance cost are lower (Copco). Highway trucks will also provide more flexibility for the responsible hauling contractor, as they are easier to buy and replace in a matter of days. The consequence of using highway trucks is that loading drifts must be higher.

7.5 Schedule

Portal, access tunnel from Dypelv, underground infrastructure and ramp from +10 to +90 will be developed prior to actual production in year 0. Drifts will then be excavated, producing ore, followed by stoping, retreating back to the ramp, while drifting continues on the -60 level. Access to stopes C and D will be granted in year two, allowing a smooth transition from stope A and B, to C and D, without production declining. Development, drifing and stoping will continue in the same way for stopes C and D. Scheduling is based on advancement rates from Norwegian tunnelling practice, (Zare, 2007) and the mining equipment described in ch.7.4 p.56. Schedule is shown in Table 17.

Table 17 Development and production schedule with tunnel meters given.

	year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Development							
Access tunnel	580						
Ramp +10 to +90	800						
AB Haulage tunnel +10		900					
Ore pass +10 to +90		92					
Ramp +10 to -60		700					
Haulage tunnel to CD			1480				
DC Ramp -10 to +180			1900				
DC Ore pass -10 to +70			92				
DC ramp -10 to -80				700			
DC ore pass +70 to +180					172		
Production							
AB Drift + 10		777					
AB Drift +50 and +90		943					
AB stoping +10							
AB Drift -20 and -60		892					
AB Stoping -60							
DC Drift -10			1150				
DC Drift +30 and +70			2300				
DC Stoping -10							
DC Drift -40 and -80				1150	1150		
DC Stoping -80							
DC Drift + 100					1150		
DC Drift+140 +180					1150	1150	
DC Stoping +100							

A stoping plan has been made, to evaluate the grade and tonnage produced each year from the block model. The order in which stopes are excavated is mainly controlled by the retreating principle of the mining method. Stopes at the end of the drill drift must be excavated first, retreating back to the ramp. Stopes at haulage level will be excavated first, and then the stopes below, followed by the stope above. Drift excavations and stoping excavations have not been separated in Datamine, introducing some inaccuracy in the scheduled production. Stopes have been added dilution by the thickness-dillution relation given in ch.5.5 p.32. Affecting tonnes and grades in the financial model. The actual output can be seen in Table 18.

Table 18 Nussir scheduled ore delivered to process plant by year

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Wt. Average
Tonnage	455 985	436 331	429 562	600 340	554 340	578 626	3 055 185
Ag ppm	9,49	9,06	11,61	12,27	12,52	11,12	11,13
Au ppm	0,12	0,07	0,17	0,10	0,12	0,11	0,11
Cu %	0,95	0,97	0,96	0,90	0,93	0,93	0,94

Table 19 Nussir scheduled production tonnage visualized by stopes and colour coding from Table 18 to mark year

TONNES	B9	B8	B7	B6	B5	B4	B3	B2	B1	A1	A2	A4	A4	A5	A6	A7	A8
+10 level	28682	34978	42803	54903	66858	72528	72584	63901	48448	28500	30315	31218	12259	12687	13421	13147	13683
-60 level							9276	9766	9497	25228	24110	28424	28678	29974	32205	28835	25404

Tonnes	D14	D13	D12	D11	D10	D9	D8	D7	D6	D5	D4	D3	D2	D1	C1	C2	C3	C4	C5	C6	C7	C8	C9
+ 100 level	36135	38959	45512	52317	57330	59681	57602	48819	39280	27714	24391	30035	35143	39771	22204	19098	15068	16113	16383	17678	19389	16494	14347
-10 level	38579	44403	48306	51836	51360	50529	54115	55421	50886	40095	37132	36163	36557	35140	28009	25904	26766	31495	29688	21266	31193	30712	31692
- 80 level	18602	16306	14415	16485	20407	22416	27596	31196	34684	36240	34788	28818	22523	16749	17984	9668	14692	23932	25325	21266	31193	18430	22445

8. Financial analysis

8.1 Costs

8.1.1 Unit costs

Unit costs for development tunnels, is the price planned to pay a contractor. Costs are based on average budget prices collected from three Scandinavian contractors (Smeberg, et al.). Cable bolting costs are based on past practice narrow mining (Läppalainen, 2012) and production costs are also based on past practice mining (Smeberg, et al.).

Table 20 Unit costs for development tunnels in footwall and production in ore. From Nussir's cost database.

Development	**Face area m2**	**Daily Advance m/d**	**Cost NOK/m**
Surface			
Access road from plant to mine			2 000
Portal and preparation			50 000
In footwall			
Access/haulage tunell	50	10	16 700
Ore pass vertical	7,1	1	12 000
Ramp	30	10	13 400
Permanent rock support			5 000
Ventilation raise	7,1	1	10 000
Ore draw points	30	12	13 300
Conveyor belt			20 000
In ore			
Drift	11,8	5	11 300
Slot hole	0,3	16,5	9 600
Cable bolting of hangwall in drift			1 000
Ore handling			**NOK/tonne**
Stoping		9	60
Ore loading on truck			12
Ore hauling to crusher			7
Primary crushing			10

8.1.2 Capital costs

Capital costs, also refered to as the mining cost (Lane, 1988), in this mine design assessment will only include development costs necessary to gain access to the minable areas described in ch.7.1 p.47. It will be the access tunnel, ramps, ore passes, haulage tunnel, ventilation shaft, conveyor belt and permanent rock support. Other necessary capital costs such as the establishment of the processing plant, infrastructure and pre investment costs, will not be included, as they should ideally be depreciated over a longer life of mine, with larger reserves than the 3 million tonnes considered in this report. Capital expenditures will be paid as the developments are being done. The cost for developing stopes C and D will therefore be paid in year 2 instead of year 0. Capital costs for each development step will be depreciated over the lifetime of the actual development. Total capital costs required for accessing the mineralised materials is:

CAPEX = 254,5 million NOK

83,3NOK per tonne produced.

8.1.3 Treatment costs

Treatment cost are the tonnage dependent costs, including all costs required to extract and process the mineralised materials defined as ore by the mine design, after mineralised material is made accessible by developments (Lane, 1988). Processing costs are given by past practice experiences from Nussir, given the annual production. Mining costs include, drifting, slot drifting, stoping, cable bolting, loading, hauling and primary crushing. Mining cost is given per tonne and is calculated as the total cost for mining production divided by diluted ore. Treatment costs are:

Prosessing cost = 64,9NOK/tonne ore produced

Mining cost = 177,18NOK/tonne produced

Major mining cost are identified by Figure 30, to be drifting and slot drifting. Stoping costs are also high on the graph, but not when considering the ore tonnage directly produced from the stoping operation. Drifting and slot drifting costs are highly influenced by mine design layout which determines sub-level, affecting the number of drifts, and pillar layout, affecting the number of slot holes needed.

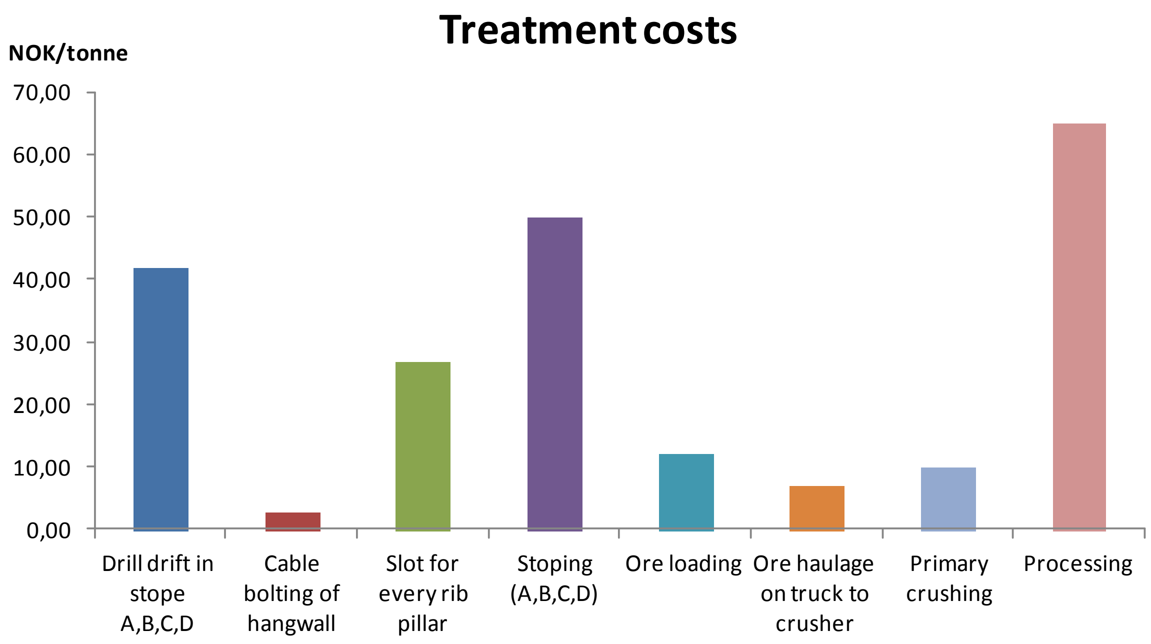

Figure 30 Treatment costs for Nussir mine design

8.1.4 Marketing cost

Marketing cost is defined as smelting, refining, transportation and selling (Lane, 1988). Costs are incurred per tonne copper produced from the processing plant. Silver and gold are bound to the copper minerals (Sandstad, 2008) and will not be given any extra costs. The marketing cost is based on numbers from Nussir's financial model, calculated to be:

$$\text{Marketing cost} = 2082{,}5 \text{ NOK/tonne copper concentrate}$$

8.2 Cash Flow

A cash flow analysis have been calculated for the mining selected areas from ch.7.1 p.47, during a 6 year period, given that mine development is the only capital expenditure. Input parameters in the analysis include:

Table 21 Metal prices and input parameters in financial analysis

Item	Rate
Copper price	6600 US$/tonne
Silver Price	20 US$/Oz
Gold Price	1000 US$/Oz
Royalty	0,75 %
Tax rate	28 %
Discount rate	10 %

The revenue per tonne after tonnage dependent costs are paid, is relatively stable at 410NOK/tonne, which is partly due to the fact that mining costs/tonne is an overall average and partly due to blending of ore qualities from two different stope sections.

The mining operation starts off with negative cash flows during the initial two years of production, due to high development costs related to the rapid development of stope D and C. By the end of year three, all developments are taken care of, leaving only production costs (Figure 31). The capital expenditure of 254,5million NOK is going to be paid back after 5 years and 9 months, only three months before end of mining.

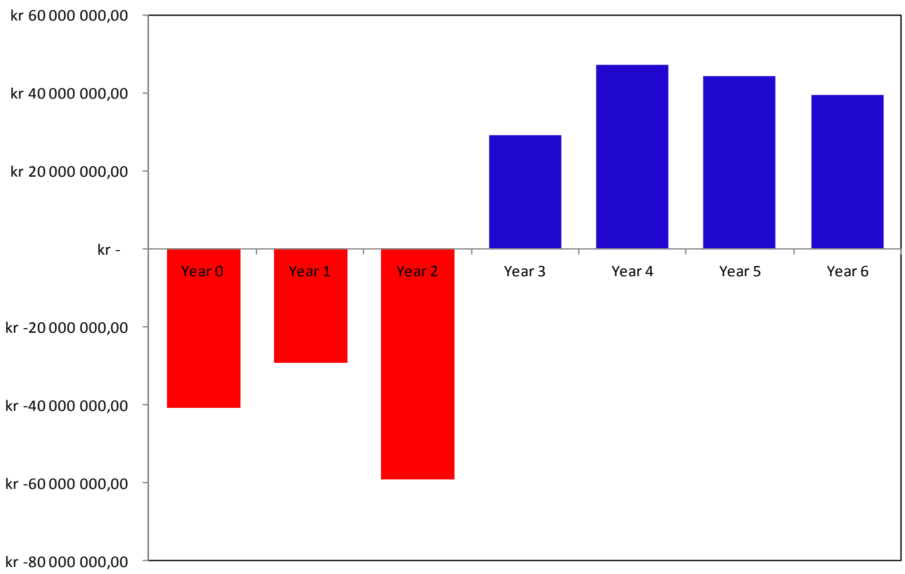

Figure 31 Project cash flows year by year

The value of the project at the end of the period, the net present value (NPV), is 32million NOK. Considering the NPV in relation to investments (CAPEX), the net present value quotient becomes 0,13, implying 13% increase if initial investment.

Net Present Value	**=**	**32 million NOK**
Internal rate of return	**=**	**17%**

8.3 Sensitivity analysis

The financial model of the mining operation is heavily relying on constant variables trough the life of mine. Only grades and tonnages have been calculated for each period, yet with high degree of uncertainty due to block model comprised of spatial sampling. A sensitivity analysis is therefore constructed for the projects NPV, with respect to variables that may change over time. Figure 32 illustrate the NPV outcome for a +-15% change in variables.

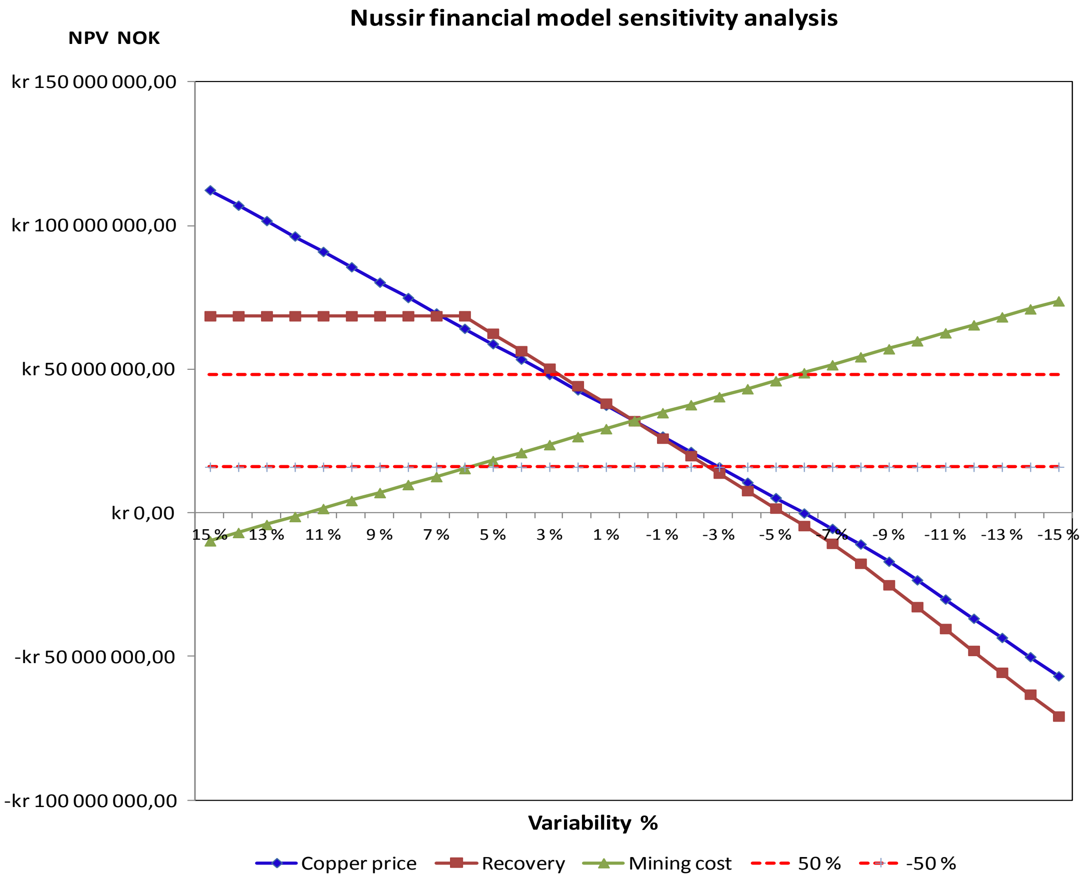

Figure 32 Sensitivity analysis of Nussir project's NPV for a +-15% variability in Cu price, process recovery and mining cost.

The Nussir mining project's overall economy is very sensitive to a range of variables. Producing raw materials, is a conjecture business, where product prices vary with world economy, making the blue line in Figure 32 highly representative for possible project outcomes. From Figure 32, we can identify the variability that cause zero NPV, which is 6% decrease in copper price from estimated 6600$/tonne to 6200$/tonne. If the price for copper stays on a stable level of 7590$/tonne, project will come out with an NPV of 112 million NOK, 350% increase from base case.

The best way to deal with price uncertainty in mining base metals, is to control the ore grade, by saving high grade blocks for periods of recession.

Mining cost is also identified as a variable with great influence on project value. A 12% increase of mining costs, will cause zero NPV according to current financial model. This implies that project is very sensitive to mine design parameters affecting mining cost, such as driftmeters per tonne, stoping costs, slot drifting and loading and hauling cycles.

9. Discussion

The technical and economical feasibility of mining the steeply dipping narrow mineralisation of Nussir have been assessed with high degrees of uncertainty in factors which have been identidyed to influence economy. The geotechnical parameters, stress, Q-values and fractures, will influence stope and pillar dimensions, which again will affect ore loss, driftmeters/tonne and efficiency in material flow. Suggested stope dimensions, are empirically estimated to be unstable, without the additional yielding pillars and cable bolting in stopes. By this, mine design is identified as dependent on favourable rock and stress conditions, for project to be economical.

The working cycles during mining operation have not been described in detail, assuming that contractor will get the job done. Producing from two stopes simultaneously on each side of ramp, suggested in schedule, will give optimal blending of grades, given current block model. The question is, whether annual production in schedule can be maintained, when working in the 600m long drifts with only one entrance? The current mine design assumes that efficiency will be maintained by meeting bays and loading drifts, allowing vehicles to pass each other in drifts.

9.1 How geology affects mine design

Current geometric wireframe model is chosen as representative for ore boundary determination in ch.5.5 p.32. Dillution estimates, stability, ore recovery and the whole design is based on that assumption. Mine design is, in my opinion feasible for mining a deposit similar to the geometric model. The mine design is also flexible to some ore boundary irregularities, as long as boundaries are not waving back and forth within 20m up dip.

Denser drilling would also improve our interpretation of the crucial parameter thickness. Modelled thickness between three samples will be given by each sample intersections length. The influence of one sample is consequently having large impact on thickness and tonnage. It is therefore unfortunate that hole NUS-DD-08-004 was stopped during drilling only 1m into the mineralised zone. Known mistakes like this, and possible mistakes that we do not know of, from core handling and assaying, strongly affects the resource, from which mine design is based on.

Either way, the geometric models is what I know at this stage, consequently affecting the plan. However, the plan is not static, it's merely for the sake of planning, helping us to understand the technical challenges in narrow mining. The plan, or mine design suggested in this thesis, should be updated, whenever new geological information come fresh out of the borehole.

9.2 Narrow ore mining practices

An interesting topic for discussion is whether the proposed mine design is realistic in terms of narrow ore mining. Narrow ore mining have been well described in this report, as a concept of mining steeply dipping ore bodies below 4m thickness, with small, but efficient mechanized equipment, seeking the optimum between dilution and ore loss. Mining practices have been well exemplified by two reports and one field visit, representing 4 different narrow mining layouts. Studying examples from industry have been preferred as background material for this thesis, rather than pure theoretical papers. Each of the 4 narrow mining practices, two from Zinkgruvan and two from Viscaria, reflect the importance of adapting mine design to geological conditions.

Both Viscaria mine an Zinkgruvan, mine an ore with twice as high value as Nussir. Zinkgruvan reserves in the Cecilia orebody is 8,1%Zn, 2,1%Pb and 49g/tAg. With current Zink price of 2000$ per tonne, the sales value per ore including additional minerals will be over 1000NOK/tonne. Mining costs are consequently allowed to be high in Zinkgruvan. When considering that Zinkgruvan have this value on ore, and 3-5m ore thickness, it is obvious that their established narrow vein mining practices cannot be initiated for the low grade Nussir deposit, even if deposits have somewhat similar geometry.

9.3 Resource to reserve definition

The resources selected for mine design assessment in this report, have a resource classification of indicated which is sufficient for defining probable reserves according to (JORC, 2004). Modifying factors such as environmental concerns are not yet sorted out, as they are waiting for final approval with authorities. Geotechnical factors are partly sortet out with remaining uncertainties regarding stresses and fracture zones. The uncertainty in stress conditions, will in my opinion reflect the question whether to go for the suggested mine design in this report or go for a double bench stoping method, which induce 12-15% higher mining costs. Uncertainties in fracture zones, may influence ore loss in minor or major areas of the mine design, implying less resources to be defined as ore. The economical feasibility of starting a mine, based on selected resources is not present at 6600$/tonne copper price. A start up project should be financially stronger, to cover the additional process plant and infrastructure capital costs.

By this, no resources will be conversed to reserves from the assessment.

10. Conclusion

- Selected mining method is sublevel open stoping with 40m sublevel interval, 40m long stopes with height 62m and 102m.
- Mining of the indicated resources above 0,9%Cu cut-off in eastern part of Nussir, including thickness-dependent dilution will produce 3,05Mt ore with 0,94%Cu, 11,13g/t Ag and 0,11g/t Au.
- Dilution increase tonnage throughput and treatment costs. It decreases head-grade of ore and may also decrease production of minerals, if treatment capacity is limiting.
- The financial model derived from the mine design indicate positive project value for 6 years life of mine, only if mining development is the only capital expense.
- Equipment should be fit to drifts and vice versa, finding the optimum between capacity and dilution.
- No reserves can be defined from the resources at Nussir.

11. Further work

1) Update geological wireframe model with interpretation of fault zone, which may cross cut and affect areas of stope C and D.
2) Asses the technical and economical feasibility of mining the Inferred resources in Nussir west, and see it contributes to the financial model in this report.
3) Numerical stability analysis of proposed pillar layout, by exporting stope wireframes to Phase 2 software.
4) Evaluate the economic situation of investing in mining equipment, instead of hiring contractor, for the proposed mine design.
5) Assess the possibility of dumping blasted ore directly onto conveyorbelt, possibly via ore pass, reducing hauling costs. Ask Vegard Olsen at Orica mining services.
6) Evaluate ventilation needs for given mine design and machinery, and add costs to financial model.
7) Evaluate possibility of opening stopes by cut-blasting instead of slot raises. Will the cost saving be worth possible blast damages to stope walls?
8) Carry out hydraulic fracturing of borehole to quantify stress conditions.

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Appendices

Appendix 1 Tunneling and drilling performance numbers

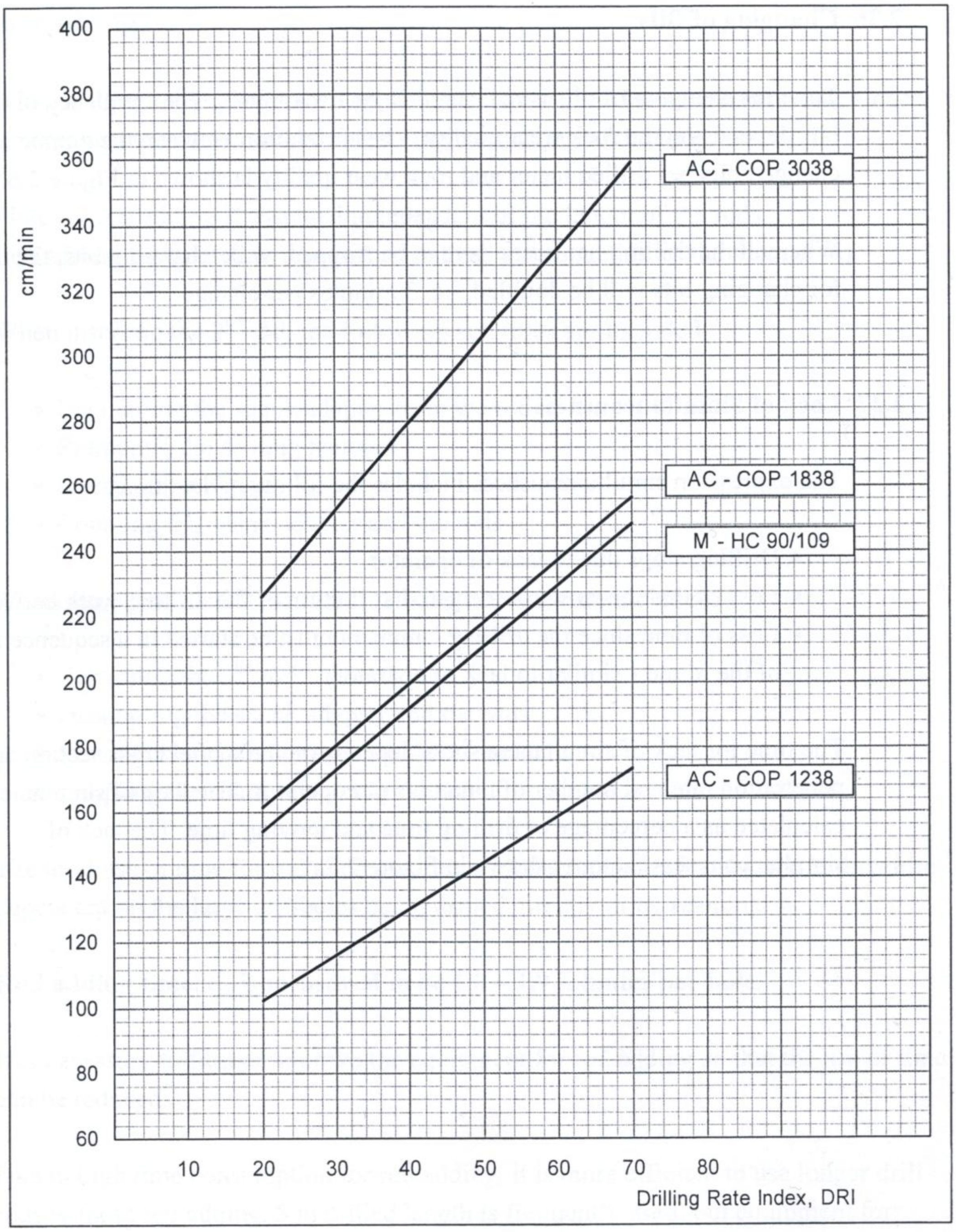

Figure 2.3 Net penetration rate.

Net penetration rate, v_b, as a function of Drilling Rate Index, DRI, measured in cm/minute for 48 mm drillholes.

The penetration rate has been recorded during standard round drilling. Not knowing if the drilling hammers are adjusted for maximum penetration or for optimum drill steel consumption.

Appendix 1

Net penetration rate vs. Drillhole diameters

Figure 2.4 Net penetration rate (in % of penetration rate for 48 mm drillhole) as a function of drillhole diameter for hydraulic drilling hammers.

Advance rate

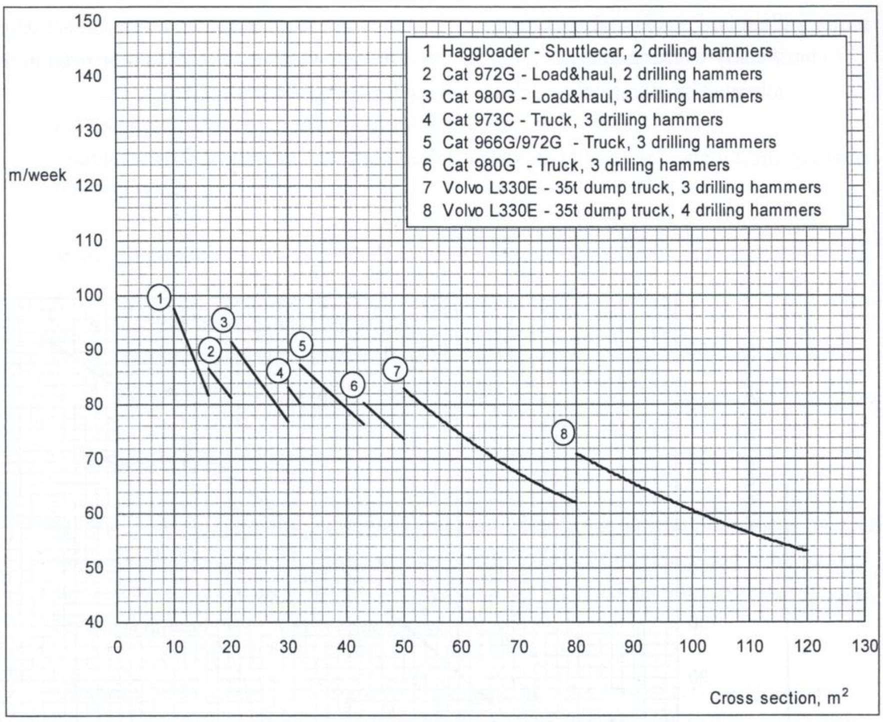

Figure 6.3 Standard weekly advance rate for best combination of loading and transport equipment as a function of cross section area for 48 mm drillhole diameter and 3 km tunnel length.

Appendix 2

Nussir block model stope evaluation

Stope ID's																	
+10 level	B-10-9	B-10-8	B-10-7	B-10-6	B-10-5	B-10-4	B-10-3	B-10-2	B-10-1	A-10-1	A-10-2	A-10-3	A-10-4	A-10-5	A-10-6	A-10-7	A-10-8
-60 level							B-20-3	B-20-2	B-20-1	A-60-1	A-60-2	A-60-3	A-60-4	A-60-5	A-60-6	A-60-7	A-60-8
TONNES																	
+10 level	20807	26371	33417	45714	57886	64070	63798	54391	38588	22545	24238	25179	9971	10385	11040	10668	10833
-60 level							7102	7536	7356	19631	18908	22614	23051	24253	26323	23281	19879
AG																	
+10 level	8,23	10,78	11,34	12,17	12,97	12,89	12,51	12,80	13,43	12,23	11,16	10,48	9,83	9,04	8,38	8,46	8,53
-60 level							9,44	8,71	7,01	9,35	10,06	10,21	10,11	9,89	8,80	8,48	8,63
AU																	
+10 level	0,14	0,21	0,25	0,22	0,18	0,14	0,10	0,07	0,07	0,06	0,06	0,07	0,08	0,09	0,08	0,08	0,08
-60 level							0,08	0,05	0,04	0,05	0,06	0,08	0,09	0,10	0,09	0,08	0,09
CU																	
+10 level	0,97	1,00	1,02	1,08	1,24	1,25	1,24	1,23	1,20	1,25	1,23	1,23	1,24	1,18	1,11	1,10	1,08
-60 level							1,00	1,05	1,12	1,25	1,22	1,23	1,24	1,16	1,13	1,10	1,10
THICK																	
+10 level	2,10	2,45	2,86	4,05	5,31	6,27	6,00	4,68	3,16	3,05	3,22	3,37	3,53	3,66	3,76	3,48	3,06
-60 level							2,62	2,71	2,76	2,82	2,92	3,14	3,31	3,43	3,63	3,39	2,89

Appendix 2

Nussir block model stope evaluation

Stope ID's

+ 100 level	D-100-14	D-100-13	D-100-12	D-100-11	D-100-10	D-100-9	D-100-8	D-100-7	D-100-6	D-100-5	D-100-4	D-100-3	D-100-2	D-100-1	C-100-1	C-100-2	C-100-3	C-100-4	C-100-5	C-100-6	C-100-7	C-100-8	C-100-9
-10 level	D-10-14	D-10-13	D-10-12	D-10-11	D-10-10	D-10-9	D-10-8	D-10-7	D-10-6	D-10-5	D-10-4	D-10-3	D-10-2	D-10-1	C-10-1	C-10-2	C-10-3	C-10-4	C-10-5	C-80-6	C-10-7	C-10-8	C-10-9
- 80 level	D-80-14	D-80-13	D-80-12	D-80-11	D-80-10	D-80-9	D-80-8	D-80-7	D-80-6	D-80-5	D-80-4	D-80-3	D-80-2	D-80-1	C-80-1	C-80-2	C-80-3	C-80-4	C-80-5	C-80-6	C-10-7	C-80-8	C-80-9

Tonnes

+ 100 level	29723	31467	37315	43608	48370	50703	48677	39970	30482	19483	15836	20838	25553	29923	16485	13378	10276	11253	11363	12639	14087	11270	9302
-10 level	30724	35922	39776	42861	42281	41415	45020	46484	42084	31812	28843	27627	27524	26200	20447	18721	19749	23756	21392	15954	22706	22337	23138
- 80 level	13720	11678	10062	12018	15501	17595	22221	25630	28988	30234	28557	23206	17705	12690	12742	6405	10545	19103	20006	15954	22706	13413	16557

AG ppm

+ 100 level	9,21	9,08	8,76	10,01	12,68	15,34	15,50	14,35	12,11	9,94	9,63	11,27	13,40	14,61	15,07	14,95	14,86	17,41	22,49	29,03	31,13	32,14	35,57
-10 level	6,57	7,71	8,75	9,57	10,95	11,84	12,71	13,53	14,07	13,97	13,33	13,97	15,80	17,20	16,47	13,67	12,44	17,00	20,85	28,08	25,03	26,80	27,90
- 80 level	5,95	6,85	6,96	7,49	8,94	9,97	11,13	13,35	15,87	16,39	16,05	15,27	15,43	16,06	31,86	12,47	16,68	22,65	23,58	28,08	25,03	29,30	27,08

AU ppm

+ 100 level	0,22	0,23	0,21	0,21	0,19	0,18	0,18	0,15	0,10	0,06	0,05	0,05	0,07	0,09	0,09	0,09	0,08	0,09	0,10	0,11	0,11	0,12	0,13
-10 level	0,37	0,31	0,23	0,22	0,21	0,19	0,17	0,12	0,09	0,08	0,06	0,07	0,08	0,09	0,09	0,08	0,08	0,09	0,10	0,17	0,12	0,13	0,12
- 80 level	0,33	0,34	0,33	0,30	0,25	0,22	0,20	0,14	0,10	0,09	0,08	0,07	0,07	0,07	0,22	0,04	0,06	0,10	0,12	0,17	0,12	0,18	0,16

CU %

+ 100 level	1,07	1,17	1,22	1,27	1,30	1,28	1,26	1,22	1,20	1,20	1,18	1,05	0,97	0,93	0,94	1,13	1,17	1,24	1,41	1,51	1,52	1,63	1,74
-10 level	0,86	0,99	1,11	1,16	1,20	1,17	1,19	1,22	1,22	1,19	1,14	0,99	1,01	1,05	1,03	1,14	1,20	1,26	1,33	1,37	1,47	1,58	1,65
- 80 level	0,80	0,87	0,89	0,95	1,04	1,11	1,09	1,08	1,14	1,17	1,14	1,02	1,04	1,11	1,41	1,19	1,25	1,31	1,34	1,37	1,47	1,51	1,48

THICK m

+ 100 level	3,76	3,40	3,69	4,08	4,41	4,62	4,46	3,66	2,78	1,87	1,45	1,79	2,12	2,43	2,30	1,85	1,69	1,83	1,79	1,99	2,11	1,70	1,44
-10 level	3,15	3,43	3,79	3,88	3,78	3,69	4,03	4,24	3,89	3,10	2,79	2,59	2,43	2,34	2,15	2,07	2,24	2,45	2,05	2,40	2,13	2,12	2,15
- 80 level	2,24	2,00	1,83	2,14	2,53	2,94	3,34	3,74	4,15	4,10	3,72	3,34	2,96	2,50	1,92	1,54	2,02	3,19	3,03	2,40	2,13	2,12	2,24

Appendix 3 Financial Moedel

1 gram = 0,035273962 Oz

Production Period	0	1	2	3	4	5	6	
	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Ore Production (tonnes)		455 985	436 331	429 562	600 340	554 340	578 626	3 055 184,63
								-
Average silver grade (g/tonne)		9,49	9,06	11,61	12,27	12,52	11,12	
Average gold grade (g/tonne)		0,12	0,07	0,17	0,10	0,12	0,11	
Average copper grade (%)		0,95	0,97	0,96	0,90	0,93	0,93	
Recovery		94 %	94 %	94 %	94 %	94 %	94 %	
Silver produced (Oz)		143 473	131 121	165 378	244 154	230 135	213 367	1 127 627,74
Gold produced (Oz)		1 888	997	2 410	1 978	2 235	2 068	
Copper produced (tonnes)		4 056	3 993	3 877	5 085	4 861	5 055	
Silver Price ($/Oz		$ 20	$ 20	$ 20	$ 20	$ 20	$ 20	
Gold Price ($/Oz)		$ 1 000	$ 1 000	$ 1 000	$ 1 000	$ 1 000	$ 1 000	
Copper price ($/tonne)		$ 6 600	$ 6 600	$ 6 600	$ 6 600	$ 6 600	$ 6 600	
Sales revenue		**kr 186 011 729,51**	**kr 176 825 405,17**	**kr 184 710 516,85**	**kr 238 504 130,68**	**kr 229 618 737,57**	**kr 234 231 179,45**	**kr 1 249 901 699,24**
(1US$ = 5,9 NOK)								
Less: Mining production Cost (OPEX)		kr 80 793 509,71	kr 77 311 280,13	kr 76 111 878,33	kr 106 371 124,42	kr 98 220 599,30	kr 102 523 688,92	kr 541 332 080,80
Processing Cost		kr 29 593 400,11	kr 28 317 913,83	kr 27 878 591,69	kr 38 962 080,69	kr 35 976 670,70	kr 37 552 825,19	kr 198 281 482,22
Fixed costs		kr 7 200 000,00	kr 7 200 000,00	kr 7 200 000,00	kr 7 200 000,00	kr 7 200 000,00	kr 7 200 000,00	kr 43 200 000,00
Royalty		kr 1 395 087,97	kr 1 326 190,54	kr 1 385 328,88	kr 1 788 780,98	kr 1 722 140,53	kr 1 756 733,85	kr 9 374 262,74
Marketing Cost (smelt. Ref. trans. Sell.)		kr 8 446 666,21	kr 8 314 581,01	kr 8 074 118,87	kr 10 590 259,15	kr 10 122 499,63	kr 10 527 677,04	kr 56 075 801,91
Operating income		**kr 58 583 065,50**	**kr 54 355 439,67**	**kr 64 060 599,09**	**kr 73 591 885,44**	**kr 76 376 827,41**	**kr 74 670 254,46**	**kr 401 638 071,56**
Depreciation (CAPEX)		kr -6 760 466,67	kr -21 905 252,67	kr -51 028 655,17	kr -58 253 721,83	kr -58 253 721,83	kr -58 253 721,83	kr -254 455 540,00
Operating Profit before Tax		**kr 51 822 598,84**	**kr 32 450 187,00**	**kr 13 031 943,92**	**kr 15 338 163,60**	**kr 18 123 105,58**	**kr 16 416 532,62**	**kr 147 182 531,56**
Less: Income Tax (28%)		kr -14 510 327,67	kr -9 086 052,36	kr -3 648 944,30	kr -4 294 685,81	kr -5 074 469,56	kr -4 596 629,13	kr -41 211 108,84
Operating Profit After Tax		**kr 37 312 271,16**	**kr 23 364 134,64**	**kr 9 382 999,62**	**kr 11 043 477,79**	**kr 13 048 636,02**	**kr 11 819 903,49**	**kr 105 971 422,72**
Depreciation		kr 6 760 466,67	kr 21 905 252,67	kr 51 028 655,17	kr 58 253 721,83	kr 58 253 721,83	kr 58 253 721,83	kr 254 455 540,00
CAPEX	kr -40 562 800,00	kr -75 723 930,00	kr -116 493 610,00	kr -21 675 200,00	kr -	kr -	kr -	**kr -254 455 540,00**
Undiscouted Net Cash Flow (x 1000)	**kr -40 562 800,00**	**kr -31 651 192,17**	**kr -71 224 222,69**	**kr 38 736 454,79**	**kr 69 297 199,63**	**kr 71 302 357,85**	**kr 70 073 625,32**	**kr 105 971 422,72**
Discount factor (10%)	1,00	0,91	0,83	0,75	0,68	0,62	0,56	
Cash Flow Present Value (NPV) (x 1000)	kr -40 562 800,00	**kr -28 773 811,06**	**kr -58 862 993,96**	**kr 29 103 271,82**	**kr 47 330 919,76**	**kr 44 273 154,37**	**kr 39 554 734,68**	**kr 32 062 475,61**
Cumulative Cash Flow	kr -40 562 800,00	**kr -69 336 611,06**	**kr -128 199 605,03**	**kr -99 096 333,21**	**kr -51 765 413,44**	**kr -7 492 259,07**	**kr 32 062 475,61**	
Net Present Value (NOK)	**kr 32 062 475,61**							
IRR	**17 %**							
Payback	**5,81**	**years**						