Bergvesenet

Postboks 3021, N-7441 Trondheim

Rapportarkivet

Bergvesenet rapport nr	Intern Journal nr	Internt arkiv nr	Rapport lokalisering	Gradering
2425	09/00084-1 vedl 4			

Kommer fra ..arkiv	Ekstern rapport nr	Oversendt fra	Fortrolig pga	Fortrolig fra dato:
Intex Resources ASA	SRK U3234	Molynor AS		

Tittel

Conceptual Mine Study for the Nordli (Hurdal) Molybdenum Deposit, Norway

Forfatter	Dato	År	Bedrift (Oppdragsgiver og/eller oppdragstaker)
Bray Chris, Riley Paul	Oktober	2007	Molynor AS v/ Intex Resources ASA

Kommune	Fylke	Bergdistrikt	1: 50 000 kartblad	1: 250 000 kartblad
Hurdal	Akershus		19154	Hamar

Fagområde	Dokument type	Forekomster (forekomst, gruvefelt, undersøkelsesfelt)
Gruveteknisk Økonomisk		Hurdalfeltet Nordli Nordliforekomsten

Råstoffgruppe	Råstofftype
Malm/metall	Mo

Sammendrag, innholdsfortegnelse eller innholdsbeskrivelse

Rapporten er på engelsk og er utført på oppdrag for Crew Minerals ASA (før Crew omorganiserte til Intex Resources ASA).
Etter gjennomgang av tilgjengelig info, går SRK for å benytte Sublevel Caving (SLC) som bryingsmetode i det foreløpige studiet av en estimering av gruvekostnaden.
Det gjennomgåtte studiet indikerer at prosjektet som det nå står er økonomisk marginalt, men det pekes på at nøyaktigheten er +/- 30-50 %. Dette studiet forutsetter en produksjon på 5 MT/år. En dbling vil gi break even.

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Conceptual Mine Study for the Nordli (Hurdal) Molybdenum Deposit, Norway

Report Prepared for

Crew Minerals ASA
Munkedamsveien 45 F
Po Box 1416 Vika,
N-0115 Oslo
Norway

Report Prepared by

SRK Consulting
Engineers and Scientists

FILE REF: U3234_Hurdal_Conceptual Study_Final_Oct 2007 .doc OCTOBER 2007

SRK Version 06.01

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Munkedamsveien 45 F
Po Box 1416 Vika,
N-0115 Oslo
Norway

SRK Project Number U3234

SRK Consulting (UK) Ltd
3rd floor Windsor Court
1-3 Windsor Place
Cardiff, UK
CF10 3BX

Tel : +44 29 20 34 81 50
Fax : +44 29 20 34 81 99
cardiff@srk.co.uk
www.srk.co.uk

FILE REF: U3234_Hurdal_Conceptual Study_Final_Oct 2007 .doc OCTOBER 2007

Report Authors
Chris Bray
Paul Riley

Reviewed by:
Martin Pittuck

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SRK Consulting
Engineers and Scientists

SRK Consulting (UK) Ltd
Floor 5 Churchill House
Churchill Way
Cardiff
United Kingdom
CF10 2HH
e-mail: cardiff@srk.co.uk
URL: www.srk.co.uk
Tel: + 44 (0)29 20 34 81 50
Fax: + 44 (0)29 20 34 81 99

Our ref: *007_U3234_Hurdal_Conceptual Study_Final_CB .doc* October 2007

Conceptual Mine Study for the Nordli (Hurdal) Molybdenum Deposit, Norway

1 INTRODUCTION

SRK Consulting (UK) Ltd ("SRK") is currently producing an Independent Mineral Resource Estimate for the Nordli deposit, near Hurdal, Norway for Crew Gold Ltd ("Crew"). The terms of reference (ToR) for the work on the Independent Mineral Resource Estimate were discussed and agreed by Steffen Schmidt of SRK and Jette Blomsterberg of Crew and included a separate desktop evaluation of principal economic factors for the exploitation of the deposit in order to to assure that the deposit has "reasonable prospects for eventual economic extraction".

In order to substantiate the classification as being "potentially economic" required for reporting any mineralisation as a Mineral Resource, SRK undertook high-level conceptual mine and process planning, which has been used to develop the related capital and operating costs.

A sublevel caving mining method has been selected for this analysis and assumes a bulk mining rate of 5 million tonnes per annum (Mtpa). Access to the deposit and proposed mining levels is via decline from the surface. A conventional porphyry treatment circuit has been assumed for the purpose of this study. This report covers the first 10 years of operation it is practical to assume that the mining costs will increase in the future and also as a result of the mine progressing deeper underground.

This study is purely a desktop estimation exercise with no site investigation or research into real costs of mining, geological, geotechnical, metallurgical, smelting, environmental and social aspects and should only be used in this context. Further investigation and research in these areas will provide a greater level of accuracy of the estimates provided in this report. The scoping level of this report provides a ±30-50% level of accuracy.

2 MINING

2.1 Selection of Mining Method

2.1.1 Selection parameter and constraints

The mining method selection requires an in depth analysis of the orebody using the following information:

- geological cross sections and a longitudinal section;
- level maps;
- block model (grade model); and
- geomechanical characteristics of the host and surrounding rock.

It is also important to find comparable deposits that are being or have been mined successfully. The following is a list of typical considerations used to determine the appropriate mining method for a deposit:

- maximise safety (integrity of the mine workings as a whole or in part);
- minimise cost (bulk mining methods have lower operating costs than selective extraction);
- minimise the schedule required to achieve full production (optimise stoping sequence);
- optimise recovery (80% or greater of geological reserves);
- minimise dilution (20% or less dilution)
- minimise stope cycle time (drill, charge, blast, load, backfill, set);
- maximise mechanisation and automation;
- minimise preproduction development;
- minimise stope development;
- maximise gravity assistance; and
- maximise natural support (partial vs complete extraction).

The mining method must also allow flexibility and adaptability based on:

- size, shape and distribution of target mining areas;
- distribution and variability of ore grades;
- sustaining the mining rate for the mine life;
- access requirements; and
- opening stability, ground support requirements; hydrogeology (ground water and surface runoff), and surface subsidence.

Open stoping is considered unsuitable as the molybdenum deposit may require substantial pillars due to low expected compressive rock strengths. Considering a mine production rate of 5 Mtpa, the possible backfill costs would also be substantial. The main opposition to block caving is the initial upfront capital development costs and the unknown suitability of the orebody to this type of caving.

2.1.2 Stoping method – Sublevel cave

After review of the available information, although limited, SRK has chosen to use a sublevel caving method for the purpose of this conceptual mining cost estimation project.

Sublevel caving (SLC) adapts to this large molybdenum deposit with steep dip and continuity to depth. The sublevel footwall drift is required to be stable and must also allow for loading of trucks. This mining method requires the hangingwall to fracture and collapse, following the cave, and subsistence of the ground surface above the orebody has to be tolerated.

Caving requires a rock mass where both orebody and host rock fracture under controlled conditions. As the mining removes rock without backfilling, the hanging wall continues caving into the voids. Continued mining results in subsidence of the surface, where sinkholes may appear. Given the close proximity of the Nordli deposit to the surface this will most certainly be the case. Continuous caving is important, to avoid creation of cavities inside the rock mass, where a sudden collapse could induce an inrush.

SLC extracts the ore through sublevels which are developed in the orebody at regular vertical spacing. Each sublevel features a systematic layout with parallel drifts, along or across the orebody. The sublevel drifts start from the footwall drive and continue across to the hangingwall.

Pre-production development to prepare SLC stopes is extensive, and mainly involves driving multiple headings to prepare the sublevels. A ramp connection is required to connect the sublevels and to link the main transport route for materials handling which is via decline linked to the surface.

Longhole rigs drill the ore section above the drift in a fan pattern well ahead of production. Blasting on each sublevel starts at the hangingwall with a slot rise into the upper level cave and mining retreats towards the footwall. Adjacent crosscuts are mined at a similar pace with upper levels mined ahead of lower sublevels to preserve the cave and avoid undermining.

Each longhole fan is blasted separately and the ore fills the drawpoint. The drawpoint is mucked out until the waste dilution reaches the set limit. Waste dilution in SLC varies between 15 and 40% and ore losses may be 15 to 25%, depending on local conditions. Dilution is less significant for orebodies surrounded by subeconomic ore or mineralised waste.

Figure 2-1 is a representation of a SLC operation. The Nordli project may anticipate mining a portion of the orebody using surface mining methods, but, for the purpose of this exercise, only mining from a underground perspective is examined. Figure 2-1 shows an orepass and rail haulage, however, for this project SRK would recommend to use a trackless decline access from the surface.

Horizontal sections were taken from the resource model which showed the continuity of the orebody to depth. It also shows an area between -100 and -200 m where the orebody almost separates into two mining targets. This may constrain the mine scheduling for sublevels in this area. The final mine design will have to take this into account so that the caving front is continuous and allows enough flexibility to achieve the required mining production schedule. A program such as Mine 2-4D can assist in the visual design and scheduling of difficult areas such as this.

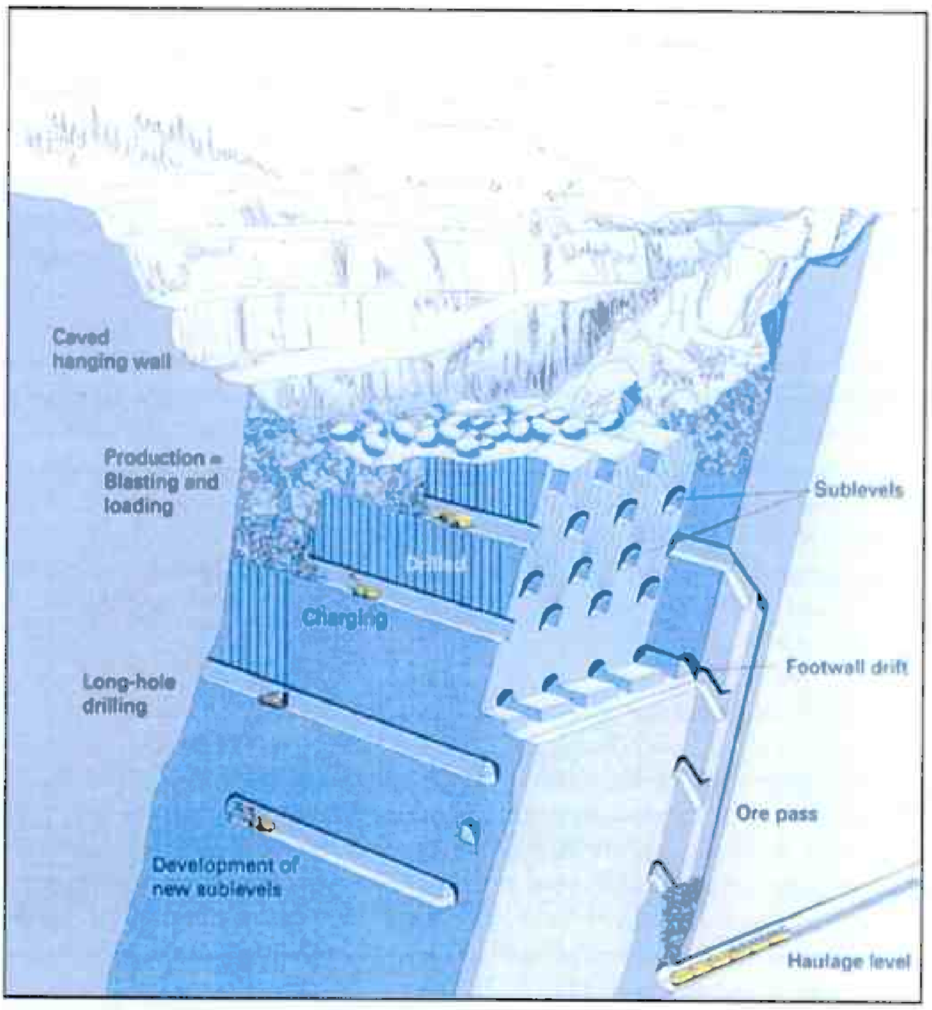

Figure 2-1 Example of SLC (Sublevel Caving) layout

2.1.3 *Geotechnical aspects of the mining method*

The geotechnical input to this project will be significant and essential. For the purposes of this cost estimation exercise, SRK has used a simple rock bolting regime in all mine development. However, if the ground conditions are poor then additional support (mesh, shotcreting, cablebolting etc) will be required which could increase mining costs substantially.

Ground water conditions will also have a significant impact on mining. It is assumed that there will be substantial water inflow from seasonal snow melting if sub-level caving is used but there may also be large water inflows from alternative underground sources. The inflow of water into stoping and development areas may have an adverse effect on the existing ground conditions, which might require costly dewatering to mitigate.

2.2 Development

Development is expected to progress using a typical mechanised approach. The initial decline access would consist of a single heading decline using a twin boom jumbo with the appropriate sized truck and loader arrangement adjusted as required.

The conceptual design allows for two exhaust rises to be developed and a fresh air intake rise which also serves as an emergency escapeway. The exhaust rises are located in a cuddy off the footwall drive at the extremities of the orebody strike. The fresh air intake raise would be located central to the decline access which also acts as a fresh air intake. The raises are expected to be developed as required in line with level development. One approach to developing the rises is to use a Down the Hole Hammer (DTH) rig for drilling followed by a staged retreat rise blasting procedure. Break through rises to the cave to initiate stoping on each level can be carried out using conventional airleg rising or an uphole rise using the longhole rig.

2.3 Equipment Selection

The initial decline development for the Project will require a twin boom jumbo, loader and ancillary equipment. Due to the close proximity of the orebody to the surface, it is expected that ore development will commence in the first year of the Project.

The production rate selected for the Project requires a fleet of 50 tonne trucks to transport the ore and waste to the surface.

Additional drainage pumps may be required if adverse water conditions exist in the mining area.

Table 2-1 Equipment requirements and estimated cost in USD

Equipment Requirements	Quantity	Unit Cost USD	Total Cost USD
Primary Stopes			
Longhole Drills	4	550,000	2,200,000
Scoop Tram (6.5m^3)	4	713,000	2,852,000
Development			
Twin boom Jumbo drills	6	692,800	4,156,800
Raise Drills	2	319,000	638,000
Scoop Tram (6.5m^3)	3	713,000	2,139,000
Rear-Dump Truck (50t)	3	541,000	1,623,000
Production			
Rear-Dump Truck (50t)	4	541,000	2,164,000
Jacklegs	6	5,310	31,860
RoofBolters	1	722,000	722,000
Shotcreters	2	50,000	100,000
Main Fans	2	171,950	343,900
Compressors	1	189,100	189,100
Anfo Loaders	2	66,000	132,000
Service Trucks	8	90,000	720,000
Diamond Drills	1	37,670	37,670
Total			**18,049,330**

2.4 Production Scheduling

The selection of an appropriate production rate is usually a function of the estimated Ore Reserve and the requirements to achieve a reasonable return on capital invested to bring a mining project into production.

For the purposes of this exercise, a production rate of 5 Mtpa has been used to determine mining costs. This is considered by SRK to be a reasonable production rate for the bulk mining of molybdenum ore using a single decline access and truck fleet. In order to increase this production rate, due consideration is required to determine alternative materials handing processes or additional capital infrastructure for the mine development.

2.5 Mining Costs

SHERPA cost estimation software for underground mines has been used to assist with estimating the mining costs for the Nordli molybdenum project. All due care has been taken to model the project based on the limited information and time available to SRK.

Where necessary, based on prior experience, the results provided by the estimation software have been adjusted to provide a more accurate or realistic estimate of mining costs.

The default engineering data used in the estimation software is mostly supplied by equipment manufacturers. The Default costs are sourced from the "Mining Cost Service", published by Western Mine Engineering Inc.

2.5.1 *Model inputs*

The model inputs used for the mine cost modelling exercise are included in Appendix 1 & 2 of this report:

- Data Entry for Cost Model; and
- Equipment Parameters for Cost Model.

Estimates were made based on the Sublevel caving mining method selected and industry standards for machinery and operating parameters. In the event where the cost information was not know then the estimate provided by the tables in the software were used.

2.5.2 *Mine Capital costs*

The mining costs were reviewed over a 10 year period for the mine. The upfront capital expenditure in the first year is USD34.8M with an additional USD10.3M for working capital in the second year. The unit prices for all items have not been adjusted for time related increases (or decreases). Additional capital and unit prices increases are expected as the underground mine gets deeper. Table 2-2 is a summary of the project operating and capital costs for the first 10 years.

Table 2-2 Capital cost summary in USD

Capital Cost Item	Year 1	Year 2
	USD	USD
Equipment purchase	18,049,330	
Preproduction development	3,222,997	
Surface facilities	5,981,004	
Working capital		10,370,660
Engineering & management	4,747,821	
Contingency	2,792,836	
Total Capital Costs	**34,793,988**	**10,370,660**

Surface facilities

Table 2-3 shows an estimate of surface facilities that may be required for the Project which are associated only with the mining activities.

Table 2-3 Surface facility requirements in USD

Surface Facilities Costs			
Facility	**Footprint**	**Unit Cost**	**Total Cost**
	m^2	**USD per m^2**	**USD**
Mine Office	3117	798	2,486,337
Workers Changehouse	1463	858	1,255,254
Mine Warehouse	832	554	460,545
Surface Shop	1803	835	1,505,974
Mine Plant Building	234	1,166	272,893
Total			**5,981,004**

Decline

A decline from the surface would be required to transport the ore and waste from the mine. The dimensions used are suitable for underground trucks with a 50 t capacity. Although no site investigation has been conducted by SRK, it is anticipated that a boxcut will be required and portal construction. For the purpose of this exercise, it was assumed that the first 50 m of the decline will require sets and shotcrete with the additional decline development using only rock bolt support. The total length of decline is 2,520 m, which is scheduled over the 10 year mining plan. The estimated costs are as shown in Table 2-4.

Table 2-4 Estimate of decline construction costs in USD per metre and total

Decline Construction Costs		
Labour	**Total Cost**	**Unit Rate**
	USD	**USD/m**
Miners	5,195,289	2,061.62
Surface Workers	1,263,914	501.55
Mechanics	2,230,429	885.09
Foreman	1,026,742	407.44
Total	**9,716,374**	**3,855.70**
Equipment Operation	**Total Cost**	**Unit Rate**
	USD	**USD/m**
Drills	82,952	32.92
Rock Bolters	6,804	2.70
Mucker	39,709	15.76
Hauler	63,721	25.29
Shotcreters	380	0.15
Total	**193,616**	**76.83**
Supplies	**Total Cost**	**Unit Rate**
	USD	**USD/m**
Powder	150,956	59.90
Caps	96,726	38.38
Boosters	163,800	65.00
Fuse	157,968	62.69
Drill Bits	63,943	25.37
Rods	14,508	5.76
Water Pipe	248,361	98.56
Air Pipe	20,160	8.00
Backfill Pipe	0	0.00
Vent Tubing	50,400	20.00
Electric Cable	70,275	27.89
Rock Bolts	75,605	30.00
Shotcrete	382,735	151.88
Timber	0	0.00
Cement	0	0.00
Liner Plate	3,832,606	1,520.91
Total	**5,328,196**	**2,114.36**
Decline Construction Cost	**15,238,186**	**6,047**

2.5.3 *Mine Operating costs*

The typical mine operating cost is USD16/t ore mined. Additional vertical development is required in years 2 and 5 of the project for ventilation purposes which slightly increases the mine operating costs. A summary of the mine operating costs over a 10 year period of the mine life is shown in Table 2-5.

Table 2-5 Mining cost per tonne ore in USD

Operating Cost Item	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr	Yr 8	Yr 9	Yr 10
	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Supplies	0	1.36	1.30	1.30	1.38	1.30	1.30	1.30	1.30	1.30
Hourly Labour	0	7.34	7.34	7.34	7.34	7.34	7.34	7.34	7.34	7.34
Salaried Labour	0	1.95	1.95	1.95	1.95	1.95	1.95	1.95	1.95	1.95
Equipment Operation	0	0.65	0.59	0.59	0.68	0.59	0.59	0.59	0.59	0.59
Miscellaneous	0	1.13	1.12	1.12	1.14	1.12	1.12	1.12	1.12	1.12
Contingency 30%	0	3.729	3.69	3.69	3.75	3.69	3.69	3.69	3.69	3.69
Total Mining Costs	**0**	**16.16**	**15.99**	**15.99**	**16.24**	**15.99**	**15.99**	**15.99**	**15.99**	**15.99**

3 PLANT AND INFRASTRUCTURE

3.1 Introduction

Limited information is available on the Nordli material, although it is understood that the deposit consists of a relatively low grade molybdenum porphyry with minimal associated copper. The levels of other possible impurity elements, which could incur either penalties or additional credits, and their deportment into the molybdenum concentrate are also unknown. It is understood that limited analytical and mineralogical data is available and that no metallurgical testwork results are available as a guide to the optimum treatment route and anticipated performance. In order to generate preliminary economic data SRK has used on generic industry practices and parameters.

The proposed treatment of the primary molybdenum sulphide (MoS_2) material consists of primary crushing, two stage milling and rougher flotation followed by fine grinding of the rougher concentrate and cleaner flotation to produce a final molybdenum (Mo) concentrate containing an estimated 50 – 55% Mo (around 90% MoS_2). The concentrate would be dewatered, dried and transported for sale.

This proposed treatment represents a conventional molybdenum treatment circuit and is similar to that typically used for the treatment of copper and copper/molybdenum porphyries, although it only considers the production of a single concentrate. Capital costs have been developed by SRK for a range of plant throughputs based on industry and in-house data. However, the full scope of the new treatment project, and particularly the supporting infrastructure requirements, is not fully known and assumptions have had to be made in this regard. Operating costs have been calculated from first principles using estimated figures for consumptions of reagents, milling steel, electric power, etc based on typical industry standards and typical unit costs to "western" standards.

A molybdenum recovery of 80 – 85% into concentrate has been estimated based on typical head grades of 0.15% MoS_2 for incorporation in the economic assessment.

3.2 Basis of Design

3.2.1 Assumed ore characteristics, head grade and metal analyses

The resource grade of the Nordli material is understood to be in the range of 0.1 to 0.3% MoS_2. Typical treatment plant feed is anticipated to average 0.15% MoS_2. No information has been provided on mineralogy or the association or deportment of the molybdenum. The Nordli material is understood to be a molybdenum porphyry hosted in granite and is therefore assumed to be relatively hard.

It has been assumed that the sulphide minerals can be liberated from gangue minerals at a relatively coarse grind and that further grinding will be required on the rougher flotation concentrate to liberate the molybdenum from the other base metal sulphides to generate a high grade concentrate by multiple stages of cleaning.

Levels of impurity elements such as arsenic, mercury, etc or possible metals for which additional credits could be accrued are also unknown and these have therefore been ignored.

3.2.2 *Process design parameters and TEM technical inputs*

Preliminary Resource Data 140 Mt at 0.15% MoS_2 using a cut off grade of 0.1 MoS_2%

Plant Design Parameters

Throughput range	Mtpa	2 - 15
Molybdenum recoveryinto concentrate	%	85
Final molybdenum concentrate grade	%Mo	50 – 55

3.2.3 *Inputs incorporated in the calculation of operating costs*

Average costs of labour and consumables are based on typical European / North American numbers and an average electrical power cost of USD0.03/kWh has been assumed, based on grid power. Further details of assumed unit consumable costs are included in Table 3-3.

3.2.4 *Design basis for costing*

Capital costs have been based on data from other copper, copper/molybdenum and molybdenum porphyry studies undertaken recently, mainly for projects in North and South America and from plant operating data. These have been factored for the different proposed throughputs and adjusted where necessary to cater for different requirements related to the anticipated scope but have not been increased over typical international costs for prevailing conditions anticipated in Norway.

3.3 Proposed Treatment Circuit and Anticipated Plant Performance

3.3.1 *Process and plant description*

A conventional porphyry treatment circuit has been assumed for the purpose of this study.

Ore from the proposed sub-level cave mining operation will be transported to surface by truck and crushed to nominally < 150-250 mm in a primary gyratory crusher. Crushed ore will be stockpile prior to the milling plant and reclaimed by feeders under the stockpile to feed the ore to the primary grinding circuit. A grinding circuit comprising a single semi-autogenous grinding (SAG) mill and ball mill will reduce the ore to a nominal grind size of 80% passing around 150µm. The SAG mill circuit will incorporate a pebble crushing circuit to handle critical size material in the mill.

The milled product from the primary grinding circuit will be fed to a bank of rougher flotation tank cells. Tailings from rougher flotation will be pumped to the main tailings storage facility. The rougher concentrate will be reground to 80% passing 20 – 30 µm and fed to a bank of cleaner / cleaner scavenger tank flotation cells. The cleaner scavenger tailings will be pumped separately to the tailings area for disposal. The cleaner concentrate will be upgraded by a further five cleaning stages, with tailings recycled. The cleaning circuit will incorporate conventional mechanically agitated flotation cells with a vertical column cell used for the final stage of cleaning. The final concentrate from the cleaner flotation circuit will be thickened, filtered, dried, and loaded into bulk storage sacks for transportation off-site.

3.4 Forecast Recoveries and Concentrate Grades

The anticipated recovery of molybdenum sulphide into concentrate is estimated to be around 85%, although no testwork is available to support this. Typical concentrates grades could be expected to be in the 50 – 55% Mo range. Both the recovery and concentrate grade are dependant upon on the grain size of the molybdenum minerals and their association and deportment when milled to liberate the molybdenum from the gangue and other sulphide minerals.

It should be highlighted that any oxidised molybdenum minerals cannot be successfully treated through the proposed treatment facility and recovered to concentrate.

3.5 Scope of Plant and Services

The scope of the plant is a complete processing operation as described above. All necessary on site support services are incorporated including:

- electrical power receipt and reticulation; and
- support buildings including offices, laboratory, plant workshops, reagent storage and make up, warehouses, etc.

The following are excluded from the scope of the plant:

- tailings storage – initial and ongoing capital costs;
- off-site costs for the transport of concentrate; and
- general and administration operating costs.

The scope of the project includes site preparation, all concrete work, processing and material handling equipment, steelwork, electrics and instrumentation. The costs are for a complete installed operation including design, erection, project management, indirect construction costs, temporary construction facilities, first fill of reagents and consumables, etc.

The plant with the exception of the stockpile would be fully enclosed in a common milling concentrator building. The primary crusher would be located in a separate structure.

3.6 Anticipated infrastructure requirements

It is understood that the Nordli site is close to existing amenities and facilities. It has been assumed that electrical power can be supplied from a suitable location within around 20 km of the mine site and that industrial and potable quality water can be sourced within a similar distance.

Access to the site is assumed to be good with no significant access roads required, except a short road directly to the mine site from existing paved roads. It is also assumed that local support facilities are available locally including accommodation, medical etc and that significant site camp facilities are not required. Construction support services including construction personnel, cranes etc. are assumed to be available close to the mine site.

3.7 Plant and Infrastructure Capital and Operating Costs

3.7.1 Overview and forecast accuracy

The capital and operating costs for processing facilities presented are based on generic industry data. Based on the assumptions incorporated, the costs are estimated to be to an accuracy of ±25 – 30%.

3.7.2 Capital cost

The estimated capital costs for the project based on the scope and parameter detailed are presented in Table 3-1.

Table 3-1 Estimated Project Cost for Treatment Plant and Infrastructure.

Plant Throughput	Mtpa	2	3	5	10	15
Total Costs Plant and Infrastructure	USDM	110	145	190	280	350
Sustaining capital	USDM	1	1.2	1.7	2.2	3

3.7.3 Operating costs

Cost of reagents and consumables are presented in Table 3-2. Estimated plant operating costs have been calculated and are presented in Table 3-3.

Table 3-2 Indicative reagent and consumable cost build-up in USD

REAGENTS, CONSUMABLES, AND SUPPLIES

Reagents, Consumables, and Supplies	Consumption Rate	Units	Unit Cost (USD/t)	Annual Cost (USD/t Ore)
Primary and Pebble Crusher Liners	12	g/t	4,000	0.05
SAG Mill Balls	650	g/t	900	0.59
SAG Mill Liners	50	g/t	2,000	0.10
Ball Mill Balls	1000	g/t	950	0.95
Ball Mill Liners	80	g/t	2,000	0.16
Regrind Ball Mill Balls	30	g/t	1,000	0.03
Regrind Ball Mill Liners	5	g/t	5,000	0.03
Diesel Oil	40	g/t	650	0.03
Pine Oil	50	g/t	2,500	0.13
Detergent (OP-6)	25	g/t	4,000	0.10
Frother (MIBC)	5	g/t	3,000	0.02
Lime	60	g/t	160	0.01
Flocculants	10	g/t	3,450	0.03
Dryer Fuel Oil	0.06	l/t	650	0.04
Other				0.10
Process Consumables				0.20
			Total Annual Cost	2.55
			Cost per Tonne Ore Processed (USD/t)	2.55

Table 3-3 Plant operating cost summary in USD

CONCENTRATOR OPERATING COST

Throughput	Mtpa / tpd	2 / 5479				3 / 8219				5 / 13699				10 / 27397				15 / 41096		
Labour	Units	USD/year	USD/year	USD/t ore	Units	USD/year	USD/year	USD/t ore	Units	USD/year	USD/year	USD/t ore	Units	USD/year	USD/year	USD/t ore	Units	USD/year	USD/year	USD/t ore
Management	12	120,000	1,440,000		12	120,000	1,440,000		13	120,000	1,560,000		15	120,000	1,800,000		15	120,000	1,800,000	
Operator	30	75,000	2,250,000		30	75,000	2,250,000		32	75,000	2,400,000		36	75,000	2,700,000		36	75,000	2,700,000	
Maintenance	20	80,000	1,600,000		20	80,000	1,600,000		22	80,000	1,760,000		25	80,000	2,000,000		25	80,000	2,000,000	
Tailings	12	65,000	780,000		12	65,000	780,000		14	65,000	910,000		20	65,000	1,300,000		20	65,000	1,300,000	
Total	74		6,070,000	3.04	74		6,070,000	2.02	81		6,630,000	1.33	96		7,800,000	0.78	96		7,800,000	0.52
Consumables	USD/t		USD/year	USD/t ore	USD/t		USD/year	USD/t ore	USD/t		USD/year	USD/t ore	USD/t		USD/year	USD/t ore	USD/t		USD/year	USD/t ore
Balls and Liner	1.90		3,796,000		1.90		5,694,000		1.90		9,490,000		1.90		18,980,000		1.90		28,470,000	
Reagents	0.31		617,000		0.31		925,500		0.31		1,542,500		0.31		3,085,000		0.31		4,627,500	
Process Consumables	0.20		410,000		0.20		600,000		0.20		1,000,000		0.20		2,000,000		0.20		3,000,000	
Other Raw Water	0.14		278,000		0.14		417,000		0.14		695,000		0.14		1,390,000		0.14		2,085,000	
Total			5,091,000	2.55			7,636,500	2.55			12,727,500	2.55			25,455,000	2.55			38,182,500	2.55
Electric Power		0.01 x USD/kWh	3			0.01 x USD/kWh	3			0.01 x USD/kWh	3			0.01 x USD/kWh	3			0.01 x USD/kWh	3	
Crushing and Milling	kWh/t	18	1,080,000		kWh/t	18	1,620,000		kWh/t	18	2,700,000		kWh/t	16	4,800,000		kWh/t	15	6,750,000	
Flotation and Other	kWh/t	10	600,000		kWh/t	10	900,000		kWh/t	10	1,500,000		kWh/t	8	2,400,000		kWh/t	6	2,700,000	
Total			1,680,000	0.84			2,520,000	0.84			4,200,000	0.84			7,200,000	0.72			9,450,000	0.63
Maintenance	%		800,000	0.40	%	3	1,100,000	0.37	%	3	1,700,000	0.34	%	3	2,200,000	0.22	%	3	2,750,000	0.18
Contingency	%	10	1,384,100	0.69	%	10	1,732,550	0.58	%	10	2,525,750	0.51	%	10	4,295,500	0.43	%	10	5,858,250	0.39
TOTAL			15,005,100	7.50			19059150	6.35			27,783,250	5.56			46,970,500	4.69			64,090,750	4.27

3.8 Project Schedule and Anticipated Metallurgical Treatment Plan

Construction of the mill circuit and concentrator would typically take around 24 – 30 months from project go ahead to the start of commissioning depending on the capacity of the circuit and the equipment selected and availability of long lead time items. This does not allow for construction constraints resulting from adverse climatic conditions, which are not anticipated. It is assumed that the necessary supporting infrastructure could be provided in the same time frame.

Ramp up to full design throughput and performance could be expected to take a further six months from commissioning.

3.9 Conclusions, Opportunities and Risks

3.9.1 *Conclusions*

Due to the lack on any information on mineralogy or metallurgical testworks, the costs generated are based purely on conceptual processing and design parameters. In addition site conditions and requirements for the necessary supporting infrastructure have been assumed with no firm information provided related to scope and quantities. The costs presented can therefore only be considered to be preliminary and indicative.

3.9.2 *Opportunities*

The flowsheet presented represents a conventional molybdenum treatment circuit and is similar to that used for the treatment of copper and copper/molybdenum porphyries. The flowsheet and design incorporate current proven practices. Significant developments are being undertaken in the treatment of copper porphyries in an attempt to reduce capital and operating costs particularly the incorporation of high pressure grinding rolls. However, the development of such circuit is still at a preliminary stage and reliable cost data is not yet available. It is possible that when the Nordli deposit is developed, advantages from these advances can be incorporated.

3.9.3 *Risks*

The costing exercise assumes the flowsheet developed will be effective in processing the Nordli material although this is still to be confirmed from further investigations and testwork.

The scope of the project and particularly the supporting infrastructure has been assumed as no better information has been presented.

3.9.4 *Recommendations*

If the preliminary economic evaluation proves attractive, it is proposed that representative samples are obtained and that preliminary metallurgical testwork is undertaken. Based on the finding and further investigation into the project requirements, a scoping level study could be undertaken to further define the project economics.

4 FINANCIAL APPRAISAL

Following the conceptual approach to mining and processing of the Nordli molybdenum deposit, a basic financial analysis has been undertaken of the first 10 years developing the project.

This takes into account the estimated mining and processing costs and uses typical Key Performance Indicators (KPI) which are experienced at other similar worldwide molybdenum operations.

A number of factors have been assumed such as the molybdenum price, roasting charges, losses etc for this conceptual study and results should be used as indicators as to the viability of the project. The base case presented herewith assumes an annual production of 5Mt of run-of-mine ore. Mining costs would be USD 16 per tonne and milling costs USD 5.6 per tonne. At a base price of USD 24 per lb of Mo, the economic cut-off grade (operation costs only) would be about 0.1% MoS2.

Table 4-1 shows a typical annual breakdown of the revenue and a number of the associated cost that could apply to the Nordli Molybdenum Project.

Table 4-1 Annual revenue / cost breakdown for Nordli (Hurdal) molybdenum project

Annual Revenue/Cost Breakdown for Hurdal Molybdenum					
Mined Ore	t	5,000,000		**Conversions Table**	
MoS_2 Grade		0.15%		lb/tonne	2204.62
Mining Dilution		10%		Atomic mass Mo	95.94
Contained MoS_2	t	6,750		Atomic mass S	32.07
Mill Recovery		85%		**Unit Costs**	
Recovered MoS_2	t	5,738		Payable (USD/lb)	24
Handling loss (t)	0.15%	9		Roasting Charge (USD/lb)	1.5
Saleable MoS_2	t	5,729		Mining Cost (USD/t)	16
				Process Cost (USD/t)	5.58
Mo produced	t	3,433			
Mo shipped	lb	7,569,508			
Loss on Roasting (lb)	1%	75,695			
Net Mo Sold		7,493,813			
Payable Mo	USDM	180			
Roasting Charge	USDM	11			
Mining Cost	USDM	80			
Process Cost	USDM	28			

Using the estimated annual revenue and costs, the information was modelled over the first 10 years of the project using assumed inflation and tax rates. The model as shown in Table 4-2 takes into account the expected timing of construction and buildup to capacity of mining and processing activities as outlined in the conceptual study.

A number of expected costs have not been applied to the model such as the expected Shipping/Transport costs for wet concentrate, interest, loans, tax deductions, permit/licensing and for carrying out the required studies to optimise the projects. These costs are highly variable depending upon the business plan of the client and the contracts negotiated.

Table 4-2 Basic DCF model for the Nordli (Hurdal) molybdenum project

Basic DCF model for Hurdal Molybdenum

	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Payable Mo Revenue	USDM			89.9	179.9	179.9	179.9	179.9	179.9	179.9	179.9
Inflation Factor	2.0%	1.00	1.02	1.04	1.06	1.08	1.10	1.13	1.15	1.17	1.20
Mining Cost	USDM			41.6	84.9	86.6	88.3	90.1	91.9	93.7	95.6
Process Cost	USDM			14.5	29.5	30.1	30.7	31.3	31.9	32.6	33.2
G&A Cost	USDM	10.0	10.2	10.4	10.6	10.8	11.0	11.3	11.5	11.7	12.0
Roasting Charges	USDM			5.6	11.2	11.2	11.2	11.2	11.2	11.2	11.2
Total Operating Costs	**USDM**	**10.0**	**10.2**	**72.1**	**136.3**	**138.8**	**141.3**	**143.9**	**146.6**	**149.3**	**152.0**
Mining Capital Costs	USDM		34.8								
Mine Working Capital	USDM			10.4							
Plant Infrastructure	USDM	95.0	47.5	47.5							
Plant Sustaining Capital	USDM			1.8	1.8	1.8	1.9	1.9	2.0	2.0	2.0
Tailings Dam Construction	USDM		0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3
Total Capital Costs	**USDM**	**95.0**	**82.5**	**59.9**	**2.1**	**2.1**	**2.1**	**2.2**	**2.2**	**2.2**	**2.3**
net revenue		-105.0	-92.7	-42.1	41.5	39.0	36.4	33.8	31.1	28.3	25.5
Taxes	30%	-	-	-	12	12	11	10	9	9	8
Cash Flow	USDM	-105	93	-42	29	27	25	24	22	20	18
Discount Factor	8%	1.00	0.93	0.86	0.79	0.74	0.68	0.63	0.58	0.54	0.50
DCF	**USDM**	**-105**	**-86**	**-36**	**23**	**20**	**17**	**15**	**13**	**11**	**9**
NPV	**USDM**	**-119**									

Does not take into account Shipping + Land Freight (wmt), Depreciation, Interest, Loans, Tax deductions, Permitting/Licensing costs and required Feasibility studies

5 CONCLUSION

The current exercise indicates that economics of the project as it stands now are marginal. However, given the preliminary stage of the investigation and the expected accuracy of ±30 to 50%, there is room for improvement, and SRK considers that the economic criterion for reporting of Inferred Mineral Resources, namely that the resource has "reasonable prospect of economic extraction" is fulfilled. There are major upfront costs associated with developing this project and investigation into possible contractual arrangements to reduce them could considerably improve the project potential.

The operating costs for processing will reduce as the mill throughput is increased. Preliminary results show that the economic model breaks even when the mining and processing rate is doubled to 10 Mtpa. The economies of scale for the molybdenum project can also be considered an integral part of improving the project potential.

SRK has taken due care in preparation of the mining and processing costs estimates as provided in this report using the information and time available. The results and analysis provided can be used as a starting point in deciding where to progress to the next level of work in the disciplines covering all aspects of the molybdenum project.

SRK has been pleased to participate in the development of the Nordli molybdenum project and welcomes the opportunity to be of future assistance.

For and on behalf of SRK Consulting (UK) Ltd

Chris Bray (BEng)
Senior Mining Engineer

Paul Riley
Process Engineer (Associate)

Martin Pittuck
Principal Resource Geologist

Appendix 1

DATA ENTRY FOR COST MODEL

Project Title

Project Name	Hurdal Molybdenum
Description	Conceptual study for underground mining of molybdenum deposit
Company Name	
Analysis	SRK (UK)
Date	10/04/2007

	Operating Data		
PROJECT	Total Mineable Resource	50,000,000	t
	Hours per shift	12	
	Shifts per Day	2	
	Days per Year	365	
	Daily Production Rate	13,700	t
	Haul to Surface	Rear-Dump Truck	
	From Beginning of Year	2	
	To End of Year	10	
PRIMARY STOPING	Method	Sublevel Longhole	
	Muck Collection	Scoop Tram	
	Haul to Transfer Point	Scoop Tram	
	Production	100	%
SECOND. STOPING	Method	n/a	
	Muck Collection	n/a	
	Haul to Transfer Point	n/a	
	Production	n/a	%
	Deposit Specifications		
	Geometry	Massive	
	Average Dip	80	deg.
	Average Length (max. horizon)	400	m
	Average Width (min. horizon)	205	m
	Average Height (vertical)	300	m
	Vertical distance to Surface	10	m
	Horizontal Distance to Surface	50	m
	Ore and Vaste		
ORE	Density	2.6	t/m^3
	Swell	54	%
	Rock Quality Designation	50	%
	Compressive Strength	100,000	kPa
	Run of Mine	60	cm
VASTE - FV	Density	2.7	t/m^3
	Swell	54	%
	Rock Quality Designation	90	%
	Compressive Strength	100,700	kPa
	Contact	Gradational	
VASTE - HV	Density	2.7	t/m^3
	Swell	54	%
	Rock Quality Designation	90	%
	Compressive Strength	100,700	kPa
	Contact	Gradational	
	Stope Dimensions		
PRIMARY	Stope Length (max. horizon)	300	m
	Stope Width (min. horizon)	25	m
	Stope Height (vertical)	25	m
	Face Width	20	m
	Face Height	25	m
	Advance per Round	3	m
	Pillar Length (max. horizon)	0	m
	Pillar Width (min. horizon)	0	m
	Pillar Height (vertical)	0	m
SECONDARY	Stope Length (max. horizon)	n/a	m
	Stope Width (min. horizon)	n/a	m
	Stope Height (vertical)	n/a	m
	Face Width	n/a	m
	Face Height	n/a	m
	Advance per Round	n/a	m
	Pillar Length (max. horizon)	n/a	m
	Pillar Width (min. horizon)	n/a	m
	Pillar Height (vertical)	n/a	m

	Adits / Inclines / Declines		
	Location	Footwall	
	Face Width	5.5	m
	Face Height	5	m
	Length	2620	m
	Gradient	12	%
	Year Begun	1	
	Year Complete	3	
	Sequence Efficiency	85	%
	Rockbolts	2470	m
	Timber	0	m
	Lagging	0	m
	Shotcrete	50	m
	Concrete	0	m
	Steel Liner Plate	50	m
	Portal Elevation	355	m
	Shafts		
	Plan Section	n/a	
	Location	n/a	
	Hoist	n/a	
	Number of Compartments	n/a	
	Face Area	n/a	m2
	Depth	n/a	m
	Year Begun	n/a	
	Year Complete	n/a	
	Sequence Efficiency	n/a	%
	Rock Bolts	n/a	m
	Timber	n/a	m
	Lagging	n/a	m
	Steel Liner Plate	n/a	m
	Concrete	n/a	m
	Collar Elevation	n/a	m
	Drifts and Crosscuts		
DRIFTS	Face Width	4.5	m
	Face Height	4	m
	Average Gradient	2	%
	Prior to Production	475	m
	Daily Advance	9	m/day
	In Ore	85	%
	Support	Rock Bolts	
	In Footwall	15	%
	Support	Rock Bolts	
	In Hanging Wall	0	%
	Support	n/a	
X-CUTS	Face Width	4.5	m
	Face Height	4	m
	Gradient	2	%
	Prior to Production	650	m
	Daily Advance	7	m/day
	Location - Primary Stoping	Footwall	
	Location - Secondary Stoping	n/a	
	Support	Rock Bolts	
	Draw Points and Access Raises		
DRAW POINTS	Average Face Area	n/a	m^2
	Draw Points (Primary)	n/a	per stope
	Draw Points (Secondary)	n/a	per stope
	Average Length	n/a	m
	In Ore	n/a	%
	In Footwall	n/a	%
	In Hanging Wall	n/a	%
	Prior to Production	n/a	m
	During Production	n/a	m/day
ACCESS RAISES	Average Face Area	1.5	m^2
	Excavation Method	Slot	
	Lining Material	None Required	
	Compartments (Primary)	1	per raise
	Compartments (Secondary)	0	per raise
	In Ore	100	%
	In Footwall	0	%
	In Hanging Wall	0	%
	Prior to Production	50	m
	During Production	0.7	m/day

	Ore Passes and Ventilation Raises		
ORE PASSES	Face Area	0	m²
	Excavation Method	n/a	
	Location	n/a	
	Lining	n/a	
	Chute	n/a	
	During Production	n/a	m/day
	Prior to Production	n/a	m
	From Year	n/a	
	To Year	n/a	
VENTILATION RAISES	Face Area	25	m²
	Excavation Method	Slot	
	Location	Footwall	
	Lining	None Required	
	During Production	0.23	m/day
	Prior to Production	150	m
	From Year	1	
	To Year	10	
	Supply Prices		
PRODUCTION	Explosives	1.2	USD/kg
	Caps	6	USD/cap
	Boosters	4	USD/booster
	Fuse	0.5	USD/m
DEVELOPMENT	Explosives	1.2	USD/kg
	Caps	2	USD/cap
	Boosters	2.5	USD/booster
	Fuse	0.5	USD/m
GENERAL	Steel Beam	0.79	USD/kg
	Timber	300.05	USD/m3
	Lagging	264.24	USD/m3
	Cement	160	USD/tonne
	Shotcrete	350	USD/m3
	Ventilation Tubing	20	USD/m
	Steel Liner Plate	2.29	USD/kg
	Diesel Fuel	1	USD/litre
	Electricity	0.1	USD/kWh
	Hourly Wages		
EMPLOYEE	Burden Factor	45	%
	Operator Efficiency	85	%
	Shift Differential - Second Shift	0	USD/hr
	Shift Differential - Third Shift	0	USD/hr
WAGES	Stope Miner	60	USD/hr
	Development Miner	60	USD/hr
	Mobile Equipment Operator	45	USD/hr
	Hoist Operator	45	USD/hr
	Motorman	45	USD/hr
	Rock Support Miner	55	USD/hr
	Exploration Drill	50	USD/hr
	Crusher Operator	40	USD/hr
	Backfill Plant Operator	40	USD/hr
	Electrician	50	USD/hr
	Mechanic	50	USD/hr
	Maintenance Worker	40	USD/hr
	Helper	30	USD/hr
	Mine Labourer	40	USD/hr
	Surface Worker	30	USD/hr
	Annual Salaries		
SALARIES	Mine Manager	120,000	USD/yr
	Superintendent	90,000	USD/yr
	Foreman	85,000	USD/yr
	Engineer	70,000	USD/yr
	Geologist	60,000	USD/yr
	Shift Boss	80,000	USD/yr
	Technician	45,000	USD/yr
	Accountant	55,000	USD/yr
	Purchasing	55,000	USD/yr
	Personnel	80,000	USD/yr
	Secretary	40,000	USD/yr
	Clerk	35,000	USD/yr
STAFF	Burden Factor	40	%

Underground Openings		
Type	Pump House #1	
Location	Footwall	
Opening Width	6	m
Opening Height	6	m
Opening Length	13	m
Face Width	6	m
Face Height	6	m
Year Begun	5	
Year Complete	5	
Rock Bolts	100	%
Timber		%
Lagging		%
Shotcrete	100	%
Cement		%
Underground Openings		
Type	Dry Magazine	
Location	Footwall	
Opening Width	6	m
Opening Height	6	m
Opening Length	20	m
Face Width	6	m
Face Height	6	m
Year Begun	2	
Year Complete	2	
Rock Bolts	100	%
Timber		%
Lagging		%
Shotcrete	100	%
Cement		%
Underground Openings		
Type	Dry Workshop	
Location	Footwall	
Opening Width	6	m
Opening Height	6	m
Opening Length	20	m
Face Width	6	m
Face Height	6	m
Year Begun	5	
Year Complete	5	
Rock Bolts	100	%
Timber		%
Lagging		%
Shotcrete	100	%
Cement		%

Appendix 2

EQUIPMENT PARAMETERS FOR COST MODEL

	Drilling and Blasting		
PRIMARY	Drill	Longhole Drills	
	Bit Diameter	8.9	cm
	Explosives Loader	Anfo Loader	
SECONDARY	Drill	n/a	
	Bit Diameter	n/a	cm
	Explosives Loader	n/a	
DRIFTS	Drill	Two Boom Jumbos	
	Bit Diameter	4.8	cm
	Explosives Loader	Anfo Loader	
CONVENTIONAL RAISES	Drill	n/a	
	Bit Diameter	3.2	cm
	Explosives Loader	n/a	
SLOT RAISES	Drill	DTH Hammers	
	Bit Diameter	13.97	cm
	Explosives Loader	Anfo Loader	
	Mucking		
PRIMARY STOPING	Machine	Scoop Tram	
	Capacity	6.5	m^3
	Bucket Fill Factor	85	%
	Time to Fill	1.1	min
	Initial Haul Distance	200	m
SECONDARY STOPING	Machine	n/a	
	Capacity	0	m^3
	Bucket Fill Factor	0	%
	Time to Fill	0	min
	Initial Haul Distance	0	m
DEVELOPMENT	Machine	Scoop Tram	
	Capacity	6.5	m^3
	Bucket Fill Factor	85	%
	Time to Fill	1.1	min
	Initial Haul Distance	50	m
	Stope Haul		
PRIMARY STOPING	Machine Size	6.5	m^3
	Haul Segment 1 - Distance	100	m
	Haul Segment 1 - Gradient	-2	%
	Haul Segment 2 - Distance	100	m
	Haul Segment 2 - Gradient	-2	%
	Haul Segment 3 - Distance		m
	Haul Segment 3 - Gradient		%
	Haul Segment 4 - Distance		m
	Haul Segment 4 - Gradient		%
SECONDARY STOPING	Machine Size		m^3
	Haul Segment 1 - Distance		m
	Haul Segment 1 - Gradient		%
	Haul Segment 2 - Distance		m
	Haul Segment 2 - Gradient		%
	Haul Segment 3 - Distance		m
	Haul Segment 3 - Gradient		%
	Haul Segment 4 - Distance		m
	Haul Segment 4 - Gradient		%
	Development Haul		
EQUIPMENT	Mucker	Scoop Tram	
	Size	6.5	m^3
	Hauler	Rear-Dump Truck	
	Size	50	t
		Av. Distance (m)	**Av. Gradient (%)**
EXCAVATION	Drifts	87.5	-2
	X-Cuts	162.5	-2
	Draw Points	0	0
	Access Raises	60	-2
	Ore Passes	60	-2
	Ventilation Raises	60	-2
	Adits	1200	-12
	Underground Openings	15	-2

	Haul to Surface		
	Machine Size	50	
	Percentage of Total Production	100	
	From Year	2	
	To Year	10	
	Haul Segment 1 - Distance	2520	m
	Haul Segment 1 - Gradient	12	%
	Haul Segment 2 - Distance	300	m
	Haul Segment 2 - Gradient	1	%
	Haul Segment 3 - Distance	n/a	m
	Haul Segment 3 - Gradient	n/a	%
	Haul Segment 4 - Distance	n/a	m
	Haul Segment 4 - Gradient	n/a	%
	Haul Segment 5 - Distance	n/a	m
	Haul Segment 5 - Gradient	n/a	%
	Crushers and Conveyors		
SYSTEM	Percent of Total Production	n/a	%
	From Year	n/a	
	To Year	n/a	
CRUSHERS	Type	n/a	
	Product Size	n/a	cm
	Machine Size	n/a	
	Utilisation	n/a	%
	Bond Work Index	n/a	kWh/tonne
	Motor Horsepower	n/a	hp
	Crusher Capacity	n/a	t/hr
CONVEYORS	Length	n/a	m
	Gradient	n/a	%
	Belt Speed	n/a	m/min
	Utilisation	n/a	%
	Belt Width	n/a	cm
	Skips and Hoists		
SYSTEM	Percent of Total Production	n/a	%
	From Year	n/a	
	To Year	n/a	
PRODUCTION HOIST	Average Distance	n/a	m
	Hoist Speed	n/a	m/s
	Utilisation	n/a	%
	Cycle Time	n/a	sec
	Skip Capacity	n/a	t
	Rope Diameter	n/a	cm
	Drum Diameter	n/a	cm
SERVICE HOIST	Maximum Distance	n/a	m
	Utilisation	n/a	%
	Capacity	n/a	kg
	Horsepower	n/a	hp
	Raise Borers and Electric Cable		
ACCESS RAISES	Bore Diameter	n/a	m
	Average Length	n/a	m
	Utilisation	n/a	%
ORE PASSES	Bore Diameter	n/a	m
	Average Length	n/a	m
	Utilisation	n/a	%
VENTILATION RAISES	Bore Diameter	n/a	m
	Average Length	n/a	m
	Utilisation	n/a	%
ELECTRIC CABLE	Shafts	n/a	USD/m
	Adits	27.69	USD/m
	Drifts & x-cuts	13.45	USD/m
	Raises	13.45	USD/m

	Rockbolters and Scalers			
PRIMARY STOPING	Drilling	Roof Bolters		
	Scaling	Manual		
SECONDARY STOPING	Drilling			
	Scaling			
HORIZONTAL OPENINGS	Drilling	Roof Bolters		
	Scaling	Manual		
VERTICAL OPENINGS	Drilling	Jacklegs		
	Scaling	Manual		
ROCK BOLT LENGTH	Ore	2	m	
	FW	3	m	
	HW	3	m	
ROCK BOLT PRICE	Ore	10	USD/bolt	
	FW	11	USD/bolt	
	HW	11	USD/bolt	
	Backfill Mixers and Pumps			
FILL SPECIFICATIONS	Type	n/a		
	Requirements	n/a	m³/d	
	Stope Preparation	n/a	worker hrs/day	
	Slurry Density	n/a	% solids	
	Cement Content	n/a	%	
	Pumped Fill Density	n/a	kg/m³	
	Settled Fill Density	n/a	kg/m³	
	Flow Rate	n/a	m³/min	
	Average Static Head	n/a	m	
	Number of Pumps	n/a	each	
	Number of Mixers	n/a	each	
PIPE SIZE	Shafts	n/a	cm	
	Adits	n/a	cm	
	Drifts & x-cuts	n/a	cm	
	Raises	n/a	cm	
	Primary Ventilation Fans			
INTAKE	Adit Number	1	Adit Number	1
	Shaft Number	0	Shaft Number	0
	Ventilation Raises	1	raises	
	Average Intake Opening FF	80		
EXHAUST	Adit Number	0	Adit Number	0
	Shaft Number	0	Shaft Number	0
	Ventilation Raises	2	raises	
	Average Intake Opening FF	80		
SYSTEM	Volume per Worker	1.4	cmm	
	F.O.S	25	%	
	Number of Fans	2	each	
	Fan Efficiency	75	%	
	Drainage and Fresh Water Pumps			
FRESH WATER PUMPS	Total Flow Rate	1000	litres/min	
	Average Static Head	94	m	
	Number of Pumps	8	each	
PIPE SIZES	Shafts	25.4	cm	
	Adits	25.4	cm	
	Drifts & x-cuts	6.3	cm	
	Raises	2.54	cm	
DRAINAGE PUMPS	Total Flow Rate	1500	litres/min	
	Average Static Head	74	m	
	Number of Pumps	2	each	
PIPE SIZES	Shafts	25.4	cm	
	Adits	0	cm	
	Drifts & x-cuts	6.3	cm	
	Raises	25.4	cm	
	Ancillary Equipment			
COMPRESSED AIR	Shafts	25	cm	
	Adits	6	cm	
	Drifts & x-cuts	6	cm	
	Raises	3	cm	
PIPE PRICES	Shafts	16	USD/m	
	Adits	0	USD/m	
	Drifts & x-cuts	8	USD/m	
	Raises	4	USD/m	
ANFO TRUCKS	Placement Rate	453.6	kg/min	
SCISSOR LIFTS	Lift Height	9.1	m	
SERVICE VEHICLES	Rating	130	hp	
SHOTCRETERS	Placement Rate	11.5	m³/hr	

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