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Intex Resources ASA	Prop\61061034	Molynor AS		

Tittel

Hurdal Molybdenum Project Scoping Study

Forfatter	Dato	År	Bedrift (Oppdragsgiver og/eller oppdragstaker)
Urbaez Edgar, Caupers Diogo, Wilson Scott,	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 av Aker Kværner på oppdrag for Crew Minerals ASA (før Crew omorganiserte til Intex Resources ASA).

Rapporten er delt i følgende kaptler: 1Open pit design 2. Underground mine design 3. TMF, Environmental and underground mine infrastructure 4. process plant design 5. Costing and preliminary financial model og 6. Conclusions and recommenndations.

Konklusjonene tyder på at det er bedre å utvikle en underjords gruve før et dagbrudd, ford de høyeste gehaltene er mot dypet.

Ellers pekes det på en rekke forhold som må bedre data. Det pekes bl.a. på at borhulls dekningen er ganske grov, det trengs mer data på den mineralogiske siden og avgangsdeponeringen må en få bedre oversikt over.

Oppredningsverket er beregnet til 8 Mt/år, men mye tyder på at dette er for stort.De finansielle beregningene tyder på at prosjektet bør ytterligere utvikles.

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Hurdal Molybdenum Project
Scoping Study

Reference: 61060034
Date October 2007

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1.0 EXECUTIVE SUMMARY

This Scoping Study for the Hurdal Molybdenum Project has been undertaken by the following consultants:

• Open Pit Mining	Edgar Urbaez
• Underground Mine Design	Diogo Caupers
• Underground Infrastructure, Tailings Disposal and Environmental	Scott Wilson
• Process Plant Design and Overall Coordination	Aker Kvaerner

Based on the available information and the data generated during the course of this Scoping Study by the study team, a number of major conclusions have been made regarding the future activities for the Hurdal project. A full list of all the conclusions and recommendations produced by the team can be found in Section 7.0 of this report.

Infill Drilling

The preliminary resource model is based on drilling previously carried out at wide drill hole spacing and therefore there are significant holes in the model where there is a lack of accurate data. It is recommended that infill drilling is carried out as soon as possible on the deposit to produce more assay data for the mineralization. This will not only significantly improve the accuracy of the resource model and the mine planning, but may in fact improve the reserve tonnage and grade.

Open Pit Mining

Current evaluations based on the available data indicate that open pit mining may not be desirable and that an underground mine only, may be more attractive. This is based on economic returns which indicated that the open pit ore, while more readily accessible, may not substantially add to the project NPV as the ore grades closer to the surface are lower then those accessible from an underground mine. In addition it is felt that the environmental permitting for an open pit operation could be significantly more difficult than an underground operation due to the greater land take, large waste dumps and social impact on the local population.

Underground Mining Only

It is felt that there may be potential for a high grade ore body of 100Mt, which could be developed over a period of approximately 15 years from underground. This would allow the possibility of high grading the deposit at say 0.08%Mo cut off with an average ore reserve grade of close to 0.12%Mo which is double that of the Spinifex Ridge Deposit.

Metallurgical Testwork

It is strongly recommended that some metallurgical testwork is carried out on representative samples of the ore as soon as possible, preferable at the same time as the infill drilling takes place. This will help to give some confidence to the process flow sheet and equipment definition, and will also generate information about the likely recoveries of Mo from the Hurdal ore and so the corresponding revenue which may be expected.

Metallurgical testwork will also help to define other components of the ore which may be economically beneficial to the project (copper?) and others which may incur penalties at a smelter.

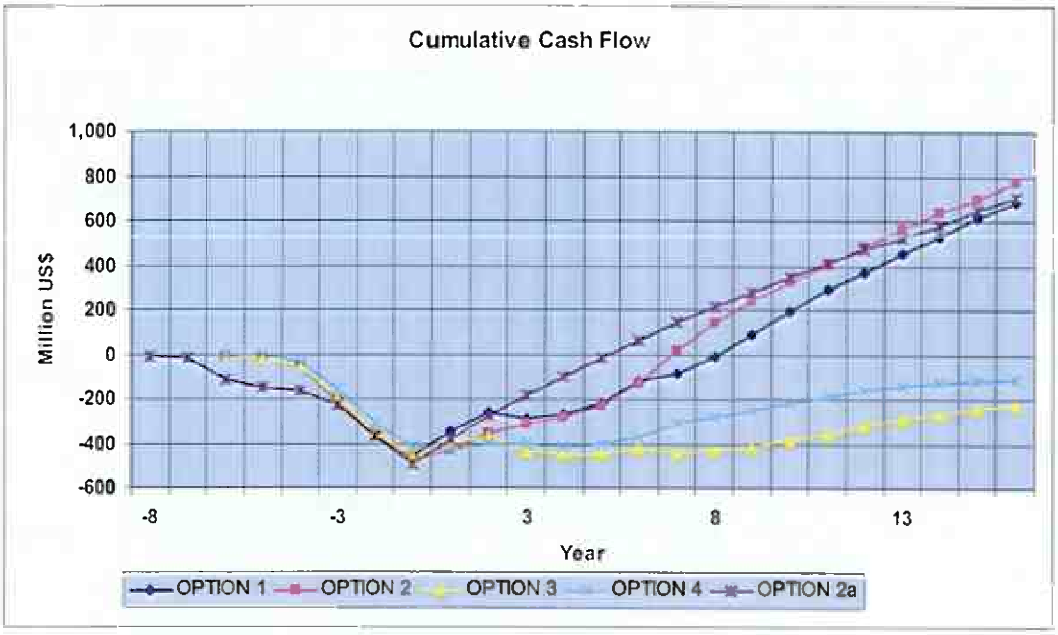
Economic Evaluation

Four open pit / underground mining options were considered and evaluated economically at a preliminary level; these generated a range of returns as follows

(NPV of Cash Flow over Project Life)		NPV $MM	@Rate %	IRR %	Life Years Project	Life Years Mill	Mo $/lb
OPTION 1	325m pit, COG 0.061% MoS2	471	5%	11.9%	41	35	30
OPTION 2	275m pit, COG 0.061% MoS2	515	5%	12.4%	42	36	30
OPTION 3	325m pit, COG 0.091% MoS2	-235	5%		39	33	22
OPTION 4	275m pit, COG 0.091% MoS2	-204	5%		39	33	22
OPTION 2a	275m pit, COG 0.061% MoS2, No open Pit	385	5%	12.0%	38	30	30

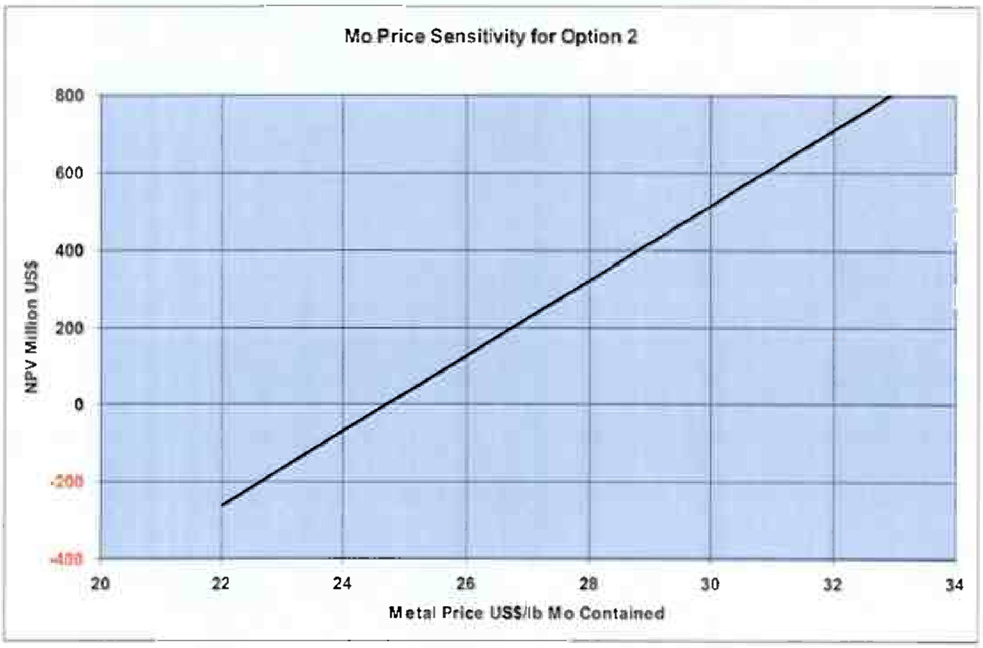
Assumed Offsite Costs for ALL OPTIONS

Roasting	4.0	$/lb Mo
Concentrate Transport	0.40	$/lb Mo
Conversion loss	1.0%	Mo



A series of sensitivities were run for the best option, Option 2. The results are shown graphically below.

The model is most sensitive to Mo price and plant recovery and least to Open Pit grades. The effect of mo price is also shown against actual Mo metal price. The break even Mo price is about US$25/t with the present model



Overall, the preliminary financial calculations carried out as part of this study suggest that the project is worthy of progressing to a further stage of development. This should be done by addressing many of the items and recommendations given in this report which will generate additional information on which future investment decisions could be made.

2.0 OPEN PIT DESIGN

HURDAL MOLYBDENUM PROSPECT
OPEN PIT DESK STUDY

Prepared for:

CREW GOLD CORPORATION
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GLOSSARY OF TERMS and ABBREVIATIONS

%	percent
bcm	bank cubic metre (unit of volume for in-situ rock)
Crew	Crew Gold Corporation
EU	Edgar Urbaez (private consultant)
GEMS	Gemcom Enterprise Mining Systems
h	hour
k	kilo or thousand
kg	kilogram – unit of weight
km	kilometres – unit of distance
LoM	Life of Mine
m^3	cubic metre – unit of volume
M	mega or million
m	metre
MoS_2	molybdenum
Mm^3	million cubic metres
Mt	million tonnes
Mtpa	million tonnes per annum
NPV	Net Present Value
RoM	run-of-mine (ore as mined, and sent for processing)
sg	specific gravity (weight per unit volume of in-situ rock - unbroken)
t	tonne metric (1,000 kg)
tpa	tonnes per annum
tpd	tonnes per day
tpm	tonnes per month
t/m^3	tonnes per cubic metre
USD	United States Dollar – currency
Whittle	Whittle 4X software for strategic analysis and pit optimisation work

HURDAL MOLYBDENUM PROSPECT
OPEN PIT DESK STUDY

1 Introduction

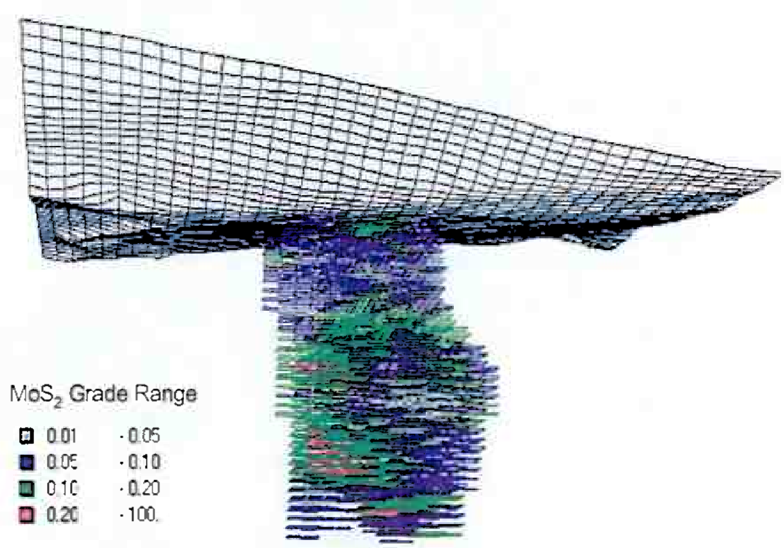
Crew Gold Corporation (Crew) requested Mr. Edgar Urbaez (EU) to provide a high level open pit optimisation desk study for its Hurdal molybdenum prospect in Norway. The scope of the desk study was limited to pit optimisation analysis only and preliminary production schedules. The following sections summarise the main assumptions for the pit optimisation analysis and the production schedules.

2 MINERAL RESOURCE

2.1 Block Model

SRK Consulting (UK) Limited (SRK) has modelled the Hurdal molybdenum mineralisation in April 2007, using the Gemcom Enterprise Mining Software (GEMS). The block dimensions are 50 x 50 x 25 m in the East, North and vertical directions, respectively. A constant density of 2.6 t/m^3 has been used for both waste and mineralised rock types. This resource block model comprises inferred resources **only** and should **not** be used for detailed mine planning. Figure 3.1 illustrates a three-dimensional (3D) view of the Hurdal molybdenum mineralisation.

Figure 0.1 : 3D View of Mineralised Blocks



3. PIT OPTIMISATION ANALYSIS

GEMS was used to create the mineral resource block model for use by the Whittle 4X (Whittle) pit optimising software. The block dimensions are 50 x 50 x 25 m in the East, North and vertical directions, respectively. The surface topography as at April 2007 was the starting surface of the pit optimisation analysis.

3.1 Optimisation Parameters

31.1 Specific Gravity

The specific gravity used in both the mineral resource model in GEMS and the one in Whittle is 2.6 t/m^3. Volumes in GEMS are calculated based on a 'percentage' field in the model, which represents the proportion of mineralisation for each block in the block model. Partially mineralised blocks are therefore recognised and the tonnages of ore-bearing and unmineralised blocks are reported correctly.

However, when a GEMS block model is imported into Whittle, the tonnage and contained metal in each complete block is loaded. The volumes and grades are then calculated from the Whittle block model framework and are reported from complete blocks calculating the contained metal for each block. Consequently, the correct choice of the block dimensions for the Whittle model determines the 'accuracy' of representing the imported block model. Small differences between GEMS tonnages and Whittle tonnages are acceptable. Table 3.1 presents a comparison between the resource block model in GEMS and the imported model in Whittle.

Table 0.1 : Comparison of the Ore Volumes in the GEMS / Whittle Models

Software	Volume Mm^3	Density t/m^3	Tonnage Mt	MoS_2 %
GEMS	103.39	2.6	268.81	0.11
Whittle	103.37	2.6	268.76	0.11

The small differences observed in Table 3.1 are acceptable and normal. They indicate that the block dimensions as selected for Whittle, are adequate. Therefore, any optimisation work performed on this model will be representative of the resource block model built in GEMS.

3.1.2 Slope Angles

An overall slope angle of 45° in all directions has been used. This parameter is irrespective of pit shell depth, and / or whether the wall face / bench is in a mineralised or non-mineralised zone. The zone of weathering has not been defined yet for any of the rock types.

3.1.3 Costs

Whittle requires revenues, mining and processing costs to be specified. These are used to determine the economic final pit.

The reference mining cost used for ore and waste is USD 1.35 /t mined. A cost adjustment factor (CAF) of USD 0.375 /t mined per 25 m vertical lift was applied to the reference mining cost in order to simulate the increase in the haulage cost as the pit gets deeper. Other costs were applied per tonne of ore treated as follows:

- Ore treatment (inclusive of G&A) USD 5.56 / t;
- Sales, royalty, freight USD 6.00 / t; and,
- Waste site rehabilitation USD 0.25 / t.

3.1.4 Mining Dilution and Ore Losses

The block model in Whittle is undiluted. Crew indicated to use 5% for mining dilution and 5% for mining ore losses. These parameters have been applied in the Whittle optimisation without further review.

3.1.5 Revenue

Mo pr. 31.okt $14.5 pr. lb

Crew requested the pit optimisation analysis to be evaluated at two molybdenum prices, USD 30 and 22 / lb of molybdenum, for revenue calculation purposes. These represent some USD 24 and 16 payable / lb of molybdenum, respectively. The average revenue was generated assuming that 84% of the treated molybdenum was recovered.

3.1.6 Ore Production Schedule

A treatment rate of 8.0 Mtpa was specified for the pit optimisation analysis. The ore production rate does have an impact on the cash flows that are generated by Whittle. The effect of reducing, or increasing, the ore treatment rate was not evaluated as part of this study. This should be considered by further work.

3.1.7 Discount Rate

Whittle uses the revenues, costs and ore treatment rate to estimate project cash flows. The cash flow is reported both undiscounted and discounted. The cash flows are calculated pre-tax, with all costs and revenues in real terms, with no inflation or escalation. For the discounted cash flows, a discount rate of 5% has been used to compensate for future economic uncertainties. The effect of applying the discount rate is to reduce the impact of future cash flow and to emphasise the importance of cash flow generated in the early years of the project life.

3.2 Selection of Optimum Pit Shells

3.2.1 General

The Whittle process uses the revenue and cost parameters as specified, to generate a series of incremental pit shells, for progressively increasing metal prices / revenue. The smallest shell is therefore the most profitable. At the economic pit limit, the incremental pit shell is exactly at break even, where the revenue equals the operating costs. This is the economic final pit limit. The smaller nested pits shells are very useful to help decide where mining should be started, as these small pit shells are mining the highest profit areas of the mineral deposit.

Whittle reports the results of each incremental nested pit on two bases, calculating:

i) The undiscounted cash flow; and
ii) The discounted cash flow.

The 'optimum' pit is then usually chosen by inspecting these cash flows and selecting the pit shell with the maximum total cash flow. The maximum undiscounted cash flow is the pit shell where the incremental pit is breaking even, and is therefore the maximum economic pit in today's revenue/cost terms. If a discount rate is used, the pit shell with the maximum discounted cash flow is always somewhat smaller. However, this smaller 'final pit' will be more profitable.

Using the discount factor specified, Whittle produces two cash flows based on different scheduling scenarios. The first case, namely the Best Case, assumes that mining progresses strictly according to a series of incremental nested pit shells. This scenario is optimistic, and is not practical, but it does indicate the highest possible project value that might be achievable. The second case, namely the Worst Case, assumes that mining progresses on a bench by bench basis, mining to the limit of the 'final' pit. This scenario indicates the lowest possible project value. In reality, the pit will operate somewhere between the two Cases, where intermediate and practical cutbacks are defined and then mined in sequence.

If the discount factor is 0%, the cash flows for the Best and the Worst cases will be equal, and therefore the 'optimum' pit shell (maximum cash flow) will be the same.

3.2.2 Optimisation Results – USD 24 / lb (Payable)

Detailed results from the pit optimisation analysis for the USD 24 / lb (Payable) scenario are contained in Appendix. The following observations can be made from these results:

The pit shell with the maximum discounted cash flow of USD 1,710 M, for the Best case, is N° 16. This is the maximum economic pit that can be mined at today's metal prices and assumed costs. Pit shell N° 16 mines a total of 190 Mt of ore at 0.11% molybdenum with 937 Mt of waste. The average strip ratio is 4.9 t:t. The economic mine life is ~24 years;

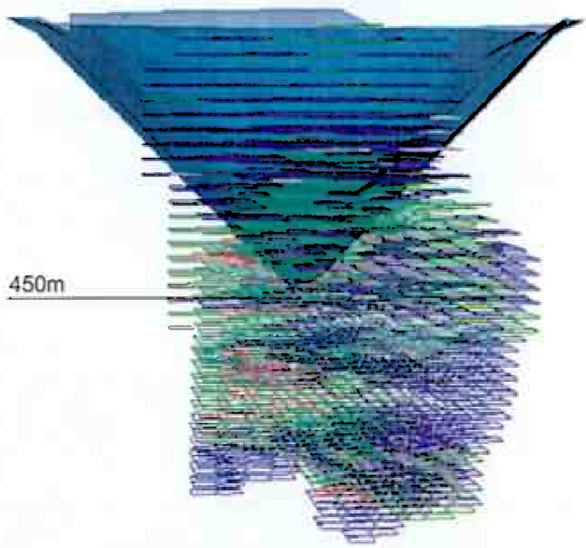
The pit shell with the maximum discounted cash flow of USD 1,193 M, for the Worst case, is N° 10. Pit shell N° 16 mines a total of 114 Mt of ore at 0.10% molybdenum with 364 Mt of waste. The average strip ratio is 3.2 t:t. The economic mine life is ~14 years;

It is of interest to note the considerable potential for underground mining of the Hurdal molybdenum mineralisation. Although it is economic to mine this deposit to a depth of 500 m, assuming the Worst case, optimised pit shell N° 6 has been selected as the maximum pit shell of interest for mining by open pit methods. Pit shell N° 6 mines a total of 91 Mt of ore at 0.09% molybdenum with 215 Mt of waste to a final pit depth of ~450 m. The average strip ratio is 2.4 t:t. The economic mine life is ~11 years. Clearly, the larger pit shells have much lower incremental profitability, although they are still economic to mine;

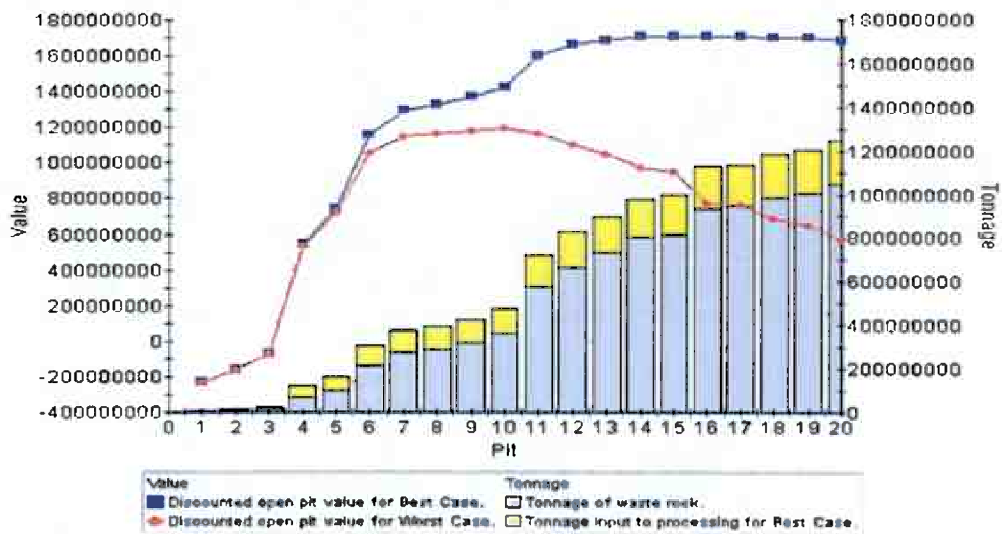
Pit shell N° 6 represents a low risk surface, where the Best and Worst cases respective cash flows are relatively similar. It is still profitable at 60% of the value of the evaluated USD 30 / lb molybdenum price.

The optimised pit surface N° 6 is illustrated in Figure 3.1.

Figure 0.1 : USD 24 / lb (Payable) Optimised Pit Surface



The data from the pit optimisation results table for the USD 24 / lb (Payable) scenario in Appendix is illustrated in Figure 3.2, where the tonnages of ore and waste for pit shells N° 1 to N° 20 are plotted. Also plotted are the Best and Worst case discounted cash flows for each pit shell.



The shape of the tonnage curves in Figure 3.2 is of great interest to the Mine Planners. It is observed that there are distinct breaks or jumps in ore tonnage for progressive pit shells. These occur at pit shell numbers 4, 6, 11 and 16. These progressive pit shells correspond directly with increasing the metal prices. These break points can be used by the Mine Planners to decide logical mining phases for longer term stripping and for developing the Hurdal over the very long term.

3.2.3 Optimisation Results – USD 16 / lb (Payable)

Similarly, detailed results from the pit optimisation analysis for the USD 16 / lb (Payable) scenario are contained in Appendix. The following observations can be made from these results:

The pit shell with the maximum discounted cash flow of USD 348 M, for the Best case, is N° 14. This is the maximum economic pit that can be mined at today's metal prices and assumed costs. Pit shell N° 14 mines a total of 107 Mt of ore at 0.09% molybdenum with 317 Mt of waste. The average strip ratio is 3.0 t:t. The economic mine life is ~13 years;

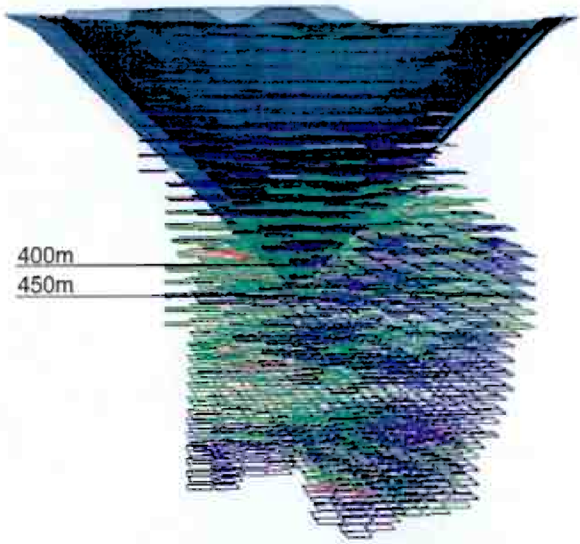
The pit shell with the maximum discounted cash flow of USD 208 M, for the Worst case, is N° 11. Pit shell N° 11 mines a total of 91 Mt of ore at 0.09% molybdenum with 215 Mt of waste. The average strip ratio is 2.4 t:t. The economic mine life is ~11 years;

Similarly to the USD 24 / lb (Payable) scenario, optimised pit shell N° 10 has been selected as the maximum pit shell of interest for mining by open pit methods. This pit shell surface corresponds to a molybdenum price of USD 18 / lb, which is on the same terms as Pit shell N° 6 in the USD 24 / lb (Payable) scenario;

Pit shell N° 10 mines a total of 78 Mt of ore at 0.08% molybdenum with 147 Mt of waste to a final pit depth of ~450 m. The average strip ratio is 1.9 t:t. The economic mine life is ~10 years;

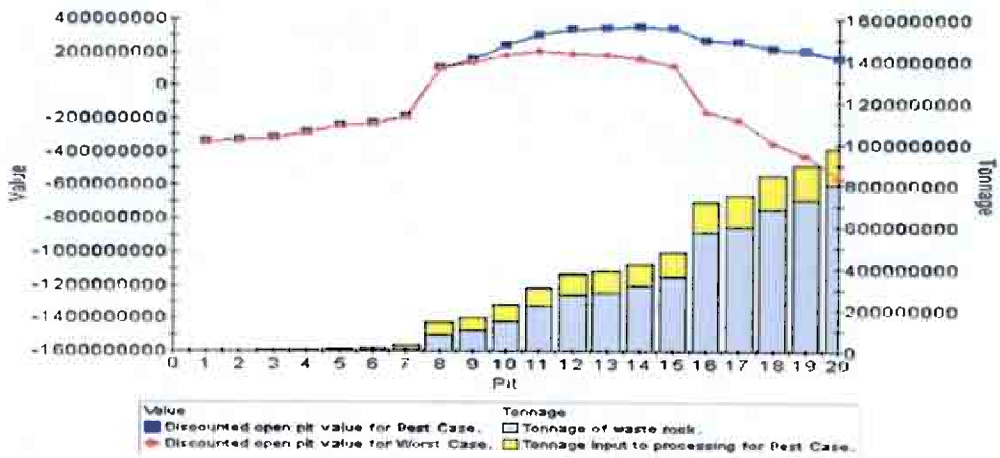
Pit shell N° 10 represents a low risk surface, where 90% of the Worst case discounted cash flow value is realised. It is clearly profitable at 80% of the value of the evaluated USD 22 / lb molybdenum price.

Figure 0.3 : USD 16 / lb (Payable) Optimised Pit Surface



The data from the pit optimisation results table for the USD 16 / lb (Payable) scenario in Appendix is illustrated in Figure 3.4, where the tonnages of ore and waste for pit shells N° 1 to N° 20 are plotted. Also plotted are the Best and Worst case discounted cash flows for each pit shell.

Figure 0.4 : USD 16 / lb (Payable) Graph of Whittle Results - Pit by Pit



4 PRODUCTION SCHEDULES

4.1 General

The Hurdal molybdenum mineralisation extends beyond a depth of 500m. Consequently, Crew will be investigating the synergy between mining this deposit by open pit and underground methods. Determining this synergy is a highly iterative process. At this preliminary stage, two optimised pit shells are selected based on their depth. One of the optimised pit shells has an approximate pit depth of 275m with a pit base diameter of 100m, and the other is 325m deep with a pit base diameter of 50m. It is important to note that both of these optimise pit shells are within the selected ones for the USD 24 and 16 / lb cases. In other words, the 275m and 325m optimised pit shells represent profitable, lower risk surfaces based on the 'Optimisation Parameters' in section 3.1. Production schedules for both optimised it shells, and main assumptions, are presented in Appendix.

4.2 Main Assumptions

Rock excavating activities will be conducted each year for 345 days, on average. The mine will operate three 8-hour shifts per day, seven days a week. The approximate total effective work hours per day is 16, after discounting time for meals, shift changes, shift briefing, labour efficiency, mechanical availability, and equipment utilisation, among other routine tasks.

1 mill. tonn/år

Mining activities will be carried out by truck and shovel method. In order to provide the plant with 8 Mtpa of ore, on average, at least some 23,188 tpd of ore must be produced from the pit. No ramp-up production for the plant has been assumed at this stage.

Given the geometry of the optimised pit shells, ore will be excavated by two shovels. On average, shovels will fill trucks in 5 passes. A bucket fill factor of 90% is used for each shovel.

4.3 Whittle Scheduler

The Whittle software does have a very basic mine scheduling capability. It should be stressed that this should only be used for long term scheduling. For detailed, short term scheduling, an alternative more suitable package should be used. For this preliminary analysis, the Whittle scheduler has been used.

The Whittle Milawa NPV option is used to generate the production schedules. Generally, the Milawa NPV scheduling algorithm will try to maximise Net Present Value (NPV). The effect is a production schedule that meets the plant constraint but that needs to be manipulated to simulate waste pre-stripping and to smooth the annual tonnages of material being mined. An external adjustment is necessary in a way that the total tonnage mined per year is kept constant for a given period of time, starting high and then dropping as mining progresses. This approach tries to maximise the utilisation of the mining equipment while supplying the ore production required.

Each optimised pit shell is mined on a bench by bench basis. However, the mining of an adjacent optimised pit shell can commence before a given optimised pit shell is completely depleted, thus allowing for the concurrent mining of different areas in the pit. This approach will not generally yield the best NPV, but represents a more realistic scenario. The preliminary production schedules are at a high level and have not been checked against operational constraints. Consequently, these production schedules may not be practical.

4.4 275m Pit Shell – Production Schedule

An optimised pit shell with a depth of some 100m was used to simulate a starter pit. This surface provides quick access to ore at the lowest cash cost possible. The maximum annual mining rates used to generate the schedules were as follows:

18 Mtpa for the first 2 years for waste stripping;
18 Mtpa for the first 3 years of ore production; and,
8.5 Mtpa from year 6 until the end.

In general, the 275m Pit Shell production schedule yields some 50.6 Mt of ore, at an average molybdenum grade of 0.07%, and in-situ molybdenum tonnes of some 37,530, of the in-pit inferred resources. The average waste strip ratio is some 1.3 t:t for a Life of Mine (LoM) of 6.3 years (excluding the first 2-years of waste pre-stripping).

4.5 325m Pit Shell – Production Schedule

Similarly to the 275m pit shell case, an optimised pit shell with a depth of some 100m was used to simulate a starter pit. Additionally, the 275m pit shell was used as an intermediate cut back between the starter pit and the 325m pit shell. The minimum mining width between these last two surfaces has not been checked since the geotechnical parameters are still very general in nature. The maximum annual mining rates used to generate the schedules were as follows:

18 Mtpa for the first 2 years for waste stripping:
18 Mtpa for the first 3 years of ore production;
16 Mtpa from year 6 until year 9: and,
8.5 Mtpa from year 10 until the end.

In general, the 325m Pit Shell production schedule yields some 64.0 Mt of ore, at an average molybdenum grade of 0.08%, and in-situ molybdenum tonnes of some 48,816, of the in-pit inferred resources. The average waste strip ratio is some 1.5 t:t for a Life of Mine (LoM) of 8 years (excluding the first 2-years of waste pre-stripping).

5. Conclusions and Recommendations

Clearly, considering only the inferred molybdenum mineral resource, the Hurdal mineral deposit can profitably be mined by open pit methods. At an ore treatment rate of 8 Mtpa, the expected mine life can be between 10 to 24 years, depending on the selected optimised pit shell. Even at molybdenum prices as low as USD 18/ lb, the Hurdal mineral deposit represents an attractive prospect with reasonable mine life by open pit methods.

This open pit desk study is largely based on the information in the mineral resource block model. Therefore, it is of high importance to confirm / understand the applied resource grade estimation methodology. In due course, these inferred mineral resource must be proven up and reclassified as measured or indicated, at least for the mineral resource falling within the optimised pit shells of interest described in this report.

Further analysis should be conducted in order to determine what treatment rate is suitable for the available resource and what ramp up production can be expected. Also, depending on the mine life, an analysis of owner vs. contractor operated should be performed.

APPENDIX

USD 30 / lb (USD 24 / lb Payable) Pit Optimisation Results

Pit Shell #	MoS_2 Price USD / lb	Cashflow Best M USD	Cashflow Worst M USD	Total Mined Mt	Waste Mt	Ore Mt	MoS_2 Grade %	Strip Ratio t:t	Mine Life Years
1	12.0	-232.7	-232.7	6.8	3.5	3.3	0.10	1.0	0.4
2	13.2	-162.5	-162.5	12.2	5.7	6.4	0.09	0.9	0.8
3	14.4	-69.3	-69.3	22.5	11.3	11.3	0.08	1.0	1.4
4	15.6	540.0	535.1	118.3	67.7	50.6	0.07	1.3	6.3
5	16.8	744.5	720.9	162.5	98.5	64.0	0.08	1.5	8.0
6	18.0	1,155.7	1,059.9	306.4	215.0	91.4	0.09	2.4	11.4
7	19.2	1,294.4	1,148.3	375.4	273.7	101.7	0.09	2.7	12.7
8	20.4	1,323.8	1,162.7	392.9	288.4	104.4	0.09	2.8	13.1
9	21.6	1,367.4	1,182.2	424.5	317.2	107.4	0.09	3.0	13.4
10	22.8	1,425.0	1,193.3	477.7	364.2	113.6	0.10	3.2	14.2
11	24.0	1,599.9	1,162.3	720.6	574.7	145.9	0.10	3.9	18.2
12	25.2	1,658.7	1,103.7	827.3	667.6	159.7	0.10	4.2	20.0
13	26.4	1,686.5	1,047.4	899.4	731.3	168.1	0.10	4.4	21.0
14	27.6	1,706.2	970.8	976.9	801.1	175.9	0.11	4.6	22.0
15	28.8	1,709.3	945.3	997.1	817.8	179.2	0.11	4.6	22.4
16	30.0	1,710.1	776.4	1,127.6	937.3	190.3	0.11	4.9	23.8
17	31.2	1,709.0	762.1	1,137.3	946.3	191.1	0.11	5.0	23.9
18	32.4	1,701.9	689.1	1,183.0	987.1	195.9	0.11	5.0	24.5
19	33.6	1,697.5	653.1	1,204.4	1,005.8	198.6	0.11	5.1	24.8
20	34.8	1,684.4	571.6	1,250.5	1,046.5	203.9	0.11	5.1	25.5
21	36.0	1,671.2	492.4	1,296.3	1,089.3	207.0	0.11	5.3	25.9
22	37.2	1,667.6	471.8	1,306.7	1,098.7	208.0	0.11	5.3	26.0
23	38.4	1,649.6	387.3	1,352.6	1,141.3	211.4	0.11	5.4	26.4
24	39.6	1,648.6	383.1	1,354.5	1,142.6	211.8	0.11	5.4	26.5
25	42.0	1,616.9	243.7	1,421.0	1,202.7	218.4	0.11	5.5	27.3
26	43.2	1,603.8	194.6	1,444.8	1,224.0	220.7	0.11	5.6	27.6
27	44.4	1,603.3	193.3	1,445.4	1,224.5	220.9	0.11	5.5	27.6
28	45.6	1,583.2	116.7	1,480.0	1,255.7	224.3	0.11	5.6	28.0
29	48.0	1,523.9	-107.1	1,583.3	1,351.8	231.5	0.11	5.8	28.9
30	51.6	1,512.1	-150.9	1,602.3	1,369.3	233.0	0.11	5.9	29.1
31	52.8	1,505.2	-175.7	1,613.4	1,380.1	233.3	0.11	5.9	29.2
32	54.0	1,500.1	-190.1	1,620.1	1,386.2	233.9	0.11	5.9	29.2
33	55.2	1,476.2	-268.6	1,653.4	1,416.8	236.5	0.11	6.0	29.6
34	57.6	1,460.8	-319.6	1,675.1	1,437.2	237.9	0.11	6.0	29.7
35	60.0	1,445.1	-372.0	1,696.7	1,457.4	239.3	0.11	6.1	29.9

USD 22 / lb (USD 16 / lb Payable) Pit Optimisation Results

Pit Shell #	MoS_2 Price USD / lb	Cashflow Best M USD	Cashflow Worst M USD	Total Mined Mt	Waste Mt	Ore Mt	MoS_2 Grade %	Strip Ratio t:t	Mine Life Years
1	9.7	-340.8	-340.8	0.5	0.2	0.3	0.14	0.7	0.0
2	10.6	-334.3	-334.3	1.0	0.5	0.5	0.14	1.0	0.1
3	11.4	-319.3	-319.3	2.8	1.7	1.1	0.12	1.5	0.1
4	12.3	-279.7	-279.7	7.0	3.5	3.5	0.10	1.0	0.4
5	13.2	-243.4	-243.4	12.2	5.7	6.4	0.09	0.9	0.8
6	14.1	-229.7	-229.7	14.5	6.8	7.7	0.09	0.9	1.0
7	15.0	-182.5	-182.7	26.6	13.6	13.0	0.08	1.1	1.6
8	15.8	110.0	104.4	136.7	80.4	56.3	0.08	1.4	7.0
9	16.7	158.4	139.7	162.5	98.5	64.0	0.08	1.5	8.0
10	17.6	236.8	186.6	224.4	146.9	77.5	0.08	1.9	9.7
11	18.5	304.0	207.7	306.4	215.1	91.3	0.09	2.4	11.4
12	19.4	337.6	193.5	375.4	273.8	101.6	0.09	2.7	12.7
13	20.2	343.1	186.7	390.3	286.3	104.1	0.09	2.8	13.0
14	22.0	348.3	166.2	424.5	317.3	107.2	0.09	3.0	13.4
15	22.9	345.7	118.7	477.7	364.3	113.4	0.10	3.2	14.2
16	23.8	266.2	-160.8	720.6	574.8	145.8	0.10	3.9	18.2
17	24.6	257.7	-204.1	752.1	601.2	150.8	0.10	4.0	18.9
18	25.5	221.9	-344.1	848.3	685.0	163.3	0.10	4.2	20.4
19	26.4	202.8	-423.1	899.4	731.4	168.0	0.10	4.4	21.0
20	27.3	164.8	-553.1	976.9	801.2	175.7	0.11	4.6	22.0
21	28.2	154.2	-588.6	995.6	816.6	179.0	0.11	4.6	22.4
22	29.0	153.2	-591.5	997.1	817.9	179.1	0.11	4.6	22.4
23	29.9	83.0	-820.3	1,119.7	930.2	189.5	0.11	4.9	23.7
24	30.8	77.8	-837.2	1,128.4	938.2	190.2	0.11	4.9	23.8
25	31.7	54.4	-908.2	1,162.3	968.0	194.3	0.11	5.0	24.3
26	32.6	41.9	-949.6	1,183.0	987.4	195.6	0.11	5.1	24.5
27	33.4	27.0	-995.2	1,204.4	1,006.1	198.3	0.11	5.1	24.8
28	34.3	-8.2	-1,097	1,250.5	1,046.8	203.6	0.11	5.1	25.5
29	35.2	-32.3	-1,169	1,283.9	1,077.9	206.1	0.11	5.2	25.8
30	37.0	-40.9	-1,195	1,296.3	1,089.5	206.7	0.11	5.3	25.8
31	37.8	-57.7	-1,244	1,317.7	1,108.7	209.0	0.11	5.3	26.1
32	38.7	-84.3	-1,322	1,352.6	1,141.5	211.1	0.11	5.4	26.4
33	39.6	-86.2	-1,326	1,354.5	1,142.9	211.5	0.11	5.4	26.4
34	40.5	-94.6	-1,351	1,364.2	1,151.3	212.9	0.11	5.4	26.6
35	42.2	-141.7	-1,487	1,421.0	1,202.9	218.1	0.11	5.5	27.3

275m Pit Shell

UNIT CONVERSIONS		
Currency		USD
Operating days in a year	days	345
Shifts per day	#	3
Hours per shift	Hrs	8
Time for meal	Hrs	0.5
Time for shift change, etc	Hrs	0.5
Work hours in a day	Hrs	21
Labour Efficiency	%	88%
Mechanical availability	%	85%
Equipment utilisation	%	90%
Effective work hours in a day	Hrs	16

LOADING &LOADING & HAULING PARAMETERS		
Loader ore rate	tpd	23,188
Loader cycles	cycles/hr	50
Truck position - arrival	min	0.5
Truck position - departu	min	0.5
Truck queue	min	4.2
Truck dump	min	0.5
Loader passes 2 fill truck	#	5
Bucket fill factor	%	80%
Truck fill factor	%	95%
Dump sites	#	1

PRODUCTION SCHEDULE		Total	Period -2	Period -1	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8
Ore mined to Plant	t	53,608,524			8,000,000	8,000,000	8,000,000	8,000,000	8,000,000	8,000,000	2,608,524	
Moly grade to Plant	%	0.07%			0.08%	0.07%	0.06%	0.05%	0.07%	0.10%	0.12%	
Waste mined	t	67,664,558	18,050,000	18,050,000	10,005,938	10,027,854	10,047,826	1,170,353	204,605	99,955	8,027	
Total mined	t	118,273,082	18,050,000	18,050,000	18,005,938	18,027,854	18,047,826	9,170,353	8,204,605	8,099,955	2,616,551	
Pit operating days in a year	days	345	345	345	345	345	345	345	345	345	112	
Mine life	years	6.3										

OPERATING COSTS		
Ore density	t/bcm	2.6
Ore swell factor	%	55%
Waste density	t/bcm	2.6
Waste swell factor	%	55%

		Period -2	Period -1	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8
Mining CAF - ore		1.39	1.43	1.48	1.52	2.19	2.67	3.02	3.46	4.04	
Mining CAF - waste		1.39	1.43	1.48	1.52	2.19	2.67	3.02	3.46	4.04	
Unit mining cost - ore	$/t mined	1.87	1.93	1.99	2.05	2.96	3.60	4.07	4.67	5.45	
Unit mining cost - waste	$/t mined	1.87	1.93	1.99	2.05	2.96	3.60	4.07	4.67	5.45	
Total Daily Production	t	52,319	52,319	52,191	52,255	52,313	26,581	23,781	23,478	23,260	
Daily Loading Capacity	t	56,994	56,994	56,994	56,994	56,994	31,529	31,529	31,529	31,529	
Daily Excess Capacity	t	4,676	4,676	4,803	4,740	4,682	4,948	7,747	8,051	8,269	

CAPITAL EXPENDITURE

Equipment Required	Size	Period -3	Period -2	Period -1	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8
Hydraulic Shovel	13.0 cu. m.		2	2	2	2	2	2	2	2	2	
Hydraulic Shovel	21.0 cu. m.		1	1	1	1	1					
Rear-Dump Trucks	86 tonne		18	18	20	22	22	12	12	12	14	
Wheel loader	2.7 cu. m.		1	1	1	1	1	1	1	1	1	
Bulldozers	179 kW		5	5	5	5	5	4	4	4	4	
Graders	112 kW		2	2	2	2	2	2	2	2	2	
Water Tankers	53,000 lt		1	1	1	1	1	1	1	1	1	
Surface Service / Tyre Trucks			4	4	4	4	4	3	3	3	3	
Powder Buggy (gvw)			8	8	8	8	8	5	4	4	4	
Light Plants	22 hp		8	8	8	8	8	7	7	7	7	
Pumps	56kW		2	2	2	2	2	2	2	2	2	
Pick-up Trucks			12	12	12	12	12	12	12	12	12	
Rotary Drills	279 mm		3	3	3	3	3	2	2	2	2	

Equipment Purchase Schedule		Unit Cost	Period -3										
Hydraulic Shovel 1	$	2,635,000	2										
Hydraulic Shovel 2	$	4,398,000	1										
Rear-Dump Trucks	$	1,148,000	18		2	2							
Wheel loader	$	550,000	1										
Bulldozers	$	390,000	5					4					
Graders	$	310,000	2				2						
Water Tankers	$	630,000	1										
Surface Service / Tyre Trucks	$	72,000	4					3					
Powder Buggy (gvw)	$	35,000	8				7						
Light Plants	$	31,000	8				7						
Pumps	$	30,140	2				2						
Pick-up Trucks	$	22,500	12				12						

325m Pit Shell

UNIT CONVERSIONS

Currency		USD
Operating days in a year	days	345
Shifts per day	#	3
Hours per shift	Hrs	8
Time for meal	Hrs	0.5
Time for shift change, etc	Hrs	0.5
Work hours in a day	Hrs	21
Labour Efficiency	%	85%
Mechanical availability	%	85%
Equipment utilisation	%	90%
Effective work hours in a day	Hrs	18

LOADING / LOADING & HAULING PARAMETERS

Loader ore rate	tph	23,158
Loader cycles	cycles/hr	50
Truck position - arrival	min	0.5
Truck position - departure	min	0.5
Truck queue	min	4.2
Truck dump	min	0.5
Loader passes 2 fill truck	#	5
Bucket fill factor	%	90%
Truck fill factor	%	95%
Dump sites	#	1

PRODUCTION SCHEDULE

		Total	Period -2	Period -1	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8
Ore mined to Plant	t	64,017,953			8,000,000	8,000,000	8,000,000	8,000,000	8,000,000	8,000,000	8,000,000	8,000,000
Moly grade to Plant	%	0.08%			0.08%	0.07%	0.08%	0.10%	0.07%	0.10%	0.08%	0.10%
Waste mined	t	98,519,226	18,950,000	18,050,000	10,305,938	10,027,854	10,647,826	8,564,855	7,403,107	7,498,457	8,300,000	368,000
Total mined	t	162,537,179	18,950,000	19,050,000	18,305,938	18,027,854	18,647,826	16,564,855	15,403,107	15,498,457	16,300,000	8,368,000
Pit operating days in a year	days	345	345	345	345	345	345	345	345	345	345	345
Mine life	years	8.3										

OPERATING COSTS

Ore density	t/bcm	2.6
Ore swell factor	%	55%
Waste density	t/bcm	2.6
Waste swell factor	%	55%

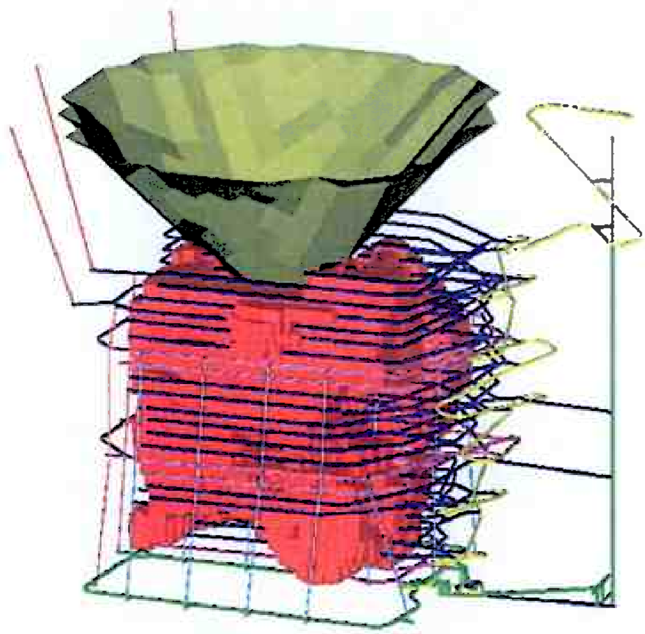
		Period -2	Period -1	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8
Mining CAF - ore		1.39	1.43	1.48	1.52	2.19	2.67	3.02	3.48	3.84	3.94
Mining CAF - waste		1.39	1.43	1.48	1.52	2.19	2.67	3.02	3.48	3.84	3.94
Unit mining cost - ore	$/t mined	1.87	1.93	1.99	2.05	2.96	3.60	4.07	4.67	5.19	5.32
Unit mining cost - waste	$/t mined	1.87	1.93	1.99	2.05	2.96	3.60	4.07	4.67	5.19	5.32
Total Daily Production	t	52,319	52,319	52,191	52,258	52,313	48,026	45,226	44,923	47,246	24,249
Daily Loading Capacity	t	56,994	56,994	56,994	56,994	56,994	56,994	56,994	56,994	56,994	31,529
Daily Excess Capacity	t	4,675	4,675	4,803	4,736	4,681	8,968	11,768	12,071	9,748	7,279

CAPITAL EXPENDITURE

Equipment Required	Size	Period -2	Period -1	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8
Hydraulic Shovel	13.0 cu. m.	2	2	2	2	2	2	2	2	2	2
Hydraulic Shovel	21.0 cu. m.	1	1	1	1	1	1	1	1	1	
Rear Dump Trucks	96 tonne	13	13	20	22	21	22	22	23	21	11
Wheel loader	2.7 cu. m.	1	1	1	1	1	1	1	1	1	1
Bulldozers	179 kW	5	5	5	5	5	5	5	5	5	4
Graders	112 kW	2	2	2	2	2	2	2	2	2	4
Water Tankers	53,000 lt	1	1	1	1	1	1	1	1	1	1
Surface Service / Tyre Trucks		4	4	4	4	4	4	4	4	4	3
Powder Buggy (gvw)		5	5	5	5	5	5	5	5	5	3
Light Plants	22 hp	5	5	5	5	5	5	5	5	5	7
Pumps	56 kW	2	2	2	2	2	2	2	2	2	2
Pick-up Trucks		12	12	12	12	12	11	12	12	12	12
Rotary Drills	279 mm	3	3	3	3	3	3	3	3	3	2

Equipment Purchase Schedule		Unit Cost	Period -3	Period -2	Period -1	Period 1	Period 2	Period 3	Period 4	Period 5	Period 6	Period 7	Period 8
Hydraulic Shovel 1	$	2,635,000	2										
Hydraulic Shovel 2	$	4,398,000	1										
Rear Dump Trucks	$	1,149,000	13		2	2				1	4		
Wheel loader	$	582,000	1							1			
Bulldozers	$	380,500	5					5					
Graders	$	310,000	2				2					2	
Water Tankers	$	630,000	1										
Surface Service / Tyre Trucks	$	72,000	4					4					
Powder Buggy (gvw)	$	38,000	5				5					4	
Light Plants	$	31,000	5				5					7	
Pumps	$	30,140	2				2						
Pick-up Trucks	$	22,500	12				12						

3.0 UNDERGROUND MINE DESIGN



Report prepared by Diogo Caupers – July 2007

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Executive Summary

This report has been prepared by Diogo Caupers at the request of Crew Gold in order that backfill mining method could be analyzed for the Hurdal Molybdenum deposit. The objective of this study is to review the possibility of mining this orebody with an alternative method to SLC (initial study done by SRK), in order to minimize the amount of tailings disposal on surface and reduce the subsidence area, thereby minimizing the environmental impact of the project. Also the use of a backfill method will allow that the open-pit could be use for tailings disposal after the surface mining is finished.

This report focuses on the underground operation below the initial open-pit that will mine the resource from surface up to the -75m level (325 m pit) or –25m level (275m pit).

In order to achieve the above goals of minimising tailings disposal on surface and reducing subsidence, the sub-level open stope method was chosen, because it uses backfill, with a reasonable productivity and a low cost when compared with other backfill methods. On this report the following pillars were left to the open-pit:

- Minimum vertical pillar of 25 m;
- Minimum horizontal pillar of 36 meters;

For each open-pit depth, a different cut-off was used in order to reflect two metal prices – 24$US payable, and 16$US payable, defining 4 options:

- Option 1 – 325 m depth pit with $US 24 payable (cut-off 0.061% MoS2);
- Option 2 - 275 m depth pit with $US 24 payable (cut-off 0.061% MoS2);
- Option 3 - 325 m depth pit with $US 16 payable (cut-off 0.091% MoS2);
- Option 4 - 275 m depth pit with $US 16 payable (cut-off 0.091% MoS2);

A stacked Bench&Fill method with primary and secondary stope mining with access pillars was chosen because it improves productivity, and it increases the number of stopes available to mine simultaneously. In order for this mining method to work good ground conditions must be found.

If ground conditions are worse than what is expected, than the upper drifts should be enlarge to full width, and thigh fill must be used in order that secondary stope mining is possible on the same level as the primary stope mining. This variation of the mining method will increase the mining costs, but productivity can be maintained.

Underground capital cost, for mobile equipment and mine development, is expected to reach €119 M$US for Option 1 and 3, and 101 M$US for Option 2 and 4. Average operational mining cost is expected to vary between 16.6$US/t for Option 2 to 17.4 €/t for Option 3.

Yearly production target is predicted to be 6 Mt/year, with 3 mining levels operating for options 1 and 2, and 5 Mt/year for option 3 and 4. A 50% recovery of the two sill-pillars was assumed.

A 8.0 Mt/year process plant capacity was considered, due to the initial bigger production targets on the open-pit. Top-up of the underground production above the underground production targets can be achieve by adding low grade that could be stocked from the initial open-pit production years.

Project life for the underground operation is expected to be +/-27 years, with a total project life of 36 years.

Figure 1 summarizes the key results for the underground mining options analysed on this report.

	Option 1 325 ($US24)	Option 2 275 ($US24)	Option 3 325 ($US16)	Option 4 275 ($US16)
Underground Production (Mtons)	157.6	166.9	110.9	116.4
% MoS2	0.122	0.121	0.138	0.137
Flat Develoment (m)	64,515	65,317	63,181	63,181
Vertical Development (m)	26,566	28,216	23,468	24,020
Opex ($US/t)	16.88	16.57	17.43	17.34
Development Capex (M$US)	93.8	93.8	90.9	90.9
Underground Mobile Equip Capex (M$US)	119.1	119.1	100.6	100.6
Plant Feed (Mtons)	221.6	217.5	174.9	167.2
Plant Feed (%MoS2)	0.109	0.110	0.116	0.118
Backfill volume (Mm3)	59.6	63.0	41.7	43.8
Tailings to TMF (Mtons)	131.9	122.4	112.2	101.4

Figure 1 – Main results for all options

Hurdal project is sensitive to production rate, head grade, and operational cost. Any changes to these outputs will have a big effect on the financial results of the project.

1. Introduction

The consultant was asked to analyse the mine design using sub-level open stoping to eliminate the subsidence due to a SLC operation in conjunction with the possibility of investigating the feasibility of tailings disposal in the completed open-pit to decrease the amount of tailings stored in the TMF. This option can only be viable with a backfill method that has good productivity and low cost.

2. SLOS mining method design

2.1 Orebody analysis

Considering an overall cost of $US 25.7/t with two different payable prices of $US 24/lb and $US 16/lb, it was possible to determine the cut-off grade (figure 2) and the surface that includes all the blocks that produces a net revenue bigger or equal than the cost.

Revenue	$/lb	$/t
Commodity Price ($/tonne metal)	**$30.00**	$66,120
Sales, Royalty, freight ($/tonne metal)	$6.00	$13,224
Payable	$24.00	$52,896
Payable metal before plant	$20.16	$44,433
Payable metal before mine	$19.15	$42,211
$ US/Euro	1.3	
Mining Parameters		
Plant recovery (%)	84.0%	
Mining Recovery (%)	95.0%	
Dilution (%)	7.5%	
Costs		
Ore Mining cost	$17.84	
Plant costs	$5.56	
Tailings	$0.09	
Concentrate Transport	$0.13	
G&A + Outsite services	$0.33	
Total Costs	$23.9	
Diluted Costs	$25.7	
Underground Design Cut-Off (%)	0.061%	

Revenue	$/lb	$/t
Commodity Price ($/tonne metal)	**$22.00**	$48,488
Sales, Royalty, freight ($/tonne metal)	$6.00	$13,224
Payable	$16.00	$35,264
Payable metal before plant	$13.44	$29,622
Payable metal before mine	$12.77	$28,141
$ US/Euro	1.3	
Mining Parameters		
Plant recovery (%)	84.0%	
Mining Recovery (%)	95.0%	
Dilution (%)	7.5%	
Costs		
Ore Mining cost	$17.84	
Plant costs	$5.56	
Tailings	$0.09	
Concentrate Transport	$0.13	
G&A + Outsite services	$0.33	
Total Costs	$23.9	
Diluted Costs	$25.7	
Underground Design Cut-Off (%)	0.091%	

Figure 2 – Underground Cut-Off grade determination

Analysing a North/South section through Hurdal (figure 3) it is possible to conclude that the ore pipe is quite steep, with a variable horizontal footprint, achieving a maximum depth close to –600 masl. The effect of the cut-off increase is more significant on the South side of the orebody, and on the upper and lower part of the mineralised area.

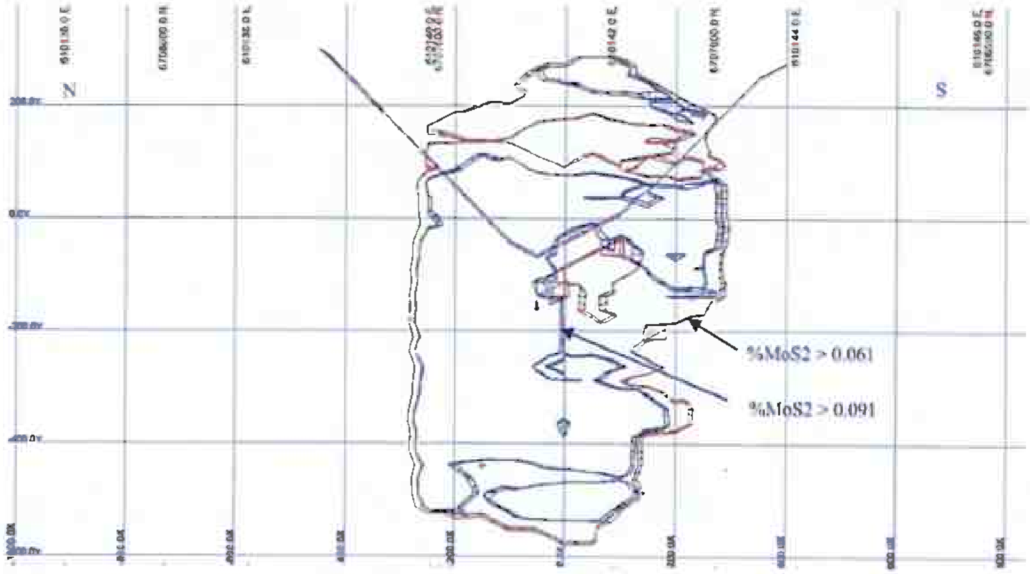


Figure 3 – Hurdal section

This 3D view, visible on figure 4, defines the mineable ore for an operation with a total cost of 25.7 $US/t, for the two metal prices.

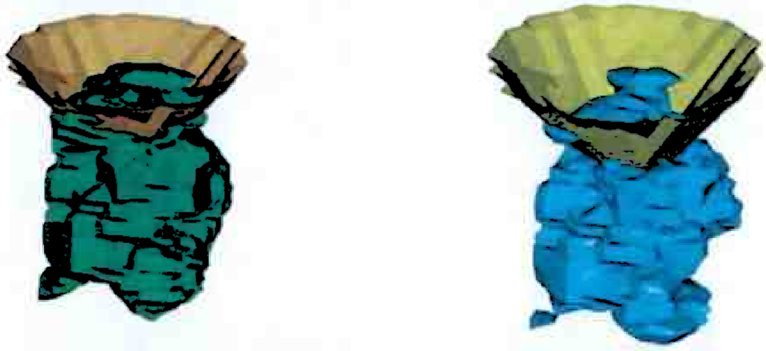


Figure 4 – $US 25.7/t NSR surface with the 325 m pit for the two different prices ($US 24 and $US 16)

Considering the lower cut-off, it was possible to analyse the footprint of the orebody, that varies from a 550m*450m rectangle (figure 6 – 125/-150 masl), to a 550m*400M rectangle (figure 7 – 300 masl) reducing to a much more irregular shape at –550/-575 masl (figure 8).

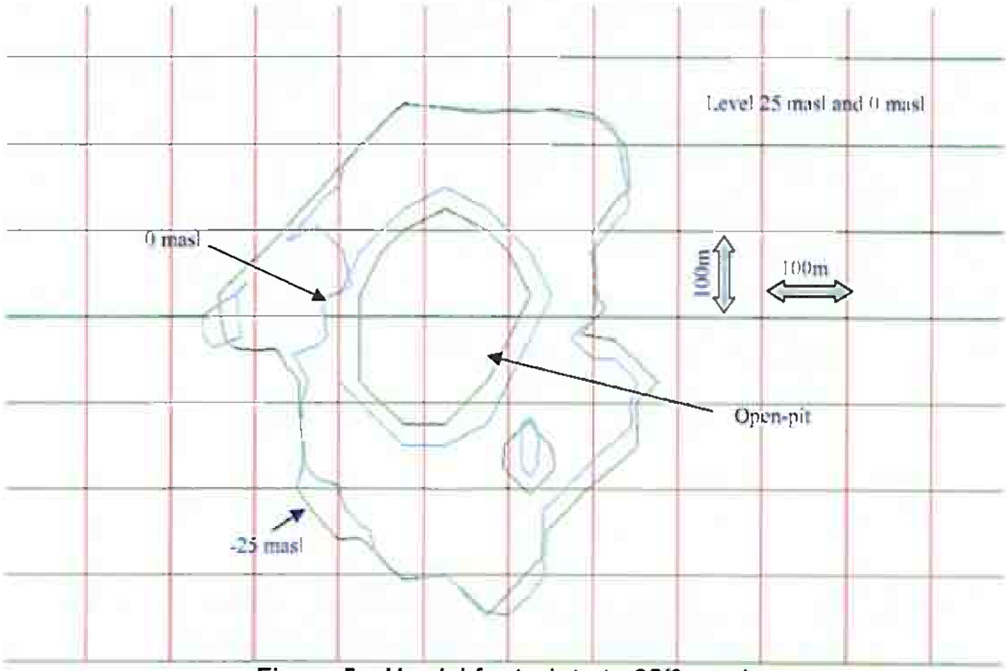


Figure 5 – Hurdal footprint at +25/0 masl

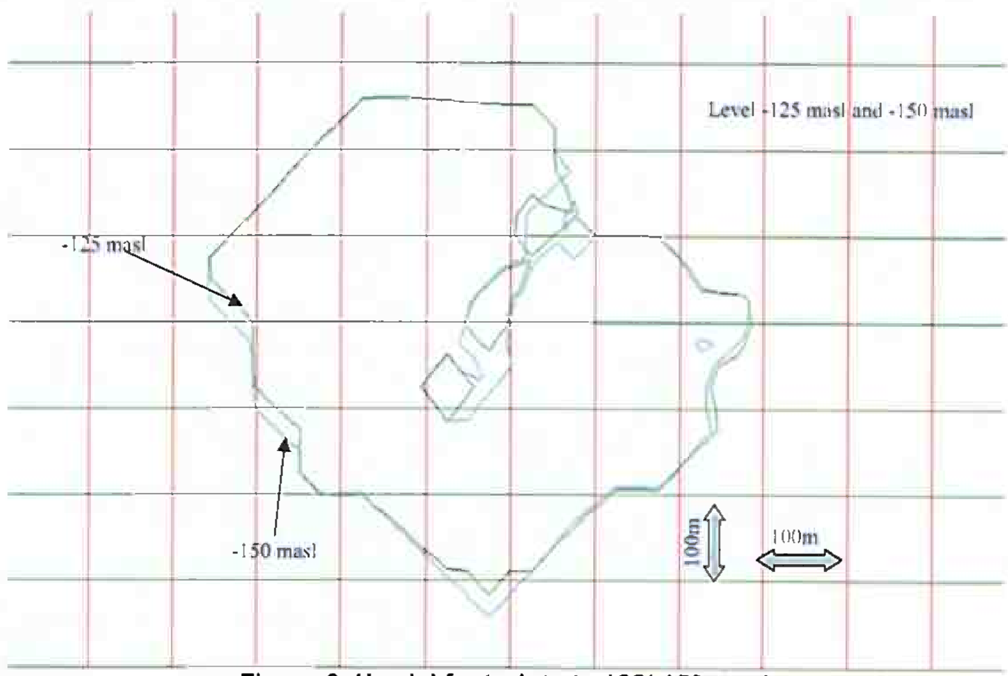


Figure 6- Hurdal footprint at –125/-150 masl

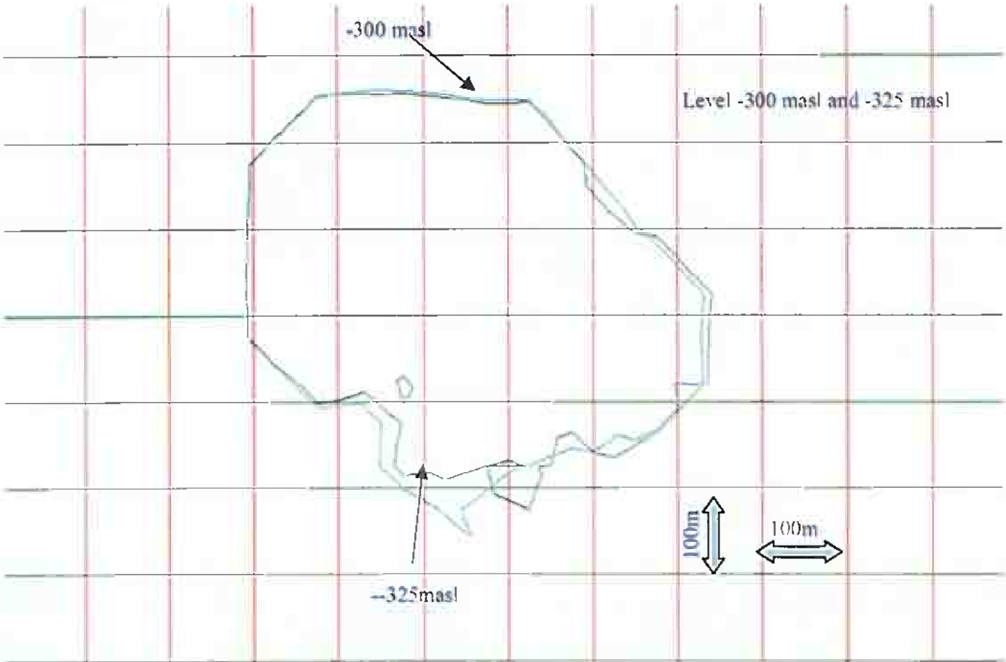


Figure 7- Hurdal footprint at –300/-325 masl

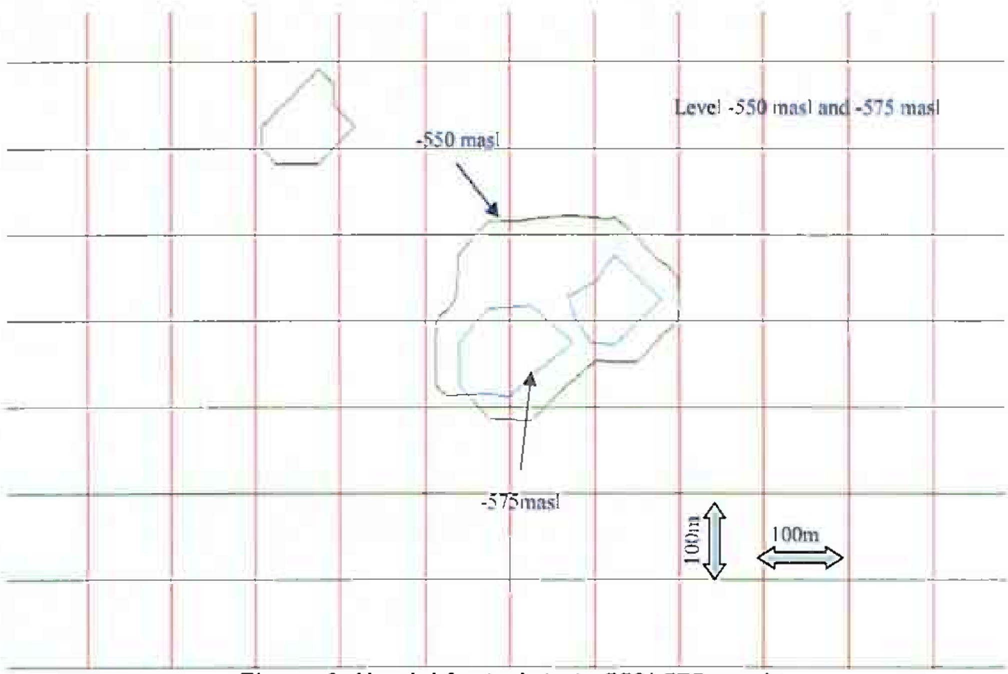


Figure 8- Hurdal footprint at –550/-575 masl

In order to increase the production rate to a minimum acceptable, Crew Gold needs to mine a lot of stopes at any one time. Thus it is important that the mine design increases the number of available stopes that can be mined simultaneously. This can be done with shorter stopes and several mining levels, with the corresponding access pillars and sill-pillars, resulting in recovery losses.

2.2 Stope dimensions

Based on the assumption that geotechnical conditions are good, the following stope dimensions were defined:

- Width – 12 meters;
- High – 25 meters;
- Length – normally 100 meters, but can go from 30 to 120 meters, at the extremity of the orebody;

2.3 SLOS schematic mining philosophy

This chapter will focus on the detailed description of the mining method proposed, together with all the sequencing aspects that are extremely important for the success of this method. Mining will be done on a cycle with 7 phases:

Phase 1 – Development of the primary stopes (Level 1)

Drilling and mucking drifts will be developed on the first mining level on the primary stopes (figure 9). Support cables will be installed on the upper drilling drift.

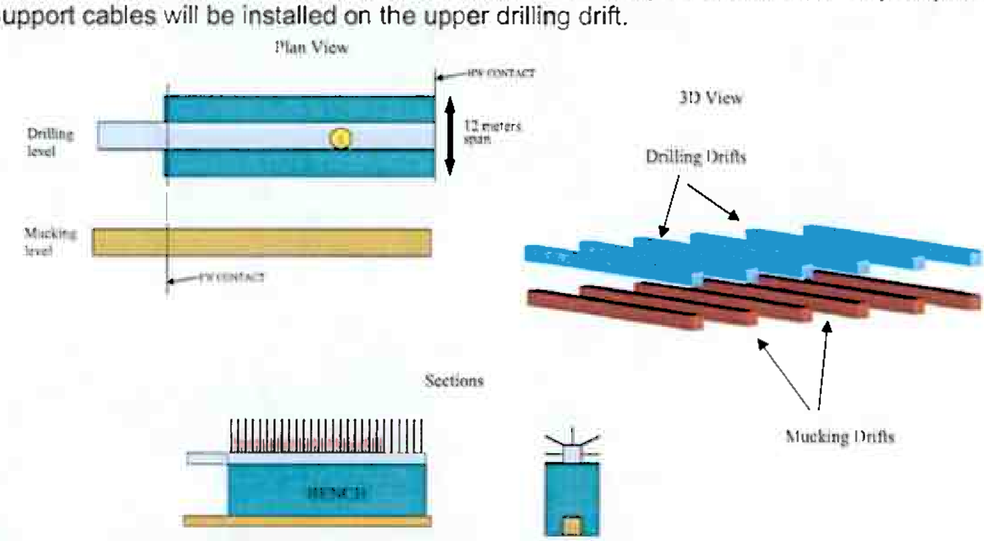


Figure 9 – Primary stopes drilling and mucking drifts development

Phase 2 – Primary stopes slot (Level 1)

Drilling and mucking drift will be linked by a slot (0.8 m diameter), that will be enlarged to a 12m * 5m (figure 10).

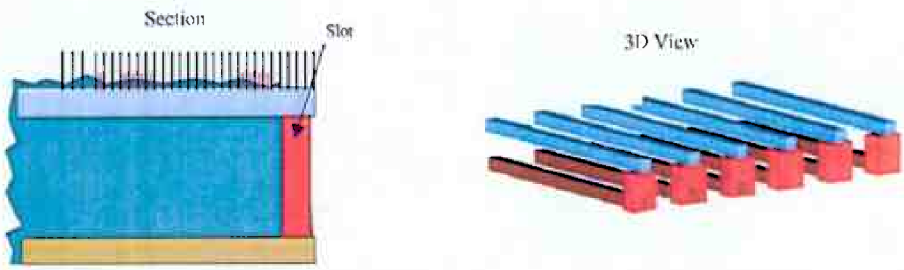


Figure 10 – Slot and slot enlargement

Phase 3 – Primary stopes bench blasting (Level 1)

After the slot is enlarged, ring drilling is done from the upper drift, and the stope is benched back. Mucking is done using LHDs with remote mucking (figure 11).

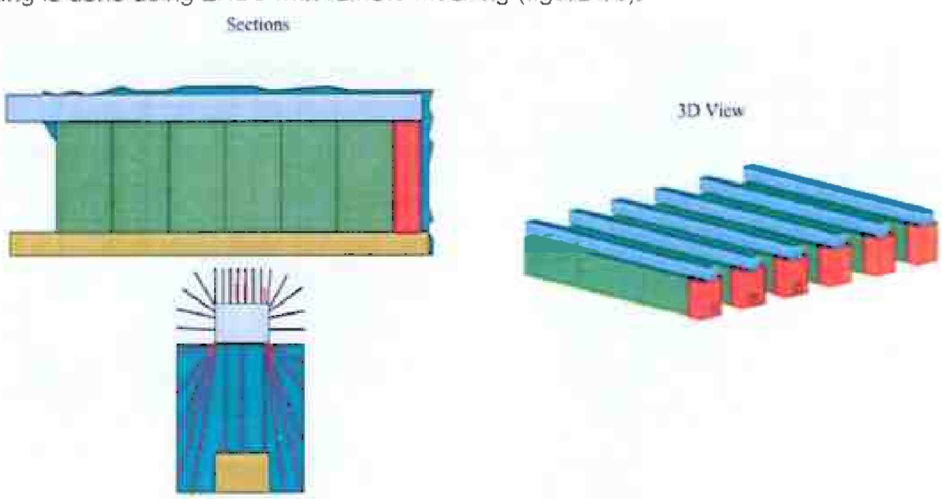


Figure 11 – Bench drilling, blasting and mucking

Drilling, hole charging, and bench blast design accuracy is crucial in order to have good sidewall stability. A detailed drilling and charging pattern should be done for each hole, as well as a detailed blasting sequence is vital for the success implementation of this mining method. Bad practices may lead to ore losses, unsafe conditions and dilution.

Phase 4 – Primary Mining (Level 2)

After the primary stopes of the first level are finished, a detailed cavity survey is needed in order to do an accurate drilling pattern for the secondary stopes. This survey is done with a CMS (Cavity Monitoring System) that will enable a quick and accurate survey of each stope in order to produce a 3D model of the cavity. With this model sections can be made as well as a proper drilling layout for the secondary stopes.

After the primary stopes are surveyed, they are backfilled with paste with 5% cement. The upper drilling drift will be left open, in order to be used as mucking drift for the next level (level 2). This will reduce the mining costs, avoiding double development.

After the backfill of the primary stopes on the first level, mining of the primary stopes of the second level will be done, following the same sequence (figure 12).

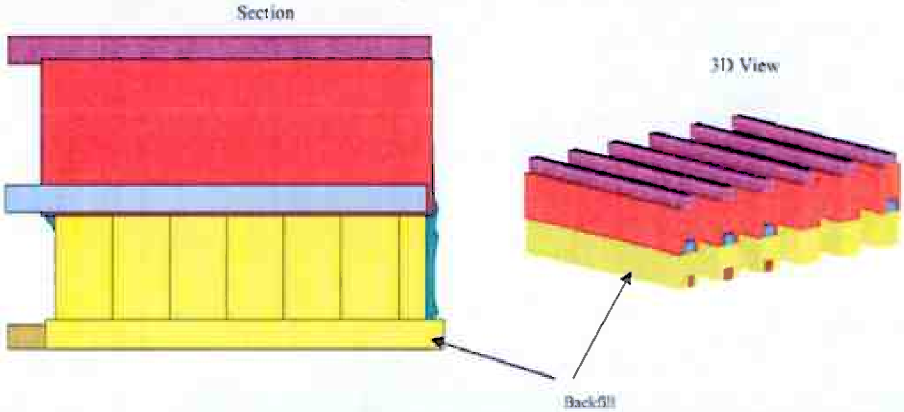


Figure 12 – Level 2 primary stopes mining

Phase 5 – Level 1 secondary stopes bench mining, and level 3 drilling level development

Backfill with 5% cement of the primary stopes of the second level is done, followed by the development of the drilling and mucking drift of the secondary stopes (level 1).

Mining of the first level secondary stopes is done exposing the 5% cement paste-fill on the sidewalls (figure 13). Drilling drifts for the primary stopes of level 3 are developed.

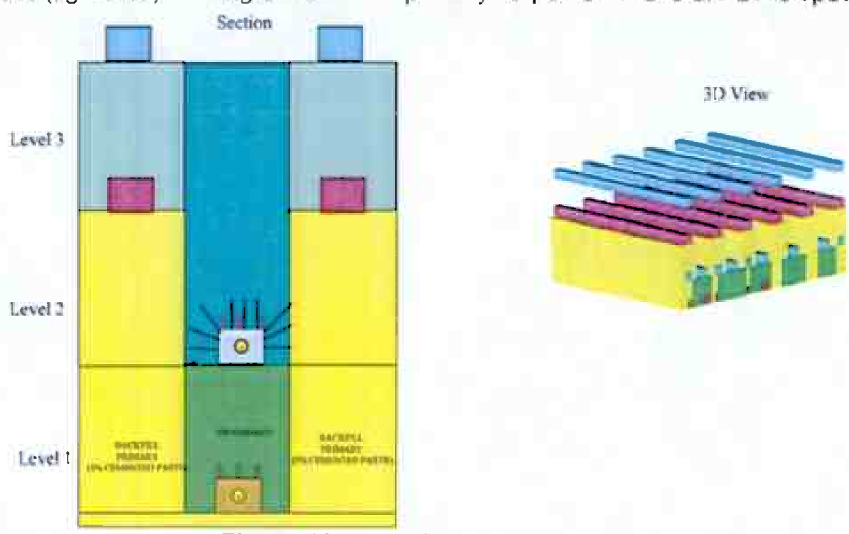


Figure 13 – Level 1 secondary stopes mining

Phase 6 – Level 1 secondary stopes backfill with 1% cement, and primary stopes mining (level 3)

Backfill of the secondary stopes for level 1 is done with 1% cement to avoid liquefaction problems.

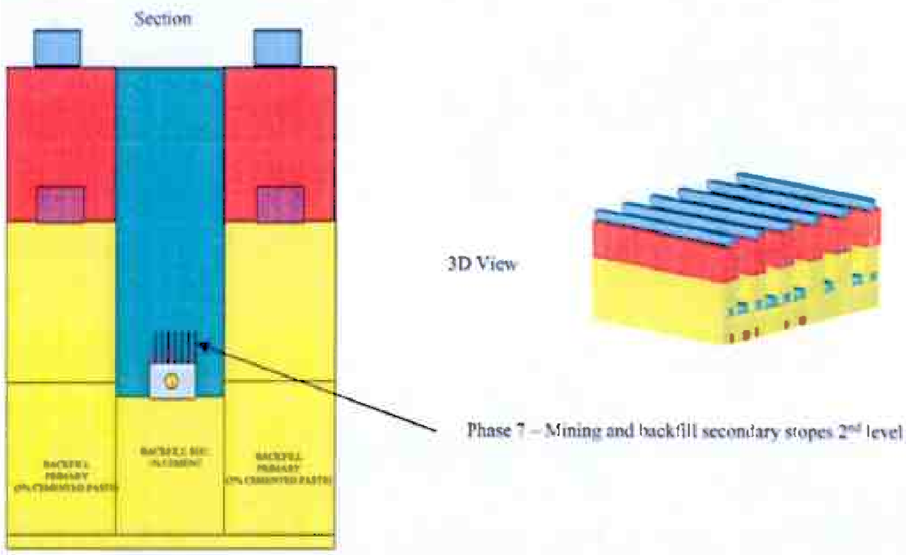


Figure 14 – Level 1 secondary stopes backfill, followed by the primary stope mining on level 3

This sequence of operation will avoid the enlargement of the upper drilling drift, and avoid the development for each stope of a mucking and a drilling drift, reducing the mining costs.

This method will oblige the use of ring drilling with the associated disadvantages of this drilling method (sidewall stability problems due to imprecise drilling).

After the backfilling of level 1 secondary stopes with 1% cement, mining of level 3 primary stopes will start. It must be pointed out that this method will lower the costs, due to the decrease development and the fact that tight fill is not needed (no hydraulic fill to do the top up of the stopes), but could cause some instability problems on the secondary stopes. As it is possible to see on figure 13 and 14, secondary stopes with only the shoulders on the backfill stopping them from being "stalactites" between the primary stopes.

After the mining and backfilling of the primary stopes in level 3 (5% cement), mining of the secondary stopes of level 2 will follow – **phase 7**.

The sequence will continue up, following the same mining and backfilling cycle.

This mining approach has not been analysed by a geotechnical specialist. Pillar dimensions should be modelled and checked, in order not to carry to much stress that could cause bursting, and unsafe conditions.

Also for this method to work it is needed good ground conditions on the ore. In case of unstable secondary stopes, the following alterations to the mining method are needed:

- Enlargement of the drilling drift, in order to do parallel drilling for the bench mining;
- Top up of the paste-fill with hydraulic backfill in order to achieve tight fill;
- Development for each stope of a drilling and a mucking drift;
- Level by level mining and backfill approach;

This will reduce the geotechnical problems, but will raise the operational costs and can reduce productivity. It is however, the tried and tested bench method used successfully at Neves Corvo and elsewhere.

2.4 Overall SLOS layout design options

For the overall stope layout design 3 options were analysed:

a) **Option A** - Primary and Secondary mining with re-slotting of stopes bigger than 100 meters (figure 15).

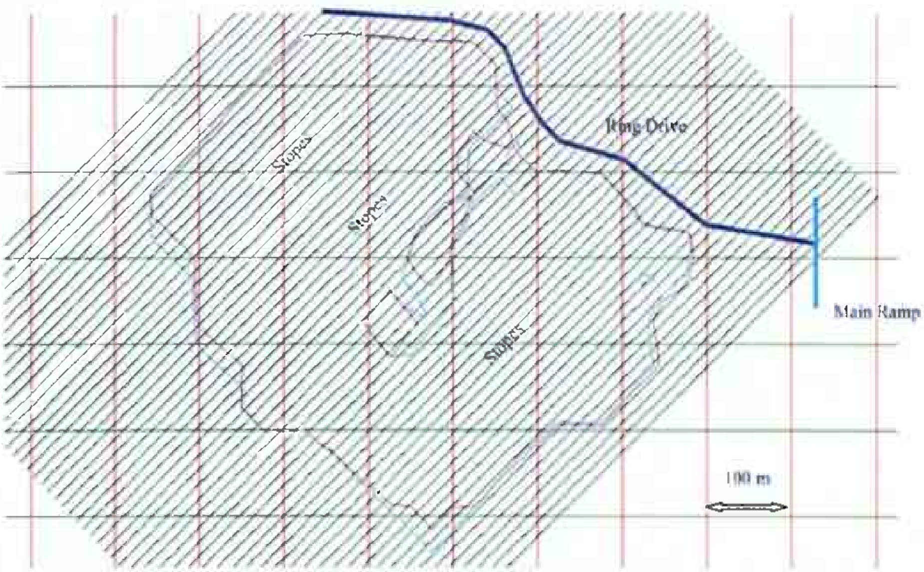


Figure 15 – Primary and secondary stopes with re-slotting

This option will maximise ore recovery, but productivity will be low on each level due to very long stopes that will require re-slotting every 100m or so. This will require pillars at the slotting points in the secondary stopes, or an increase of cement content to 5% in all stopes, with high operational mining costs.

Low productivity will be aggravated by the fact that it is not possible to mine two adjacent primary stopes due to stability reasons, reducing even more the number of available productive faces.

b) **Option B** - Define mining blocks with pillars for the accesses and pillars between the stopes in order to maximise the number of faces and reduce the cement cost. Backfill is done with 1% cement in all stopes (figure 16).

This option will improve productivity, because of the number of available faces per mining level, reduces the operating mining cost, but orebody recovery will be reduced due to the access pillars, and pillars between the stopes.

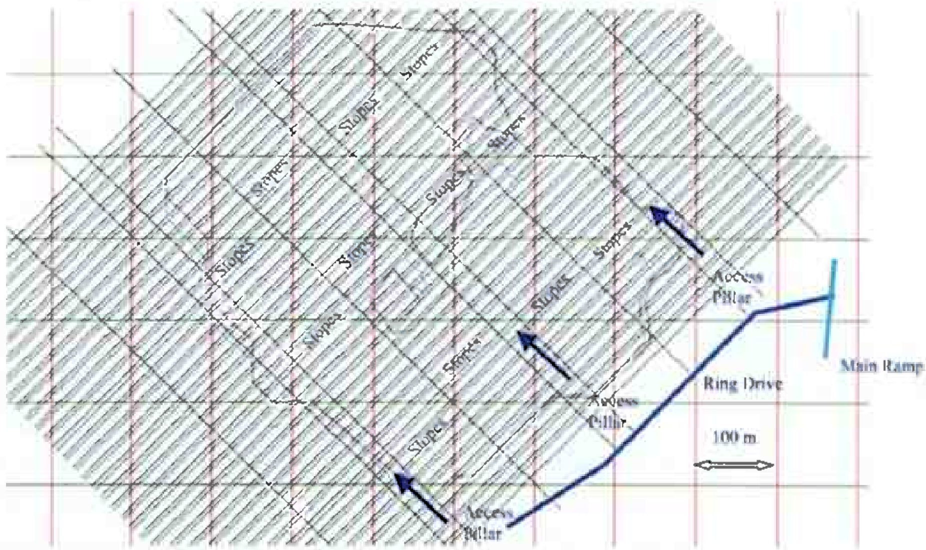


Figure 16 – Primary mining with pillars between mining blocks and the stopes

c) **Option C** - Primary and Secondary mining with paste-fill with 5% cement on the primary stopes and 1% on the secondary stopes, but leaving pillars for the accesses in order to maximise the number of faces (figure 17).

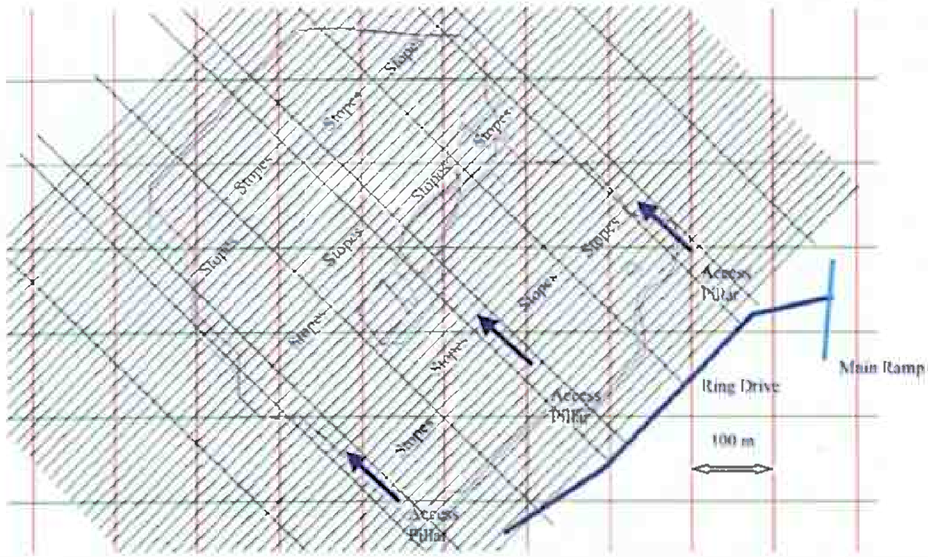


Figure 17 – Primary and secondary mining with unrecoverable pillars for the accesses

Comparing each option it is possible to define the advantages and disadvantages of each design option (figure 18).

Opions	Adavantages	Disadvantages
Option A vs Option B	No pillars	• Low Productivity – longer stopes less face availability; • Re-slotting in primary and secondary stopes due to stope length; • Unrecoverable pillars on secondary stopes due to re-slotting, or backfill secondary stopes with 5% cement; • High cost due to minimum of 3% cement on backfill; • Increase dilution due to sidewall backfill exposure; • Less favourable stope stability when mining the secondary stopes;
Option B vs Option A	• Improved productivity – more stopes; • Paste-fill with 1% cement – less costs; • Better stope back stability;	• Ground control issues due to pillar stability between stopes and high stresses on access pillars;

Opions	Adavantages	Disadvantages
Option C vs Option B	• Improved productivity – more stopes; • Less ore loss on pillars; • No stability problems on the pillars between the stopes;	• Higher cost due to a minimum of 3% cement on backfill; • Increase dilution due to sidewall backfill exposure; • Less favourable stope stability when mining the secondary stopes:

Figure 18 – Advantages and disadvantages of each mining option

Detailed design on this report, will focus on option C, because of the low productivity achieved with option A, and due to the big decrease in ore recovery on option B.

2.5 Stope level design

Sub-level Open stope mining method was used assuming the following mining philosophy:

- Each mining level is divided in mining blocks;
- Each mining block will have a length equal the Northwest/Southeast width of the orebody but with a maximum width of 200 meters (length of two stopes) – figure 19;

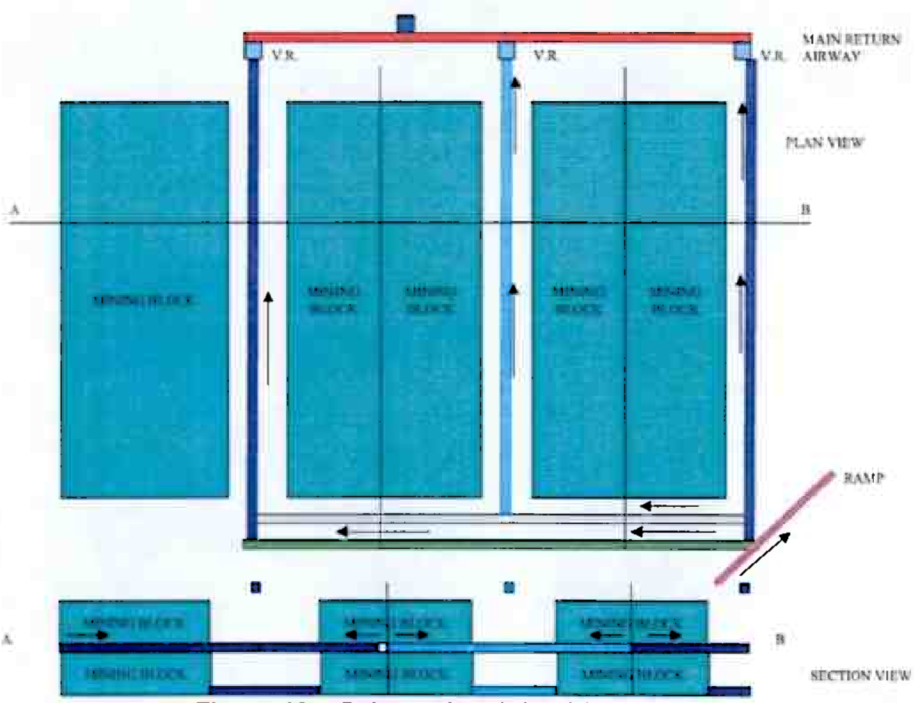


Figure 19 – Schematic mining block design

- 30 meter unrecoverable pillars between each mining block. This pillars will allow the development of the accesses to the stopes and ore passes positioned near the stopes entrance in order to decrease mucking time;
- The pillar between each mining block has the accesses to the stopes. In order not to duplicate development the upper drift (drilling level) of a mining block will be used for mucking drift on the level above of the same mining block (figure 19);
- Each drift (drilling and mucking levels) will be connected to ventilation raises at the extremity of the orebody, that in turn will be connected to a main return airway drift;
- The upper return airway drift will have the main ventilation raises connected to surface. This ventilation raises will have exhaust fans, in order to ventilate the mine;

In order to illustrate the mining design done, the –325/-350 level is presented in figure 20 and 21.

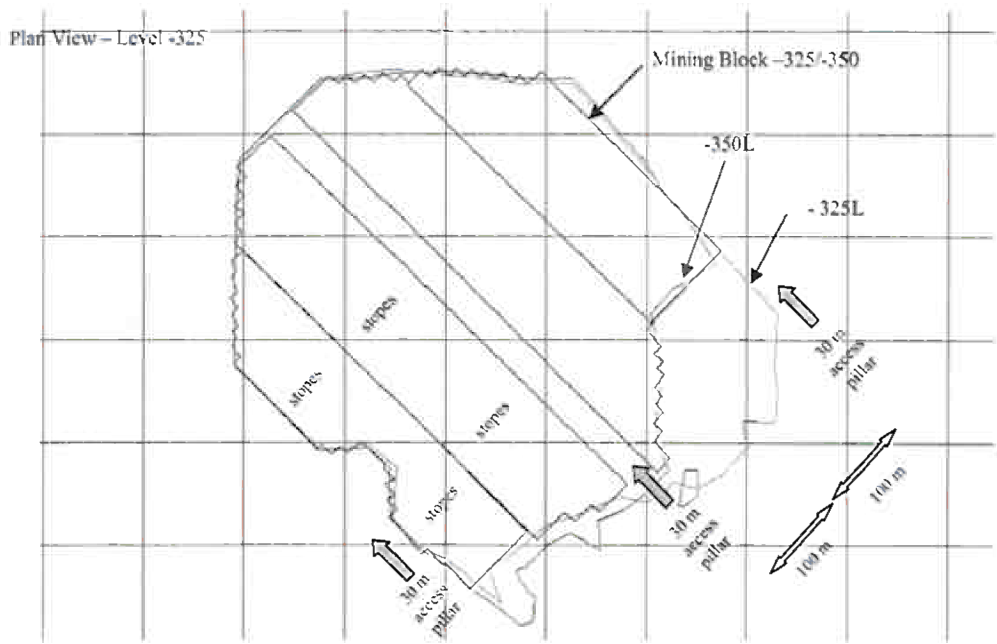


Figure 20 – Stope layout for level -325 (-325 to -350 elevation)

Figure 21 shows a 3D view of the stope design for the same level (ore from -325 masl to -350 masl) as well as all the accesses needed. Ore passes will be developed inside the access pillars in order to reduce mucking distance.

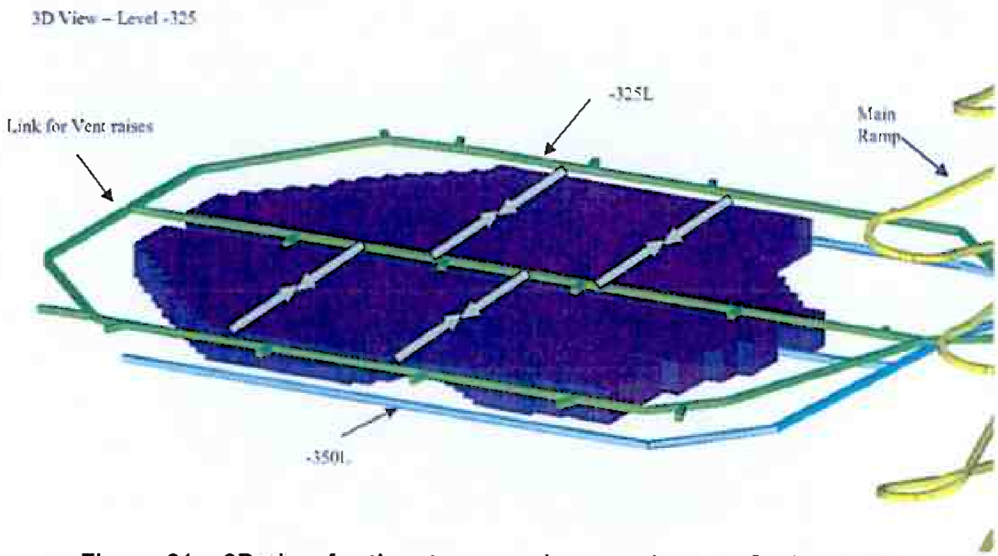


Figure 21 – 3D view for the stopes and access layouts for level -325

All stopes for each level were designed, as well as the access drives for each one of the stopes (figure 22).

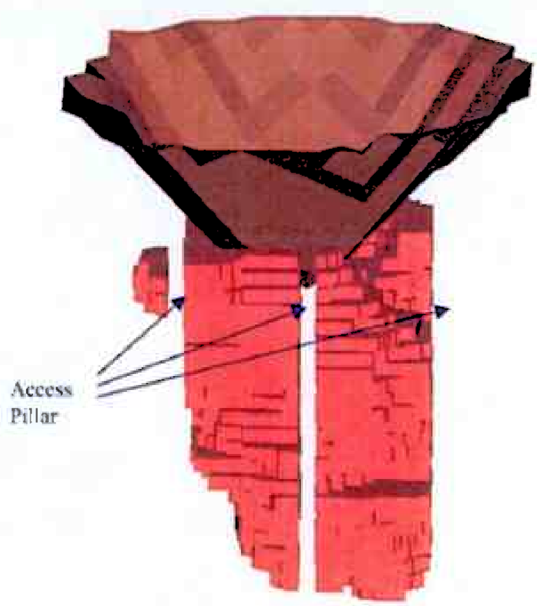


Figure 22 – Open pit with 325 m depth, and underground SLOS stopes

3. Stope productivity, number of mining levels and mining reserve

3.1 Stope productivity

In order to calculate an average level rate, a stope rate was calculated form first principles, based on 5 phases:

- **Stope development** – Drilling drift
- **Support** – Cables drilling and introduction on the drilling drift
- **Slot and slot enlargement**
- **Bench Drilling**
- **Bench charging, blasting and mucking**

This cycle will continue up to the completion of the stope, followed by the backfill of the void.

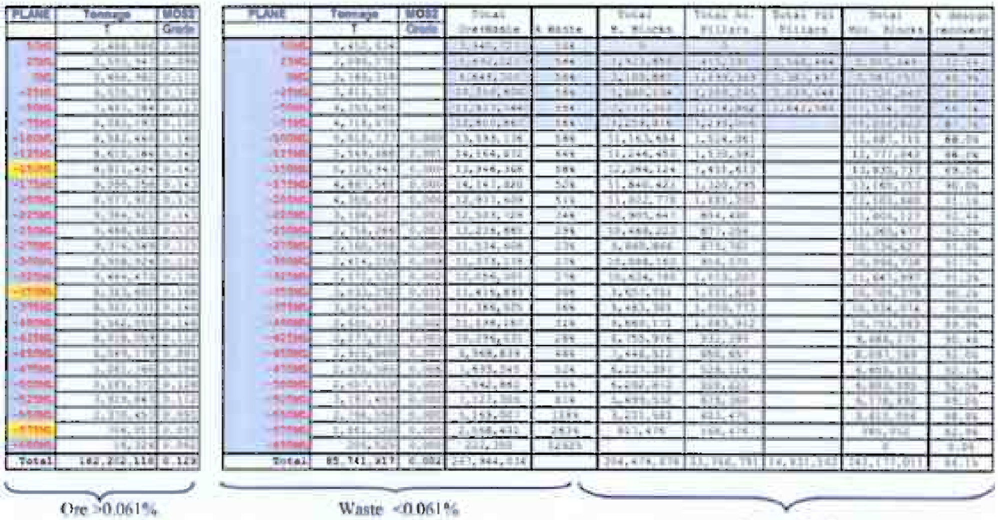
Considering the average tonnage per stope (55,000 tons), and the production target per year of 6,000,000 tons, this will oblige 110 stopes (3.3 working days per stope) to be mined and backfilled per year.

This will mean that Crew Gold needs to backfill simultaneously several stopes, and that an independent paste-fill system for each mining area needs to be built.

According to previous calculations done for similar orebodies with similar mining methods, an average productivity of 300 tons/day per stope can be achieved, assuming an 80% efficiency of the process due to breakdowns, face availability, backfill availability, etc.

3.2 Mining Reserve

Based on the definition of the surface that includes mainly blocks above the cut-off, it was possible to evaluate the tonnage per level of ore (blocks above cut-off) and waste (blocks below cut-off). This mining method (Bench&Fill), obliges that all the material between the drilling drift and the mucking drift should be considered as ore. So it is possible to have some selectivity when developing the drifts, or by defining waste areas inside the stopes that will not be blasted, but this selectivity is limited. As the resource blocks were not available, the consultant did the mining blocks design in Autocad on the areas where there was an overlap between the upper and lower level, defining the access pillars and the pillars to the open-pit, achieving a design recovery for each mining level (figure 23).



PLANE	Tonnage T	MOS2 Grade
Total	182,202,116	0.129

PLANE	Tonnage T	MOS2 Grade	Total Orebodie	% Waste	Total N. Blocks	Total Rib Pillars	Total Sill Pillars	Total Min. Blocks	% design recovery
Total	85,741,917	0.002	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

Figure 23 – Gemcon evaluation and Autocad design (Cut-Off grade = 0.061% MoS2)

With this method a 84.1% design recovery was achieved, and based on the Gemcom evaluation, inside the 0.061% MoS2 shell there is 182Mtons at 0.129% MoS2, and 86Mtons of waste at 0.002% MoS2.

Based on the above figures the design recovery per level was applied to the ore tonnage for each level, and a 5% internal dilution and 7.5% backfill dilution was considered, as well as a 95% recovery on the blasted tonnes, in order to calculate the mining reserve.

Based also on the fact that it is preferable to have simultaneously the same number of faces, avoiding too much stopes waiting for backfill (and a stable mining rate), it was considered that the maximum number of stopes per level that could work simultaneously was 16% of the total. With the above figures, and the total number of stopes per level it was possible to calculate an average rate per level and a total number of 3 levels working in parallel in order to achieve a production rate of 6.0 Mt/year (figure 24). A 50% recovery of the sill pillars was considered.

Mining Recovery	95.0%
Internal Dilution	5.0%
Backfill Dilution	7.5%

Number of working days	365
% of working stopes	16%
Tons/day	300

50% recovery of the sill pillars – 3 levels

PLANE	% design recovery	Available Resource	External Dilution	% MoS2 Int. Dil.	Backfill Dilution	Total Blasted mat.	Mining Reserve	% MoS2 M. Reserve	Number of stopes	Tons per year	Mining Reserve Final	Mining Res. per area
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]				
25m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	1,255,020	[illegible]
0m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	2,393,121	
-25m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4,071,831	
-50m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5,339,087	
-75m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	7,600,417	
-100m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	8,097,577	
-125m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	8,131,507	
-150m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	
-175m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4,476,505	[illegible]
-200m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	8,376,562	
-225m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	9,221,462	
-250m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	9,381,485	
-275m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	9,232,492	
-300m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	8,811,431	
-325m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	9,285,648	
-350m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	
-375m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4,035,706	[illegible]
-400m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	8,256,215	
-425m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	7,770,496	
-450m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5,965,371	
-475m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4,997,093	
-500m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5,143,320	
-525m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	3,750,698	
-550m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	2,258,173	
-575m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	628,500	
-600m	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]				
Total	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	155,093,634	[illegible]

Figure 24 – Hurdal underground SLOS mining reserve and average rate per level
Cut-Off grade = 0.061% MoS2

On top of this mining reserve, it will be added the development of the pillar accesses

For the $US16 payable cut-off (0.091% MoS2), the same procedure was applied, and the results are summarised on figures 25 and 26.

Ore >0.091%

PLANE	Tonnage T	MOS2 Grade
25m	[illegible]	[illegible]
0m	[illegible]	[illegible]
-25m	[illegible]	[illegible]
-50m	[illegible]	[illegible]
-75m	[illegible]	[illegible]
-100m	[illegible]	[illegible]
-125m	[illegible]	[illegible]
-150m	[illegible]	[illegible]
-175m	[illegible]	[illegible]
-200m	[illegible]	[illegible]
-225m	[illegible]	[illegible]
-250m	[illegible]	[illegible]
-275m	[illegible]	[illegible]
-300m	[illegible]	[illegible]
-325m	[illegible]	[illegible]
-350m	[illegible]	[illegible]
-375m	[illegible]	[illegible]
-400m	[illegible]	[illegible]
-425m	[illegible]	[illegible]
-450m	[illegible]	[illegible]
-475m	[illegible]	[illegible]
-500m	[illegible]	[illegible]
-525m	[illegible]	[illegible]
-550m	[illegible]	[illegible]
-575m	[illegible]	[illegible]
Total	129,695,135	0.147

Waste <0.091% / Autocad Design

PLANE	Tonnage T	MOS2 Grade	Total Ore+Waste	% Waste	Total M. Blocks	Total Ac. Pillars	Total Fit Pillars	Total Min. Blocks	% design recovery
25m	[illegible]		[illegible]	[illegible]			[illegible]	[illegible]	[illegible]
0m	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
-25m	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
-50m	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
-75m	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
-100m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-125m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-150m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-175m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-200m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-225m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-250m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-275m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-300m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-325m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-350m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-375m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-400m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-425m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-450m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-475m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-500m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-525m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-550m	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]
-575m	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]		[illegible]	[illegible]
Total	53,938,516	0.012	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	

Autocad Design

Figure 25 - Gemcon evaluation and Autocad design defining the design recovery
(Cut-Off grade = 0.091% MoS2)

Mining Recovery	95.0%
Internal Dilution	5.0%
Backfill Dilution	7.5%

Number of working days	365
% of working stopes	16%
Tons/day	300

50% recovery of the sill pillars – 3 levels

PLANE	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	Mining Reserve Final	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]		[illegible]	0	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	27	[illegible]	1,421,251	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	3,310,307	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4,409,952	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4,515,024	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5,420,947	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5,879,914	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	6,304,369	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	6,783,344	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	2,705,939	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	7,074,881	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	6,892,108	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	7,049,163	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	6,505,961	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	6,921,325	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	3,036,951	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	5,945,811	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	4,793,896	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	2,061,155	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	2,715,480	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	3,702,008	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	828,357	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	0	[illegible]		[illegible]	0	
Total	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	105,761,667	[illegible]

Figure 26 – Hurdal underground SLOS mining reserve and average rate per level
Cut-Off grade = 0.091% MoS2

The same procedure was done for the other two options (open-pit depth of 275 m), and the mining reserve of the stopes (excluding the ore that can be recovered on the access pillars and accesses to the stopes), are summarised on the figure 27.

	Option 1 $US 24 - 325m	Option 2 $US 24 - 275m	Option 3 $US 16 - 325m	Option 4 $US 16 - 275m
Mining Reserve (Mtons)	155.1	164.2	108.4	113.8
%MoS2	0.122	0.121	0.138	0.137

Figure 27 – Stopes Mining Reserve for all 4 options

4. Main infrastructure

4.1 Main infrastructure design

The basic SLOS mine design was based in 3 levels that will be mined simultaneously, accessed by a main ramp, positioned close to the orebody. The base level for each mining horizon will be linked to an exhaust vent raise, creating a main return airway. Intermediate ventilation raises will link each level with the closest main return airway drift. A main intake ventilation raise, close to the main ramp will be one of the three main air intakes (shaft, main ramp, and intake ventilation raise).

The ore will be mucked from the stopes by LHD and dropped down to a haulage level (-580), where loaders will charge 50 tons trucks, that will transport the ore to the main crusher. After crushing, the ore will be transferred to silos (with a conveyor) where a skip loading facility will allow the transport of the ore to surface by a shaft (figure 28).

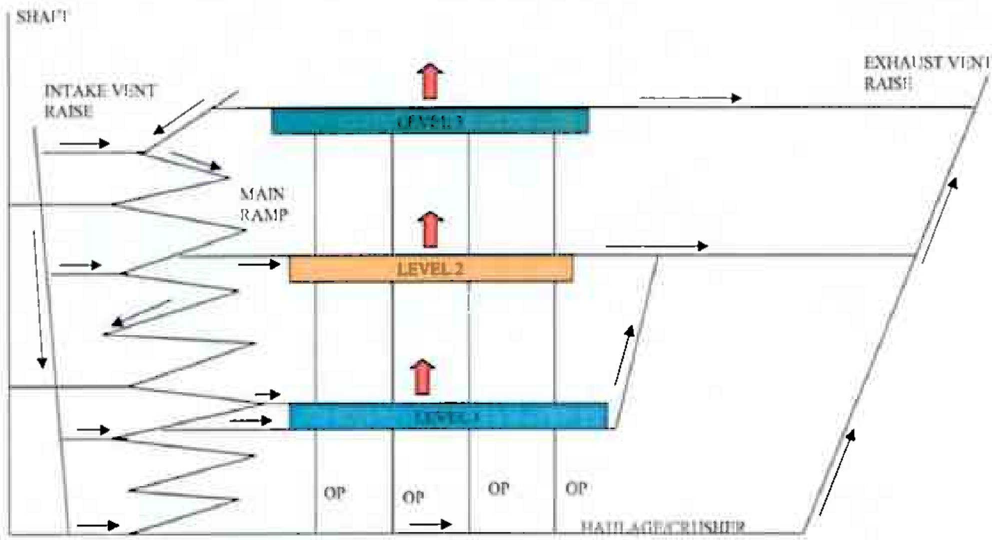


Figure 28– SLOS schematic infrastructure

As main subsidence problems will be avoided with the SLOS mining method thus reducing the catchment from storm events, the amount of water storage capacity was reduced, and a classical sump / thickener / clear water bays / pump station arrangement was design (figure 29). Near the thickener an intermediate pump arrangement will pump the water up to the main pump station near the distribution level (shaft).

The main crusher will receive the ore from the trucks, and with the help of a conveyor will transport the ore up to the ore storage silos near the shaft. Skip loading level, distribution drift and main pump station will be accessed by a ramp driven from the shaft bottom drainage drift.

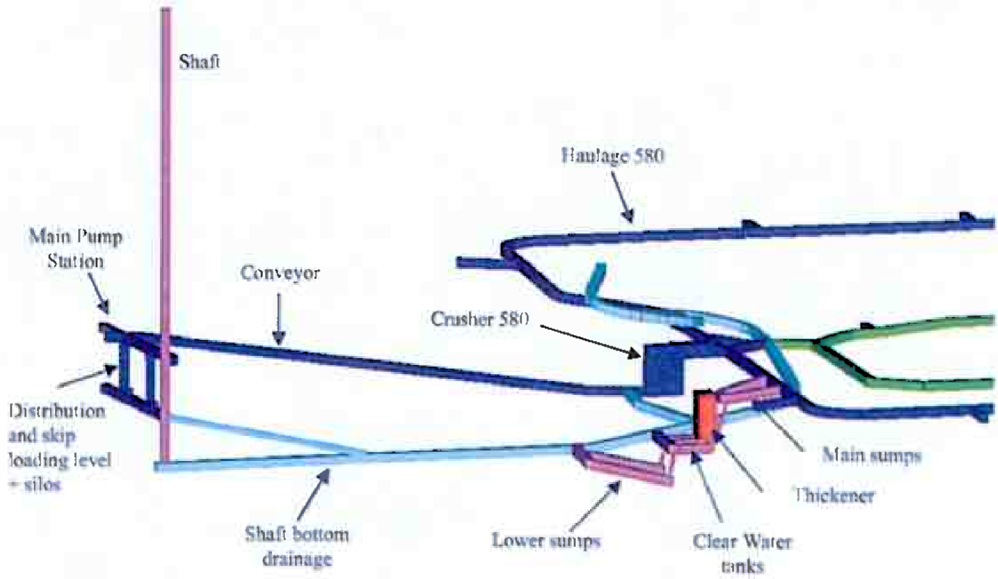


Figure 29 – Crusher and water management arrangement

Figure 30 shows the entire mine infrastructure needed for the Sub-level Open Stopes design, with the 3 production areas. For the second cut-off the basic mine design was maintained, and some adaptations and assumptions were made.

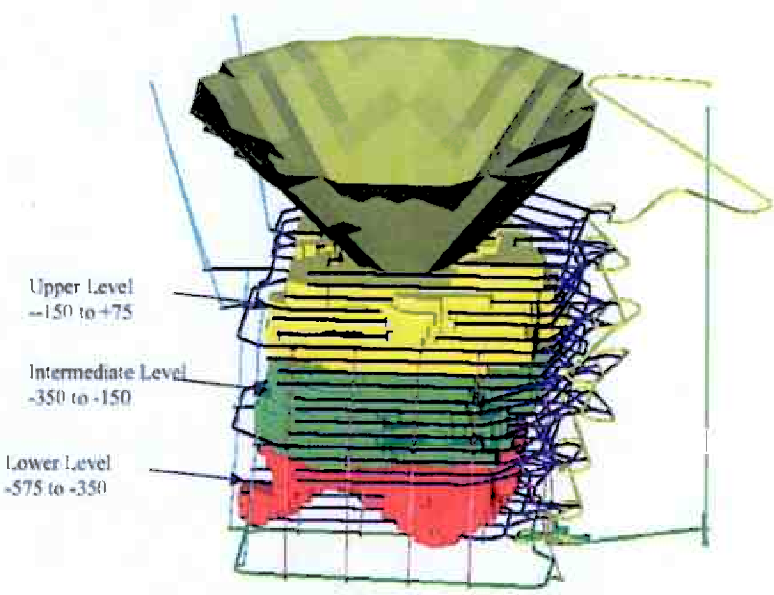


Figure 30 – Hurdal infrastructure and production levels COG 0.061% MoS2

4.2 Capital development schedule

Capital development, must start 6 years before production. The initial development will be focus on the main ramp that will allow the access of equipment and labour to the ore, and the shaft, that will be done by conventional sinking in order to reduce the amount of time needed to complete all the development. From the main ramp connections will be done to the shaft in order to ventilate the main ramp. A 10% contingency factor was introduced on all the developments (with the exception of the shaft) in order that mucking cross cuts, safety bays, workshops, or any additional main intake ventilation raise is included.

The shaft and the portal of the ramp are positioned at surface to the East side of the pit, on the initial slopes of the north side of the valley where the orebody is positioned, in order to avoid drainage problems, and decrease the amount of altered rock.

Rates of development used were 1,200 m/year on the main ramp, and 1 m/day on conventional shaft sinking. During year -1 small scale production will start form the development of the accesses to the stopes, and main mechanical set-up will be installed. Production will start in year 1, with 3 mining levels, as shown in figure 31 (Levels -550/-575, -225/-250, and -125/-150).

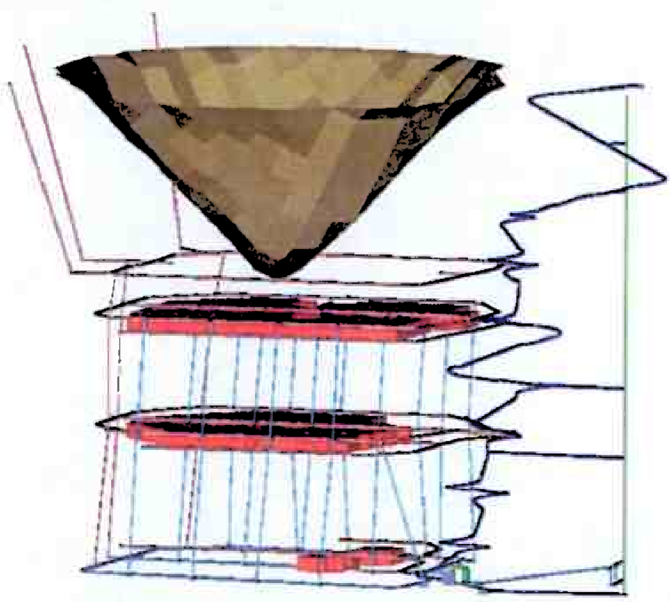


Figure 31 – Capital development, and mining levels for year 1

4.3 Underground capital development expenditure

Development prices used to calculate the Capex for the development are summarised on figure 32. Development prices achieved on other European mines were used and a $US/€ ratio of 1.3 was used to convert those figures.

Capital Development	2.600	$US/m
Shaft	22.750	$US/m
Enlargement	156	$US/m3
Raise 4.5 m	2.275	$US/m
Raise 3.0m	1.625	$US/m
Raise 2.0m	975	$US/m

Figure 32 – Development costs used

Figure 33 summarises the development schedule, where the critical path is coloured in red. Total capital expenditure for the underground mine development is expected to reach 94 M$US, with picks in year –3 and –2. Installation of all the main fixed equipment will be done on year -2 and –1.

	-6	-5	-4	-3	-2	-1
Ramp						
Xcut to shaft 175						
Xcut to shaft 90						
Level -125						
Level-150						
Xcut to shaft -255						
Level-325						
Level-350						
Xcut to shaft -380						
Level -550						
Level -575						
HAULAGE -580						
CRUSHER -580						
SHAFT Bottom arrang.						
Water arrang.						
Ore passes						
Vent Raises						
SHAFT						
TOTAL CAPEX - 93,780,739	11,522,550	11,304,150	11,098,750	16,684,200	26,997,789	16,173,300

Figure 33 – Expected capital expenditure

For the 0.091%MoS2 cut-off, the amount of development was corrected, as the bottom of the orebody is 25 meters higher. So this will reduce the length of the main ramp and the shaft, with a total Capex expenditure for this option of 91 M$US.

The development schedule for all the options will be the same (with the 3m$US Capex reduction for the higher cut-off), as the Capital Development differences are small.

5. Production schedule

SLOS mining schedule was done in order to reflect the average mining rate per mining area, and it does not take into account the fact that Crew Gold will be mining simultaneously in 2 levels per sill level, due to the displacement between the primary stopes and secondary stopes mining. This schedule is a first estimate, and does not have all the details of the complex mining sequence. It is assumed, that for the accuracy of the actual resource this approach is a reasonable assumption, and it will not affect significantly the final figures, giving a reasonable approach to the mix of ore between the several operating mining levels.

The production sequence was done in order to mine at a rate close to 6.0 Mtons/year, and all the accesses were sequenced in order that they will be finalised on a just in time philosophy.

Figure 34 shows the mining schedule and figure 35 the total tonnes produced and average grade proposed for the SLOS stopes of Hurdal, achieving a production rate of 6.0 Mt/year, with 3 mining areas (Option 1 – 325m pit with $US 24 payable).

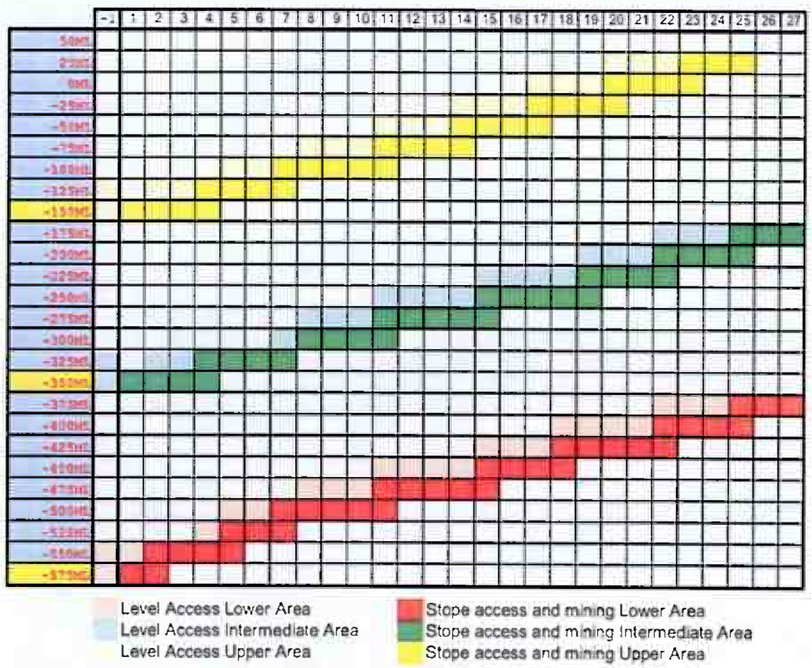


Figure 34 – Mining schedule
Option 1 (325 m pit - $US 24 payable)

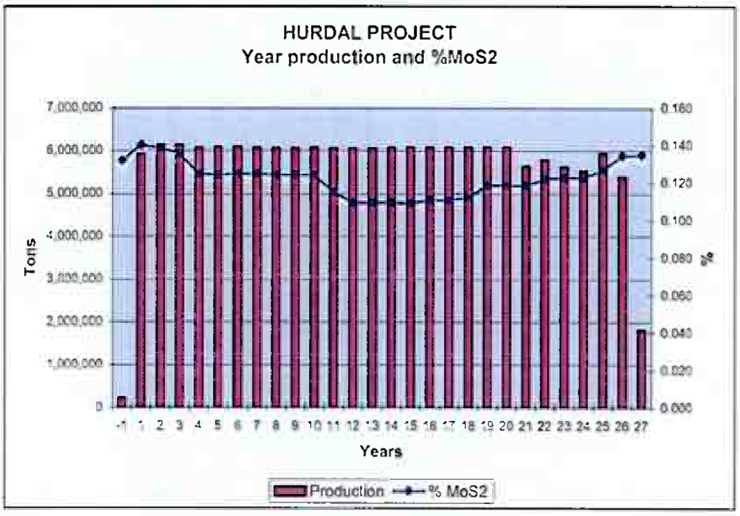


Figure 35 –Production plan
Option 1 (325 m pit – COG 0.061%)

The schedule presented above, includes all the development done inside the stopes, namely the drilling level.

On figures 36 to 38 the production schedule for Option 2, 3 and 4 is summarised.

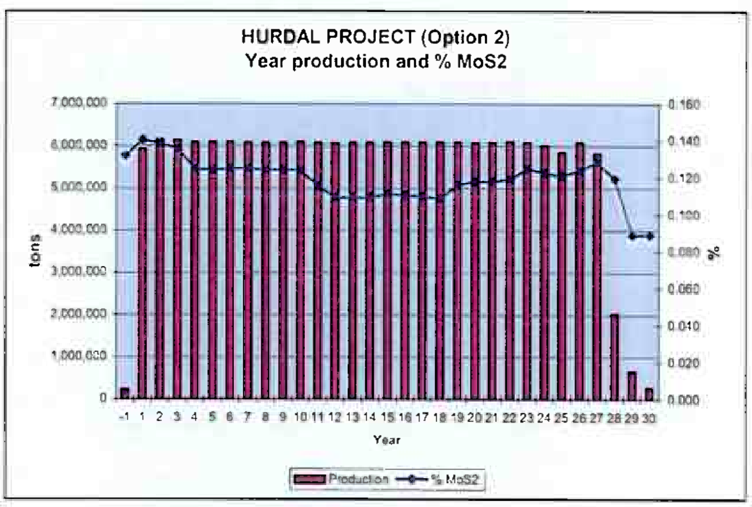


Figure 36 – Production plan
Option 2 (275 m pit – COG 0.061%)

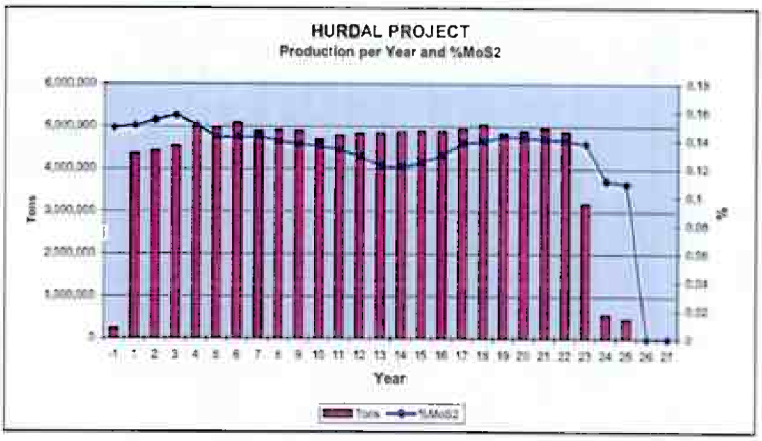


Figure 37 – Production plan
Option 3 (325 m pit – COG 0.091%)

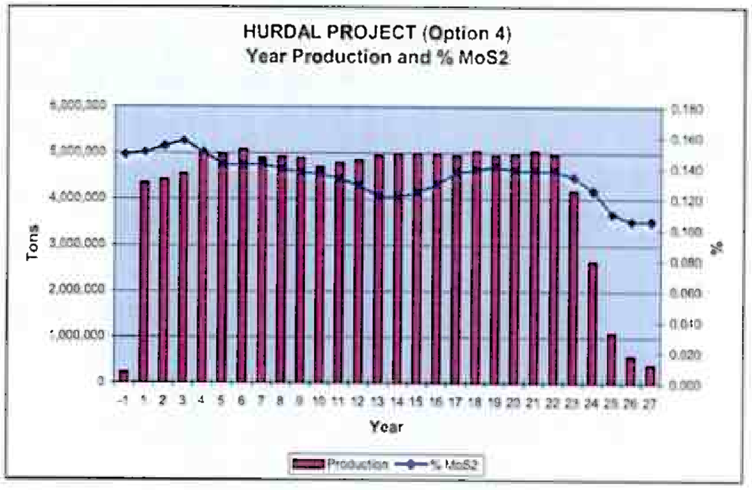


Figure 38 – Production plan
Option 4 (275 m pit – COG 0.091%)

It is possible to conclude that the increase in cut-off reduces the production from 6 to 5 Mtons/year due to the reduction of the number of stopes per level, but with a higher feed grade.

5.46 Operational development schedule

Flat development includes all the development of the footwall drifts and main access drives per level outside the cut-off grade shell (figure 39).

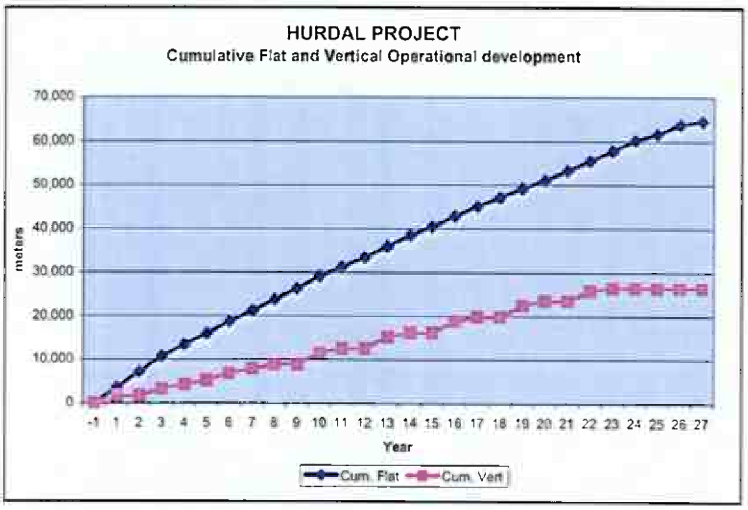


Figure 39 – Flat development schedule (Option 1- 325m pit, COG 0.061%)

Based on those drifts, stopes can be accessed by the mining equipment in order to achieve production. Should be pointed out that this schedule was done on a just in time scenario in order to maximise the NPV, delaying expenditure to the maximum. During the operational phase a more realistic approach should be considered in order to maximise equipment and labour utilisation.

Each level will have some development in order to connect them to the main return airways system, and to connect to the orepass system, that will be located on the access pillars or the extremities of each access drive (considered on the 10% contingency factor).

For the operational vertical development, the following assumptions were made:

- All ore passes and ventilation raises connecting the initial production levels will be substituted during the project life;
- Ore passes and ventilation raises from the Upper level will be extended up, in order to allow the evacuation of the ore and the ventilation of the stopes;
- Contingency factor of 10% for the lower cut-off, and 5% for the higher cut-off;

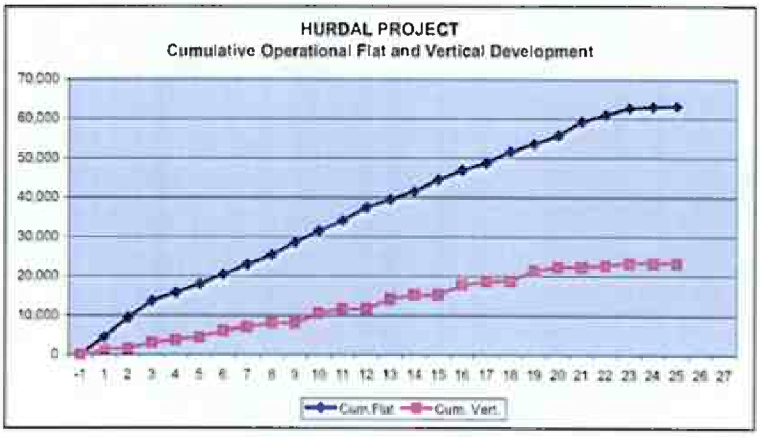


Figure 40 – Flat development schedule (Option 3- 325m pit, COG 0.091%)

For the higher cut-off the following assumptions were made (figure 40):

- Reduce contingency from 10% to 5%, because the amount of development per level will decrease, as the amount of ore per level will reduce;
- Eliminate the levels, that due to the ore reduction will no longer be mineable.

6.7 Operational Costs

Operational costs estimation was done based on actual European mines and projects costs (with similar mining methods), with a 10% increase for contingency factor. Figure 41 summarises the operational cost estimation for Bench&Fill stopes on Hurdal.

SLOS Costs	Value	
Cables	0.89	$US/ton
Drilling	1.36	$US/ton
Charging	0.56	$US/ton
Mucking	0.98	$US/ton
Services	0.25	$US/ton
Haulage	1.76	$US/ton
Hoisting/Crush/Surf	0.69	$US/ton
Paste-Fill	3.72	$US/ton
Supervision	1.92	$US/ton
Main Pumps	0.18	$US/ton
Main Fans	0.19	$US/ton
Electrical distribution	0.06	$US/ton
Other	0.70	$US/ton
Total Mining	13.26	$US/ton
Drilling and Mucking access	2.14	$US/ton
TOTAL SLOS	15.40	$US/ton

Plus Operational Flat and Vertical Oper.Development

Figure 41 – Total mining cost for SLOS

On figure 43 costs, it is still missing the operational flat and vertical development outside the cut-off grade shell.

From the production schedule it is possible to know the tonnage produce per year from stope mining and access development in ore, as well as all the operational development outside the cut-off grade shell. With those values and assuming an average cost for operational development of $US 2,210/m (€1,700/m), year operational costs can be estimated (figure 42 and 43 summarises the operational costs per ton for option 1 and 3 – 325 m pit with 0.061% MoS2 cut-off grade and 325 m pit with 0.091% MoS2 cut-off grade).

All operational costs were considered variable, assumption that can produce discrepancies when the production target is not achieved (end of mine life).

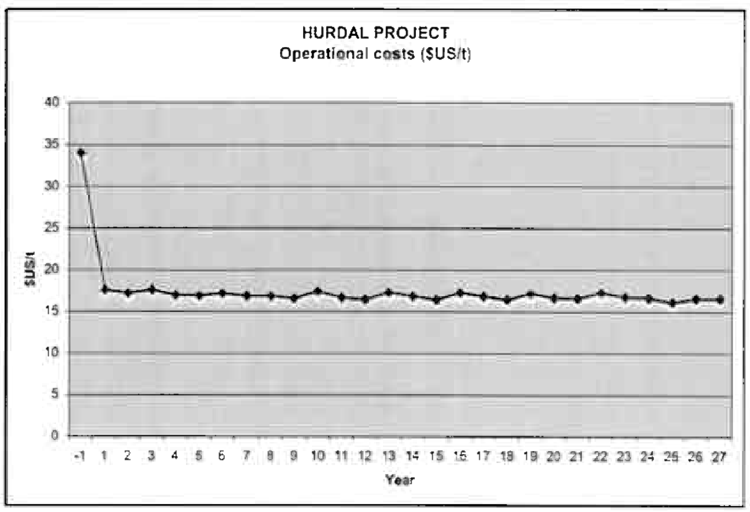


Figure 42 – Operational costs estimation per year (Option 1)

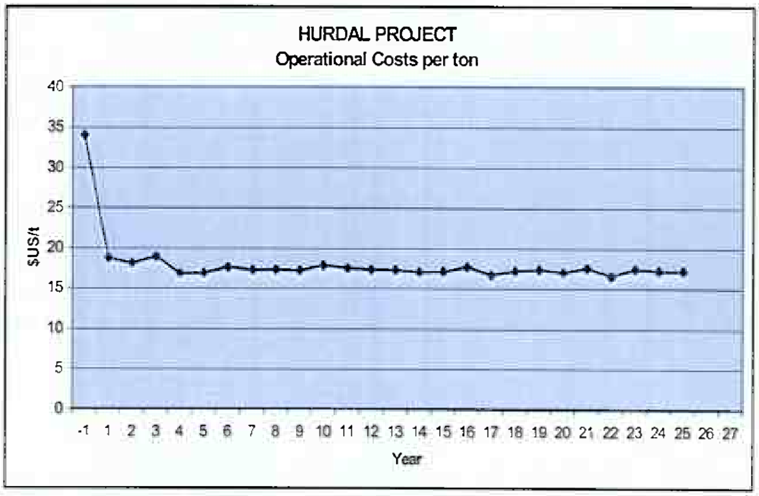


Figure 43– Operational costs estimation per year (Option 3)

7.8 Plant feed and metal production

Combining the Open-pit and Underground schedule it is possible to predict the plant feed grades. Initial plant feed will be 8.0 Mtons/year during the open-pit years, followed by a decrease on production (6,0 Mtons/year – 0.061% MoS2 cut-off, and 5,0Mtons/year – 0.091% MoS2 cut-off) but a increase on the average feed average grade,

Figures 44 and 45 summarises the plant feed estimation for Options 1 and 3.

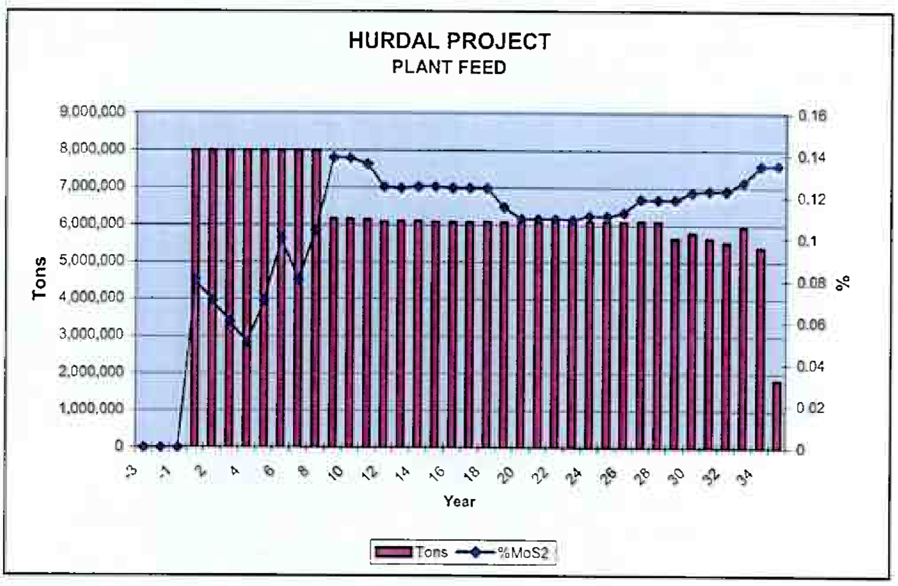


Figure 44 – Hurdal Mine Plant Feed (Option 1- 325m pit, COG 0.061%)

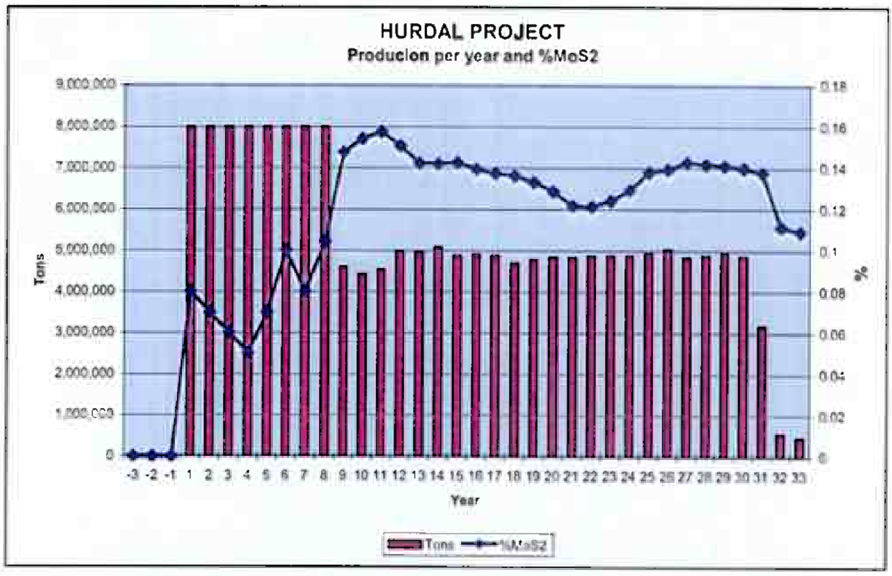


Figure 45 – Hurdal Mine Plant Feed (Option 3- 325m pit, COG 0.091%)

Figure 46 summarises the Mo production during the project life, for all the options, assuming a 85% plant recovery, a 50% Mo concentrate grade, and 59.9% Mo in the Molybdnite.

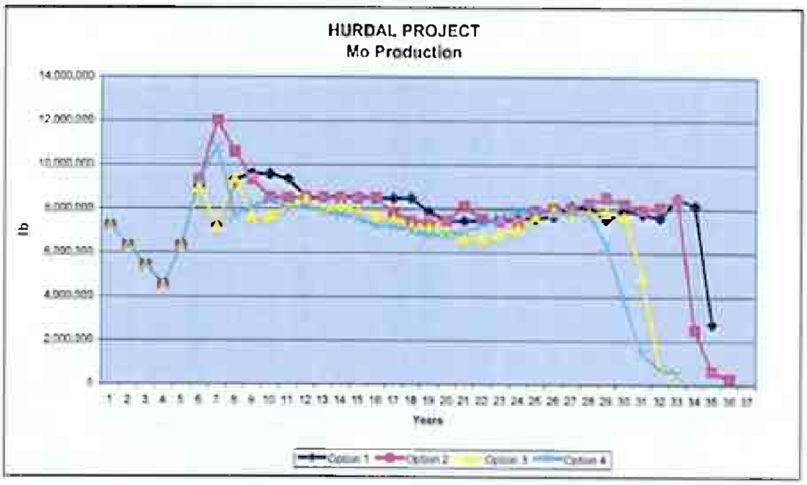
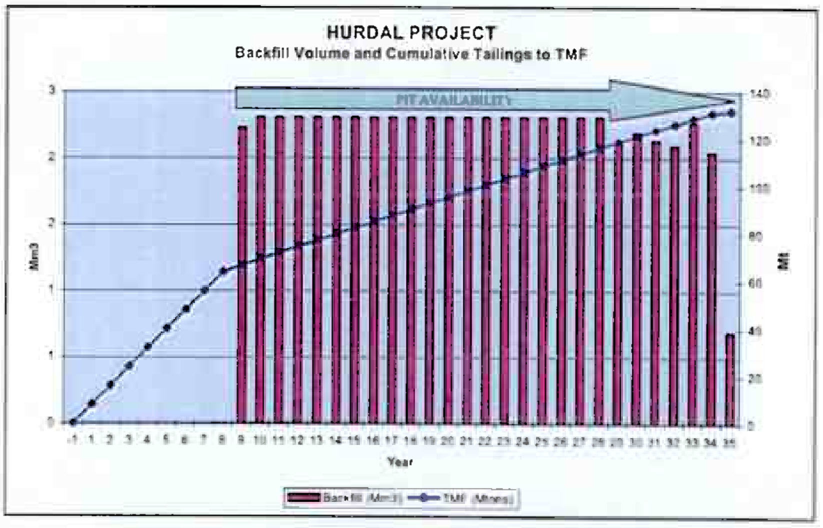


Figure 46 – Mo production (lb) during the project life for all the options

8.9 Tailings disposal

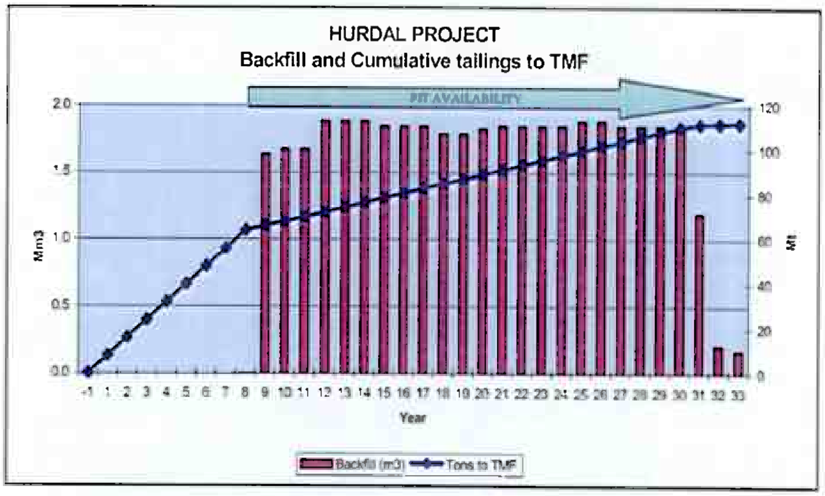
Assuming the same plant parameters (85% plant recovery, a 50% Mo concentrate grade, and 59.9% Mo in the Molybdnite), it is possible to calculate the amount of tailings produced per year. Knowing the void open due to stope mining per year and assuming a paste density of 1.5 tons/m3, it is possible to calculate the amount of tailings for the TMF disposal (figure 47 and 48) . Open-pit capacity is sufficient for the amount of tailings produced during the underground phase of the project.



(assuming 85% recovery, 50% concentrate grade and 59.9% Mo for the MoS2)

Free Capacity at end of Mine Life [illegible] tons of paste

Figure 47 – Backfill volume per year and cumulative tailings disposal (Option 1 – 325m pit 0.061% MoS2 COG)



(assuming 85% recovery, 50% concentrate grade and 59.9% Mo for the MoS2)

Free Capacity at end of Mine Life [illegible] tons of paste

Figure 48 – Backfill volume per year and cumulative tailings disposal (Option 3 – 325m pit 0.091% MoS2 COG)

10.10 Underground Mobile Mining Equipment

Assuming the performances and unit costs, summarised on figure 49, it is possible to calculate the amount of mobile mining equipment, needed per year in order to achieve the production targets for each option.

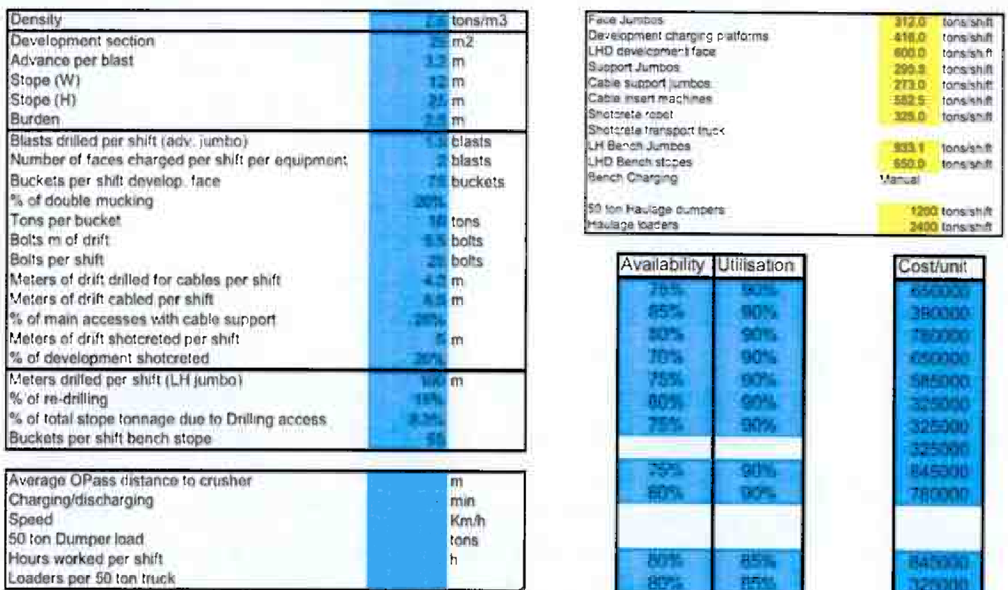


Figure 49 – Performance and unit costs for the underground mobile equipment

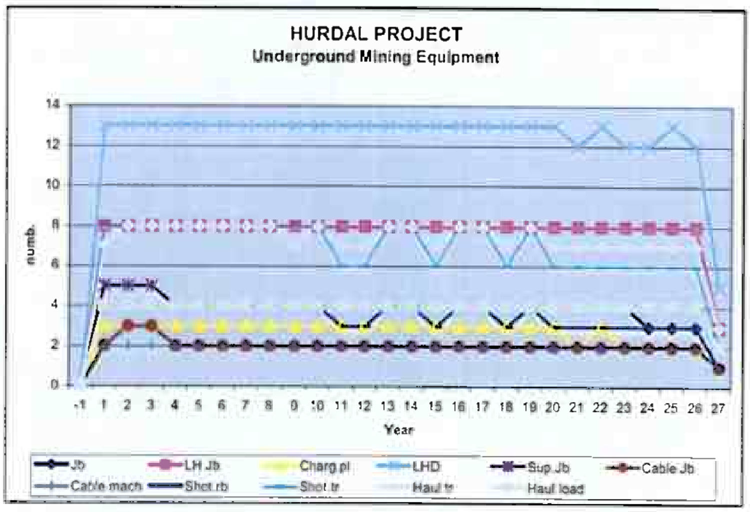


Figure 50 – Underground mobile equipment per year (Option 1 – 325 m pit COG = 0.061%MoS2)

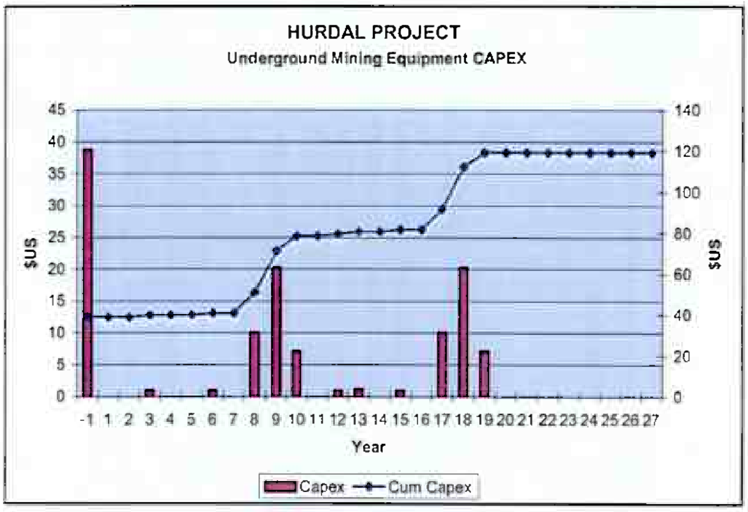


Figure 51 – Underground mobile equipment Capex per year (Option 1 – 325 m pit COG = 0.061%MoS2)

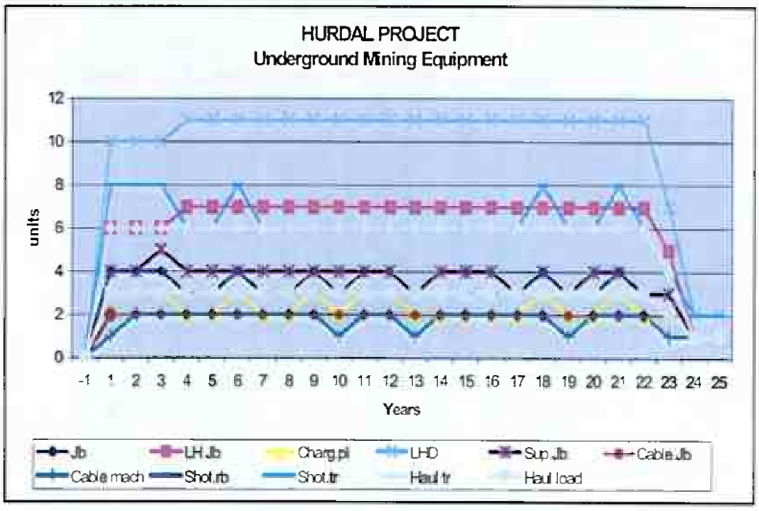


Figure 52 – Underground mobile equipment per year (Option 3 – 325 m pit COG = 0.091%MoS2)

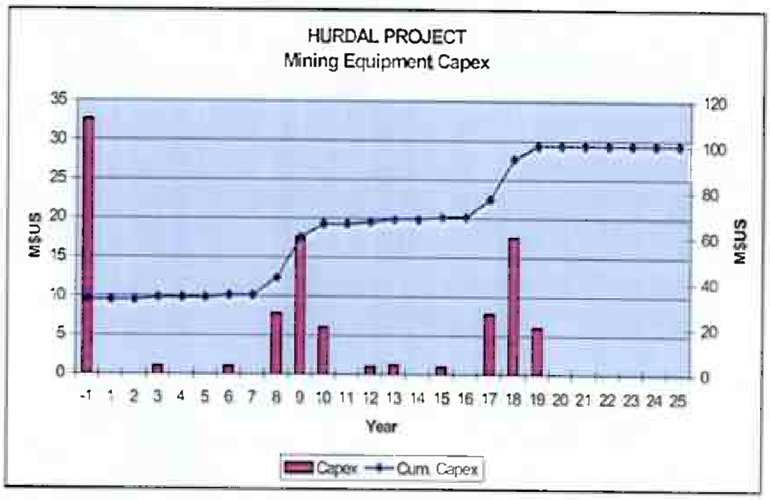


Figure 53 – Underground mobile equipment Capex per year (Option 3 – 325 m pit COG = 0.091%MoS2)

Figure 50 to 53 summarises the amount of mobile mining equipment needed per year, as well as the underground mobile mining equipment Capex (119 M$US for option 1 and 2 and 101 M$US for option 3 and 4).

11 Conclusions

Positive aspects of applying the SLOS method to Hurdal:

- Eliminates subsidence cone landtake.
- Reduction in landtake on the TMF, due to the backfill volume
- Possibility to use the open-pit to dispose the tailings, decreasing the landtake on the TMF, and reducing the environmental impact of the pit,.
- Increases feed grade to the plant, as Bench&Fill (SLOS) is a more selective mining method than for example SLC.
- Decreased underground water management problems, when compared with SLC;

Negative aspects of applying the SLOS method to Hurdal:

- Increased OPEX when compared with SLC.
- Underground mining reserve is reduced due to a higher cut-off, and access pillars.
- Potential geotechnical problems on the access pillars.
- Chances of increasing the estimated dilution and have secondary stopes stability problems due to imprecise ring drilling.
- Secondary stopes stability problems due to the "stalactite effect".
- All this potential geotechnical problems could affect expected productivity and decrease SLOS production.

Figure 54 summarises the main results for all the four options.

	Option 1 325 ($US24)	Option 2 275 ($US24)	Option 3 325 ($US16)	Option 4 275 ($US16)
Underground Production (Mtons)	157.6	166.9	110.9	116.4
% MoS2	0.122	0.121	0.138	0.137
Flat Develoment (m)	64,515	65,317	63,181	63,181
Vertical Development (m)	26,566	28,216	23,468	24,020
Opex ($US/t)	16.88	16.57	17.43	17.34
Development Capex (M$US)	93.8	93.8	90.9	90.9
Underground Mobile Equip Capex (M$US)	119.1	119.1	100.6	100.6
Plant Feed (Mtons)	221.6	217.5	174.9	167.2
Plant Feed (%MoS2)	0.109	0.110	0.116	0.118
Backfill volume (Mm3)	59.6	63.0	41.7	43.8
Tailings to TMF (Mtons)	131.9	122.4	112.2	101.4

Figure 54 – Main results for all options

4.0 TMF, ENVIRONMENTAL AND UNDERGROUND MINE INFRASTRUCTURE

Hurdal Molybdenum
Scoping Study for the Tailings Management Facility, Environmental and Social Review and Underground Mine Infrastructure

Scoping Study

Date 20 September 2007

CONTENTS

1. Introduction

The Hurdal molybdenum project is proposed by Crew Gold Corporation (CG)to mine a molybdenum ore body (approx. 155 M tonnes) over a 36 year period. The project includes for the first nine years production from an open pit down to the –75 m level (275 m deep pit) or the –25 m level (325 m deep pit), the remaining ore will be mined by underground methods. The mining studies are contained in reports by Edgar Urbaez (open pit design) and Diogo Caupers (underground mine design) presented direct to CG.

Scott Wilson Mining Group (SWM) has been requested by Aker Kvaerner (AK), who are coordinating the whole project report, to prepare a scoping level study to include the following works;

(1) Tailings Management Facilities;
(2) Environmental and Social Review;
(3) Mine Infrastructure (Shaft and underground infrastructure).

This report has been prepared using drawings and reports supplied by CG.

2. TAILINGS MANAGEMENT FACILITY

2.1 Introduction

This section of the report looks at the tailings management facilities (TMF) for permanent storage of the tailings produced throughout the mine life.
For the purposes of this report it is assumed that the tailings will be disposed of as a paste, with a portion of the tailings being used for underground backfilling. The total amount for deposition on the surface is estimated to be 133 M t, which equates to 67 million m^3 (at a paste discharge density of 2 t/m^3).

The tailings produced from the initial open pit production will be deposited in prepared surface containment embankments. In order to minimise the storage volume required in surface tailings embankments, it is proposed to use the open pit for storage of the tailings produced from the underground operations, once the open pit has been completed and underground mining has commenced. Careful consideration will have to be given to this proposal to ensure the safety of the underground workings.

SWM have also looked at possible after-use of the open pit and the following closure options have been considered:

- Construct an engineered domestic landfill cell above the deposited paste tailings in the open pit, and utilize the subsequent bio gas to generate electricity;
- Construct a recreational lake above the tailings deposited in the open pit.

2.2 Battery Limits

The battery limits for the TMF, for all of the TMF sites, is the first flange at the discharge side of the paste tailings pumps.

2.3 Design Parameters

The following initial design quantities have been used in this study based on densities of 2.8 t/m^3 for the ore and 2 t/m^3 for the tailings.

Description	Tonnage	Volume
Total tonnage of tailings for disposal	133 Mt	67 Mm^3
Volume of tailings from the first 9 years of open pit production for surface disposal in tailings management facilities	67 Mt	34 Mm^3
Remaining volume of tailings, from underground workings, for disposal in open pit	65 Mt	33 Mm^3
Total volume in open pit available for tailings disposal		56.4 Mm^3
Total volume in open pit available for landfill/lake		23 Mm^3

It will be necessary to carry out testing of both the tailings and waste rock and also carry out comprehensive site investigations of the proposed sites prior to any detailed design works being carried out.

2.4 Site Selection

No visit has been made to the site, and all proposals with regards to site selections, have been carried out using the plans (topographical, properties, water courses etc) provided by Crew Gold.
From the above plans it is evident that the mine site is within a steeply sided valley approx. 500 m wide, and given the required storage capacity (34 Mm^3) for surface disposal of tailings, the choice of suitable tailings management facility locations is very limited, and will possibly involve more than one site.

From a study of the available plans it is evident that there will be several engineering and environmental considerations to take into account with regards to the siting of the tailings management facilities including:

- water courses;
- physical topography;
- habitation,
- utilities;
- recreational use (skiing);
- land after use.

At this stage of the project and with the information available no definitive recommendations can be made as regards a location of the TMFs. However two possible sites have been identified which may prove suitable for the location of the TMFs. It should be noted that considerable further studies will be required to confirm the suitability of these sites.

The sites are shown on Drg. No. D116468/TMF/001 and are further detailed as follows.

2.41 Valley Impoundment

A valley to the north west of the proposed mine site has been identified as a possible site for containment of the tailings behind an engineered embankment. The top of the embankment would be at the 430 m level, at the location shown on Drg. No. D116468/TMF/001. The contained tailings volume would be 36 Mm^3. The volume required for the embankment construction would be in the region of 4 Mm^3.

There are several engineering problems associated with this option and are as follows:

(1) Contained in the valley is a river, which is fed from a lake above the valley. It is not possible from the plans to determine the size of this river but it will certainly need either diverting or possible culverting under the TMF. Depending on the ground conditions this work could incur significant costs, but an estimate for the works has been included in the CAPEX exercise.

(2) There are several residential or farm properties within the containment area, but it is not possible to determine the exact nature of these properties. Again there will be a cost involved in purchasing these properties. No cost has been included for this item. These costs are assumed to be in the owners costs for the project.

(3) From the plans it appears an overhead power line runs along the valley. This power line will need to be diverted again at a cost. No cost has been included for this item.

(4) No information is available to determine the vegetation/tree cover in this valley.

(5) There appears to be minor roads or tracks within the footprint of the lagoon that will need diverting around the TMF (approx. 2 km);

(6) Approximately 4 Mm^3 of suitable material for the construction of the tailings embankment will be required. A source for this material will need to be identified.

(7) There will clearly be environmental problems with this option and these are considered in the environmental review section of this report.

2.4.1.1 Tailings Production Facilities

The tailings produced will be as "paste tailings", and due to the distances between the plant and the site, it will be necessary to site the paste tailings plant, recovered water return pumps and the final paste tailings pumphouse adjacent to the tailings management facility site.

2.4.1.2 Tailings Delivery System

The embankment would be constructed in phases over the period of the open pit operation (9 years) and at each level, tailings would be discharged from a perimeter manifold type pipe system (raised at each embankment lift) to allow even distribution of the tailings over the full TMF area at each of the successive lifts. This would require careful monitoring to ensure falls towards the embankment to prevent surface water being trapped on top of the tailings.

2.4.1.3 Tailings Management Facilities Design

Due to the nature of paste tailings the leachate is considerably reduced, however in accordance with current International best practice, and most probably to comply with Norwegian legislation, SWM consider that an HDPE liner within the tailings management facility will be required, to minimise the possibility of leachate entering the environment.

A protection cushion layer of fine material (approx 300 mm thick) will be required beneath the HPDE liner, to prevent puncturing of the liner.

Basal drains will be included in the design to facilitate drainage of the tailings.

The embankment would be constructed from compacted suitable material (possible waste rock from the open pit excavation) in phases to suit the tailings production over the nine year period of the open pit works. The total quantity of suitable material required is 4 Mm^3. ARD test work will be required to check the suitability of the open pit waste rock for use in the embankment construction.

The embankment would be constructed using the down stream method, which will allow the successive lifts of liner to overlap the previous section.

At each level it will be necessary to construct a concrete surface water overflow channel to control surface water. This water will flow through water settlement ponds, which are considered necessary to prevent potential pollution of the water courses.

As mentioned previously it will be essential to either divert the valley river around the TMF or culvert the river under the TMF.

2.4.1.4 Tailings Management Facility Restoration.

In line with best International practice, SWM consider it will be necessary to cap and restore the site, (using material stripped from the TMF site initially and from the open pit site), once the tailings have gained sufficient strength to allow machine access. Replacement tree planting may also be required as part of the restoration works.

2.4.2 Paddock Type Deposition

This type of deposition normally requires a relatively "flat" area, and although there is no obviously "flat" area identifiable on the topographical plans, there is an area to the north of the mine site where it may be possible to construct a series of "paddock type containment cells. However it must be emphasized that until a detailed on site survey is carried out no definite conclusions can be made about the suitability of this site. This location is shown on Drg. No D116468/TMF/001

2.4.3 Tailings Delivery System

If the area identified on Drg.No.D116468/TMF/001 proves to be suitable, a possible method of disposal into a tailings management facility would be by a central disposal tower. At this stage it is not possible to identify the size of the cells from the plans available, the following simplistic conceptual layout for a flat site has been considered to illustrate the quantities involved in the disposal of the tailings.

- Using a central tower for disposal of the tailings with a height of 50 m and an angle of repose of 6° for the tailings, a cone of approximately 1000 m diameter will be formed. This will give a volume of approximately 13 million m^3. The tailings volume for disposal at Hurdal is 34 million m^3, which equates to three such cones or the equivalent area.

As each of the cones is completed it will be necessary to extend the delivery pipework to the next area and construct another discharge tower.

2.4.4 Paddock Type TMF Containment

Although in some environments it is possible to deposit paste tailings without constructing containment embankments, SWM consider that given the nature of the Hurdal environment and the location of the potential tailings management facility sites, it will be necessary to construct containment embankments to restrict and control the impact (surface run-off, bleed water, fines etc.) of the tailings deposition on the environment.

Again due to the local environment it will be necessary to construct water settlement ponds to prevent potential pollution of the water courses by run off from the tailings facilities.

Due to the nature of paste tailings the leachate is considerably reduced, however in accordance with current International best practice. SWM consider that an HDPE liner within the tailings management facility will be required, again this will be to mitigate the effects of any leachates on the environment. This may also be a mandatory requirement of the Norwegian legislation.

It should be noted that the tailings management facility design, based on the cone disposal as outlined in Section 0, is very conceptual in design and has been used only to illustrate the required quantities involved in disposing of the tailings.

Before any further design works can be carried out, significantly more information will be required to design a site specific tailings management facility for the Hurdal project. These requirements are listed in Section 0.

The perimeter containment bund will be constructed from compacted spoil removed from the tailings management facility area and waste rock from the open pit mining. Based on the disposal cone described in Section 0 approximately 120,000 m^3 of spoil will be required to construct each bund. To protect the HDPE liner a protection cushion layer (fine grained material) will be required over the base area of the tailings management facility. At 300 mm thick this will require 240,000 m^3 of material, and further studies will be required to identify sources for this material, with one possible source being suitably sized material from the open pit waste.

A cross section of a conceptual tailings management facility is shown on Drg. No. D116468/TMF/002. This is based on the above outline construction and the disposal system as described in Section 0.

In line with best International practice SWM consider it will be necessary to cap the tailings management facilities with suitable material to allow restoration of the tailings management facility sites.

During the open pit operations there is no requirement for paste tailings for underground backfill, and it is therefore possible that the open pit tailings could be deposited in a slurry form, offering an initial potential CAPEX saving on a paste plant. However this would present additional problems as listed below:

- Increased area required for disposal,
- Larger containment embankments;
- Increased risk from larger contained volumes of water;
- Increased times before sites can be restored.
- This option could only be pursued further on completion of detailed site studies.

2.5 *Open pit after use*

Scott Wilson has carried out a conceptual review of the following two options for the afteruse of the open pit, following the completion of the disposal of the underground tailings.

2.5.1 Landfill cell

The remaining volume in the open pit, after deposition of the underground tailings, will be in the region of 23 M m^3 (approx. 40 m deep). One of the closure options is to construct an engineered domestic landfill cell in this void, and use the bio gas from the waste to power an electricity generating station. An approximate rate of production of household waste for 1 million people is in the region of 500,000 m^3 per annum. If waste were available from a population centre of this size then at this rate it would take 46 years to fill the proposed cell; clearly not viable.

In order for gas to be available for generation purposes it could be possible to construct smaller cells within the main void to allow gas to be produced at an earlier stage.

The nearest large population centre is Oslo some 60 km distant and transport charges of waste to the Hurdal site may make this option uneconomic at the outset.

The ability of the local road infrastructure to support the increased road traffic will certainly have to be assessed, along with other significant environmental impacts.

At this stage of the project the rate of gas production/capacity of generating station is unknown, but as a guideline cost, a bio gas fuelled 1 MW generating facility will be in the region of $1 M.

There are several other considerations that would need further investigation to determine the overall viability of this proposal. These are listed in Section 0

2.5.2 Amenity Lake

Following completion of the underground mining works it would be a relatively straightforward exercise to divert the river into the open pit void to form a lake for recreational afteruse. If the open pit is used for tailings disposal only, the depth of the lake would be in the region of 40 m. If this proves too deep for recreational afteruse, additional waste will have to be sourced and tipped into the open pit void. In any event a rock cap will be required.

As with other aspects of the Hurdal proposal further studies (engineering and environmental) would be necessary to determine the viability of this option. These are listed in Section 0

Dependant on the permitting conditions it may be necessary to carry out some engineering and landscaping works to provide an acceptable lake profile and facilities to satisfy the permitting authorities.

At this stage of the project there is insufficient information to be able to cost this closure option.

2.6 River diversion

From the conceptual open pit profiles it appears that the south western section of the pit intersects the main river course in the Hurdal valley. There is also a smaller river coming from the north of the open pit that also intersects the open pit.

The main river will have to be diverted to a safe distance to the south west of the rim of the open pit, involving a diversion channel approximately 1 km long and some reprofiling of the hillside to facilitate the construction of the channel; or if the reprofiling works proved to too excessive, a diversion tunnel would need to be constructed.

From the CAPEX costings, presented later, it is clear that the open cut river diversion is the most economic. However it should be noted that no account has been taken of possible environmental impacts of an open cut channel, nor of any geotechnical and construction restraints.

The smaller river would require a diversion channel of approximately 200 m.

Excavated materials could be utilized for the construction of the tailings management facility embankments.

2.7 CAPEX

The following cost estimates have been generated from typical database construction and material costs.

2.7.1 Valley Impoundment Tailings Management Facility

The CAPEX estimate for this type of facility are shown below

Table 2

	Valley Dam TMF				
Item	**Item description**	**Quant**	**Unit**	**Rate ($)**	**Cost ($)**
Sect 1	**Earthworks**				
1.1	Grub out trees	200	Ha	$2,000	$400,000
1.2	Prepare site & soil strip (300 dp)	600000	m^3	$2	$1,200,000
1.3	Stockpile soils (200m distant)	600000	m^3	$2	$1,200,000
1.4	Excavate for bund foundation	200000	m^3	$6	$1,200,000
1.5	Build compacted rock dam (Rock from open pit and/or site strip material)	4000000	m^3	$2.5	$10,000,000
1.6	Excavate for perimeter cut off drains	27000	m^3	$6	$162,000
1.7	Load, haul and place material for TMF capping	84000	m^3	$3	$252,000
1.8	Restoration to TMF capping	200	Ha	$4,000	$800,000
	Total for Section 1				**$15,214,000**
Sect 2	**HDPE Liner**				
2.1	Lay compacted liner protection layer	600000	m^3	$3	$1,800,000
2.2	Install HDPE liner	200000	m^2	$5	$1,000,000
2.3	Install basal drains	15000	m	$20	$300,000
2.4	Install leak detection system	1	Item	$100,000	$100,000
	Total for Section 2				**$3,200,000**
Sect 3	**Concrete works**				
3.1	Pipe plinths	100	m^3	$400	$40,000
3.2	Pump station	100	m^3	$400	$40,000
3.3	Drainage spillways (6No.@ 100 m^3)	600	m^3	$400	$240,000
	Total for Section 3				**$320,000**
Sect 4	**Ancillary works**				
4.1	Supply and install stock proof fencing	10000	m	$10	$100,000
	Total for Section 4				**$100,000**
	Sub total for Section 1 to 4				**$18,834,000**

Sect 5	Mechanical works				
5.1	Supply & install tailings delivery pipe	1000	m	$80	$80,000
5.2	Supply & install tailings distribution pipework	10000	m	$80	$800,000
5.3	Supply & install return water pipework	1000	m	$50	$50,000
5.4	Return water pump house	1	item	$10,000	$10,000
5.5	Return water pump	1	item	$4,000	$4,000
	Total for Section 5				**$944,000**
Sect 6	River Culvert/Diversion				
6.1	Excavate for concrete culvert	3000	m^3	$3	$9,000
6.2	Construct concrete culvert	3000	m	$2,000	$6,000,000
6.3	Associated concrete works	200	m^3	$400	$80,000
	Total for Section 6				**$6,089,000**
Sect 7	Prelims and General				
7.1	Engineering & supervision - Civil	2.75%	item	$24,923,000	$685,383
7.2	Engineering & supervision - Mechanical	8.00%	item	$944,000	$75,520
7.3	Site investigation - Civil	0.50%	item	$24,923,000	$124,615
7.4	Contracators prelims & general	7.50%	item	$24,923,000	$1,869,225
	Total for Section 6				**$2,754,743**
	Total cost				**$28,621,743**

2.7.2 Paddock Type Tailings Management Facility Construction.

The CAPEX estimate for this type of facility are shown below

Table 3

	Typical cone deposition 50m high x 1000m dia.				
Item	**Item description**	**Quant**	**Unit**	**Rate ($)**	**Cost ($)**
Sect. 1	**Earthworks**				
1.1	Grub out trees	79	Ha	$2,000	$157,100
1.2	Prepare site & soil strip (300 dp)	235650	m^3	$2	$471,300
1.3	Stockpile soils (200m distant)	235650	m^3	$2	$471,300
1.4	Excavate for bund foundation	18852	m^3	$10	$188,520
1.5	Build compacted rock bund (Rock from open pit and/or site strip material)	113112	m^3	$20	$2,262,240
1.6	Excavate for toe drains	3142	m	$3	$9,426
1.7	Load, haul and place material for TMF capping	785500	m^3	$10	$7,855,000
1.8	Restoration to TMF capping	79	Ha	$4,000	$314,200
	Total for Section 1				**$11,729,086**
Sect. 2	**HDPE Liner**				
2.1	Lay compacted liner protection layer	785500	m^2	$3	$2,356,500
2.2	Install HDPE liner	785500	m^2	$5	$3,927,500
2.3	Install basal drains	6000	m	$20	$120,000
2.4	Install leak detection system	1	Item	$100,000	$100,000
	Total for Section 2				**$6,504,000**
Sect. 3	**Concrete works**				
3.1	Pipe plinths	20	m^3	$400	$8,000
3.2	Pump station	100	m^3	$400	$40,000
3.3	Drainage spillways	200	m^3	$400	$80,000
	Discharge tower foundation	100	m^3	$400	$40,000
	Total for Section 3				**$168,000**
Sect. 4	**Ancillary works**				
4.1	Supply and install stock proof fencing	3142	m	$10	$31,420
	Total for Section 4				**$31,420**
	Sub total for Section 1 to 4				**$18,432,506**

Sect. 5	**Mechanical works**				
5.1	Discharge tower	15	tonnes	$6,000	$90,000
5.2	Supply & install tailings delivery pipe	1000	m	$80	$80,000
5.3	Supply & install tailings distribution pipework	500	m	$80	$40,000
5.4	Supply & install return water pipework	1000	m	$50	$50,000
5.5	Return water pump house	1	item	$10,000	$10,000
5.6	Return water pump	1	item	$4,000	$4,000
5.7	Return water pump access jetty	1	item	$10,000	$10,000
	Total for Section 5				**$284,000**
Sect. 6	**Prelims and General**				
	Engineering & supervision - Civil	2.75%	item	$18,432,506	$506,894
	Engineering & supervision - Mechanical	8.00%	item	$284,000	$22,720
	Site investigation - Civil	0.50%	item	$18,432,506	$92,163
	Contractors prelims & general	7.50%	item	$18,432,506	$1,382,438
	Total for Section 6				**$2,004,214**
	Total cost for one tailings cone				**$20,720,720**
	Total number of cones required =	**3**	**Total lagoon cost**		**$62,162,161**

2.7.3 River diversion around Open Pit

Costs estimates for the river diversion are provided below based on preliminary estimated quantities and database unit costs.

Table 4

Item	Item description				
Sect. 1	**Earthworks**	**Quant**	**Unit**	**Rate**	**Cost**
1.1	Grub out trees	2	Ha	$2,000	$4,000
1.2	Soil strip over length of diversion (300 dp)	3600	m^3	$2.00	$7,200
1.3	Stockpile soils (200m distant)	3600	m^3	$2.00	$7,200
1.4	Excavate for channel in hard rock	14400	m^3	$88	$1,267,200
1.5	Reprofiling of hillside	60000	m^3	$5	$300,000
1.6	Restoration of construction area	2	Ha	$4,000	$8,000
	Total for Section 1				**$1,593,600**

2.7.4 Tunnel diversion around Open Pit

Costs estimates for placing the river diversion within a tunnel are provided below based on preliminary estimated quantities and database unit costs.

Table 5

Item	Item description				
Sect. 1	**Earthworks**	**Quant**	**Unit**	**Rate**	**Cost**
1.1	Construct 2 m dia. Tunnel in hard rock	1000	m	$3,000	$3,000,000
1.2	Concrete works	100	m^3	$400	$40,000
	Total for Section 1				**$3,040,000**

2.8 Conclusions

From the above cost estimates it is clear that the valley embankment option is the most economical at a total cost of $28.6 M. However this option will still present several engineering and environmental challenges that will require further studies to be carried out.
These studies along with other recommended studies are listed in Section 2.9 below.

2.9 *Recommendations*

To move to the next stage of project study, it is considered necessary to carry out further engineering and environmental studies as listed below.

2.9.1 Site Selection

- Detailed on-site topographical surveys to determine suitability of proposed tailings management facility sites;
- Prepare Environmental Impact Assessment;
- Ecological studies;
- Determination of environmental legislation applicable to tailings management facility construction;
- Site investigations (boreholes, trial pits, lab testing etc) to determine:
  - Geotechnical properties of materials to be used for construction (paste tailings, waste rock, overburden etc.)
  - Quantities of overburden to be stripped at tailings management facility sites;
  - Hydrology and hydrogeology of proposed tailings management facility sites;
  - Meteorological studies (rainfall, snowfall, winds etc.);
- Assessment of affected properties, and likely compensation costs.
- Likely requirements for restoration of TMF sites.

2.9.2 Open Pit After use

2.9.2.1 Landfill Cell

- This proposal would have to be considered within the scope of the overall project EIA;
- Geotechnical, hydrology and hydrogeology studies of the landfill site and surrounding areas, to include boreholes, trial pits etc;
- Ecological studies;
- Prepare a financial model to determine financial viability, to include:
  - Cell construction costs (based on a approximate cost of \$200/m^2, this cost is likely to be in the region of \$100 M for the full area of the open pit);
  - Possible phased cell construction to allow early generation of gas/electricity;
  - Infrastructure survey and possible improvement costs;
  - Power station construction costs; (approx. \$1M/MW)
  - Study to determine sources of waste and likely quantities;
  - Likely income from disposal charges;
  - Site reclamation costs and possible maintenance costs (possibly "until the cell is shown to present no significant impacts on the environment") of the cell in line with legislation.

2.9.2.2 Recreational Lake

- This proposal would have to be considered within the scope of the overall project EIA;
- Geotechnical, hydrology and hydrogeology studies of the proposed lake site and surrounding areas, to include boreholes trial, pits etc;
- Discussion with authorities to determine acceptable final use (sailing, fishing etc.);
- Engineering studies/designs for landscaping and social facilities (jetties, boat houses etc.);
- Carry out cost study following agreement of final use and required amenity facilities;
- Ecological studies.

2.9.3 River Diversion

- This proposal would have to be considered within the scope of the overall project EIA;
- Geotechnical, hydrology and hydrogeology studies of the proposed diversion route and surrounding areas, to include boreholes trial pits etc;
- Ecological studies.

2.10 References

(1) Paste – The Future of Tailings Disposal, Golder Associates (UK) Ltd. England. (Phil Newman, Roger White, Alistair Cadden) (http://www.golder.com/archive/skelleftea2001pastedisposal.pdf)
(2) SPONs Pricing Guide Book 2007.

3.0 Environmental and social review

3.1 Introduction

Conceptual plans are being drawn up for a proposed molybdenum mine in the vicinity of Hurdal, Norway. As part of the planning process, Scott Wilson has been asked to provide an environmental and social review. The purpose of this review is two-fold:

(1) To provide a review of Norwegian environmental legislation and to identify Acts and regulations relevant to the proposed mine; and
(2) To comment on the environmental and social setting of the mine and to identify potential significant environmental and social issues and highlight "showstoppers".

3.2 *Background Information*

The mine will comprise an open pit operation for the first 9 years followed by 26 years of underground mining. Therefore the pit and underground workings, waste rock disposal, and tailings facilities all need to be considered in terms of the environment. Furthermore a river runs through the site, so impacts on this river and possibly other watercourses need to be considered.
Background information presented to Scott Wilson includes a basic description of the proposed mine, and topographic plans. A conceptual mine layout has been included on the topographic plan and potential sites for the conceptual Tailings Management Facility (TMF) are indicated.
Based on the above available information, the environmental and social review is a high level assessment that highlights the most pertinent environmental issues.

3.3 *Relevant legislation*

3.3.1 General

In Norway, the Ministry of Environment is responsible for overall environmental policy. Environmental Impact Assessment (EIA) has been adopted as part of the Planning and Building Act of 14 June 1985 (14.6.1985 No 77) since 1990 and, in 1999, the field of application was expanded with devolution of tasks to local authorities.

3.3.2 Mining Legislation

Mining operations may be subject to the following regulations.

- Mining Act, As Amended And Supplemented, 1990.
- Act No. 70 of 30 June 1972 (Effective 1 April 1974) On Mining, With Regulations (Replaces 1842 Mining Act).
- Royal Decree of 31 January 1969 Relating To Scientific Research For Natural Resources On The Norwegian Continental Shelf etc.
- Act 1 of 21 March 1952 Relating To The Surrender Of Land etc. For The Working Of Non-Claimable Mineral Deposits.
- Concession Act of 17 June 1949.
- Act of 15 February 1946 To Amplify Mineral Industry Legislation (Reserving (Reserves) Minerals Of Special Importance To The State For Licensing).
- Mining Ordinance For Svalbard Given By Royal Decree of 7 August 1925.
- Spitsbergen (Svalbard) Mining Code 1920.
- Act of 14 December 1917, (No. 16), Relating To The Acquisition Of Waterfalls, Mines And Other Real Property Etc. (As Amended).
- Act of 11 March 1905, Embodying Amendments & Additions To The Mineral Industry Act Of 14 July 1842.
- Mineral Industry Act of 14 July 1842.

3.3 Environmental Impact Assessment

The provisions of the Planning and Building Act, Chapter VII-a, apply to plans as specified in the Act (Section 16.2) and to certain specified plans and projects pursuant to other legislation, which may have significant effects on the environment, natural resources or the community. Mining operations are classified in the Act under Appendix 1 as a development that requires EIA prior to authorisation. Furthermore, the Strategic Environmental Assessment Directive was implemented on 1 April 2005, which may be applicable in this case.

The current regulations are:

- Regulations on Environmental Impacts Assessment, laid down by Royal Decree of 1 April 2005 (1.4.2005 No 276).

These regulations amend the Planning and Building Act comparable to the following EU legislation:

- The EEA Agreement, Directive 2001/42/EC of the European Parliament and of the Council of 27 June 2001 on the assessment of the effects of certain plans and programmes on the environment.
- Council Directive 85/337/EEC on the assessment of the effects of certain public and private projects on the environment.
- Council Directive 97/11/EC of 3 March 1997 amending Directive 85/337/EEC on the assessment of the effects of certain public and private projects on the environment.
- Directive 2003/35/EC of the European Parliament and of the Council of 26 May 2003 providing for public participation in respect of the drawing up of certain plans and programmes relating to the environment and amending with regard to public participation and access to justice.
- Council Directive 96/61/EC of 24 September 1996 concerning integrated pollution prevention and control.

Note that although Norway is not a member of the European Union, it is a member of the European Economic Council and therefore implements many EU Directives and Legislation.

These regulations also relate to the UN-ECE Convention on Environmental Impact Assessment in a Transboundary Context (Espoo, Finland, 25 May 1991) and the UN-ECE Protocol on Strategic Environmental Assessment (Kiev, Ukraine, 21 May 2003) and repeal the 1999 Regulations Relating to Environmental Impact Assessment (21.5.1999 No 502).

3.3.4 Water Resources Legislation

Under Norwegian Law, a watercourse belongs to the owner of the land it runs through so important limitations are placed on the use of watercourses with respect to neighbouring properties.
The general statute governing fresh water resources, including ground water, is the Water Resources Act (21.11.2000 No 82). Other possible legislation affecting watercourses is as follows:

- The Watercourse Regulation Act Course Act No.16 of 14 December 1917 relating to the regulation of watercourses.

- Industrial Licensing Act, Act No. 16 of 14 December 1917 relating to the acquisition of waterfalls, mines and other real property etc.
- The Neighbouring Properties Act, Act No. 15 of 16 June 1961 relating to the legal relationship between neighbouring properties.
- The Pollution Control Act, relating to protection against pollution and relating to waste (13.3.1981 No. 6).

Further regulations may be applicable if any work is to be undertaken in the watercourse itself.

3.3.5 Appropriate Authority

The Ministry of the Environment and the Ministry of Trade and Industry are the highest authorities for development projects requiring EIA. Appendix 1 of the Regulations on Environmental Impacts Assessment (1.4.2005 No. 276) outlines the relevant authorities and Acts under which developments require EIAs.

The Norwegian Directorate of Mining is a department within the Ministry of Trade and Industry with the responsibility of administrating extraction of mineral resources. The Directorate's authority includes administration of the mining legislation, registering claims and approving mine plans, as well as being the supervisory authority for EIAs for planned extraction sites. The professional basis for decisions regarding EIAs is the responsibility of the Department of Regional Planning, Section for EIA, while the Norwegian Pollution Control Authority is responsible for providing the professional basis for decisions for the Ministry of the Environment in connection with pollution issues.

3.4 Review of Environmental Issues

3.4.1 Topography

The proposed mine lies within rugged mountainous terrain and is sited in a relatively broad (approximately 500 m wide), u-shaped river valley. Thus, there are severe limitations on the possible location and layout of the infrastructure, plant and tailings management facilities. The implications of the rugged topography will have to be assessed as the conceptual layout plans develop e.g. potential for erosion/creep/slope failures if tailings are stacked on undulating terrain, and numerous points where surface water discharges in and around the mine site.

At this conceptual design stage, the diameter of the pit at surface level is approximately 800 m wide. This implies that re-profiling and landscaping of the northeast side of the valley will be a requirement. This has additional implications for the visual impact of mining.

3.4.2 Surface Water

The mine site lies in a main river valley with numerous tributaries and surface depressions draining from the adjacent mountains. Based on this observation it is apparent that surface water, in terms of quality and quantity, will be an issue for the mine, and relevant legislation will need to be strictly adhered to.

From the topographical plans it is apparent that the main valley river runs through the proposed open pit, and the remainder of the mine lies between two river courses, the confluence of which lies approximately 250 m to the southeast of the open pit. If the conceptual plan layout is adhered to, river diversion and protection works will be required.

The areas marked for potential TMFs will have to be assessed in terms of river catchment impact because they lie on a series of watersheds and rivers and could present significant impacts on the quality of surface water.

Specialist hydrological studies will be required to assess the impact on the immediate and greater catchment areas. It is apparent that extensive mitigation, monitoring and management measures will be required throughout the life of the mine and following closure.

3.4.3 Groundwater

No information could be gleaned from available data. However, given the setting it is most probable that groundwater will be a significant issue.

3.4.4 Visual Impact

The visual impact could be a significant factor as this area is semi-rural and a developed ski-ing resort. The TMFs in particular, due to their potential size and locations, could present a significant impact on the landscape, e.g. those on mountain tops will be clearly visible from certain areas and possibly from ski lifts.

3.4.5 Climate

The extreme climate will present some issues for the mine e.g. freeze-thaw that may affect the integrity of the TMF.

Similarly, intense rain and heavy surface flows associated with post-freezing thaw could be a significant issue to be managed as part of an overall storm-water management system.

3.5 Socio-economic Issues

Hurdal is an established ski centre with 4 ski lifts and 10 runs (internet reference). Due to the close proximity to Oslo, it is readily accessible and will therefore have a significant amenity value.

There are numerous buildings shown on the topographic plan but no details of their use are given.

Ski lifts cross the area though the exact routes are not referenced on the plan. Visual impact will be a significant issue due to the proximity of the resort to the proposed mining operations.

It is likely that at least some of the buildings are holiday homes as well as residential properties. The main concentration of buildings lies along the main river valley, the lower slopes of the mountains, along the power line routes, and adjacent to lakes.

Many buildings lie within the proposed layout plan of the mine and indicate that resettlement is likely to be a significant issue entailing an extensive negotiation process, including compensation claims. Consultation with stakeholders will be key to the success of the project.

3.6 End-Use

Initial proposals for end-use include the following

3.6.1 Developing a landfill with potential for developing a bio-gas facility

Developing a landfill raises the issue of sourcing sufficient volumes of waste to support a bio-gas facility. It is estimated that the volume of the open pit, after the mine has closed, will be approximately 23 M m^3 with an estimated depth of 40 m.

The conceptual proposal for a landfill is that an engineered cell will be constructed within the pit for the disposal of household waste. Bio-gas generated from the waste could then be used to power an electricity generating station.

It has been estimated that it would take 46 years to fill the pit, based on an average waste stream produced by a population of one million people. (500,000 m^3 of waste per annum). It is possible that constructing a number of smaller cells, instead of one big one, will enable gas to be produced at an earlier stage.

However, transporting waste to site will be a significant issue e.g. the number of lorries and frequency of journeys, access to the site during the winter months. In addition, the suitability of the terrain for developing a landfill will have to be investigated, particularly given the tourism potential of the area and this will need to be subject to an EIA either within the overall mining project assessment or as a separate project.

3.6.2 Creating a lake for amenity value

There is obviously potential for developing the pit as a recreational site e.g. a lake. There are numerous lakes in the area and this would seem to have the potential to be acceptable as an end-use.

However, the end-use will have to be part of the consultation process to allow stakeholders to give input and ideas and get their overall "buy-in" to the proposals.

3.7 TMF

The tailings will be disposed of as a paste. It has been proposed that paste is pumped and allowed to settle to the natural angle of repose. As the tailings will be paste there will be reduced seepage of water, and in certain countries and environments the tailings have been deposited without containment bunds. However, with reference to the prevailing climatic conditions and the rugged terrain in the vicinity of Hurdal, the potential for erosion makes this seem an unsuitable design.
It is also considered necessary that the TMFs have a HDPE liner to ensure there is no contamination of the environment from the contained materials. This is in line with best International practice and probable Norwegian legislation.

At any of the possible TMF sites it is probable that extensive restoration works will have to be carried out following completion of the surface disposal of the tailings.
The options for sites for the TMF include:

3.7.1 Plateau areas on the tops of the mountains

In this rugged terrain plateau areas are not common and they are very limited in extent. The net result being that, to make maximum use of the land for a TMF, tailings will need to be stacked high or more than one site may be required to accommodate the volume of tailings. In any event, there will be visual impact, the significance of which can only be determined as plans develop and the exact site(s) and extent are confirmed. If more than one site is used for the TMF this will have a negative impact because the mine footprint will increase and, for example, result in increased change of land use and increased visual impact.

Using mountain tops for the TMF also increases the risk that any products of erosion e.g. soils and / or tailings, will be carried downhill and spread over wider areas. In the event of failure of the TMF, the impacts would be significant as several catchments would be affected by material flowing away from the TMF.

3.7.2 Valley Impoundment.

The second option is to site the TMF in a valley by constructing an embankment across the width of the valley. The most significant and direct impact will be on the rivers. Structures to divert the flow and separate it from the tailings e.g. culverts, will have to have sufficient capacity to ensure that both the quantity and quality of water will not be impacted by the TMF.

Consideration will have to be given to the affect of freeze-thaw as the seasons change.

The possible site indicated on the plan, shows that flow into the valley also takes place from the sidewalls. Thus, should this option be pursued, cut-off drains will have to be constructed along the valley sides to divert surface run-off around the TMF.
A hydrological study will be essential if this option is to be pursued beyond the conceptual design stage. The entire river catchment, or catchments, will have to be identified to assess the extent of potential impacts.

From the plans supplied it is apparent that several properties lie within the footprint of the embankment and will need relocating/purchasing.

3.8 Conclusions and recommendations

It is concluded that the following will be potentially significant environmental and social issues at Hurdal;

- Surface water;
- Although there is no data available it is apparent that groundwater will need assessing;
- Existing land use;
- Landowners and other stakeholders;
- Almost certainly there will be a need for resettlement;
- Landscape and visual impact;
- Restoration of TMF sites;
- It is further concluded that there is a comprehensive range of existing environmental legislation. The Norwegian Directorate of Mining has the responsibility of administrating extraction of mineral resources and should be the first point of contact should the proposed mine be pursued;
- The above conclusions are based on the limited information provided to Scott Wilson and it should be noted there may be other significant issues identified if the plans to mine are pursued.

It is recommended that:

- Based on a preliminary assessment of the TMF, it is recommended that the design includes a system of containment bunds and HDPE lining;
- A consultation process with stakeholders, initially government, to assess the accessibility of placing the mine and associated infrastructure in this ski-ing area. It would be useful to use this opportunity to discuss possible locations for tailings disposal, including hill tops. A lack of acceptance of the project concept at an early stage could prove to be a show-stopper; and
- Once a clearer idea is obtained of how the project is to be progressed it would be useful to engage with other stakeholders, such as local communities, tourism organisations and regional authorities. This will need to be carefully timed and managed, as it appears that the Project will have significant social impact potentially requiring resettlement and it will certainly impact on the existing landscape.

4. UNDERGROUND MINE INFRASTRUCTURE

4.1 introduction

The overall methodology of ore extraction for the Hurdal Mine is planned to be a combination of open cast and underground mining

The underground mining method and layout has been developed in another section of the report, generated by D. Caupers.

As presently envisaged the open pit is to be operational for 9 years, during which time the underground mine will be developed and will start operations at the end of open pit mining. The underground mine has a life of a further 26 years.

In this review an alternative mining method to the Sub Level Caving (LC) method originally envisaged by SRK has been selected. The chosen extraction scheme is a stacked Bench and Fill system. This mining method has many advantages in terms of the environmental impact of the project in that surface caving is eliminated and the placing of backfill reduces the volume of surface tailings disposal.

The generalised mine layout is shown below in Figure 1.

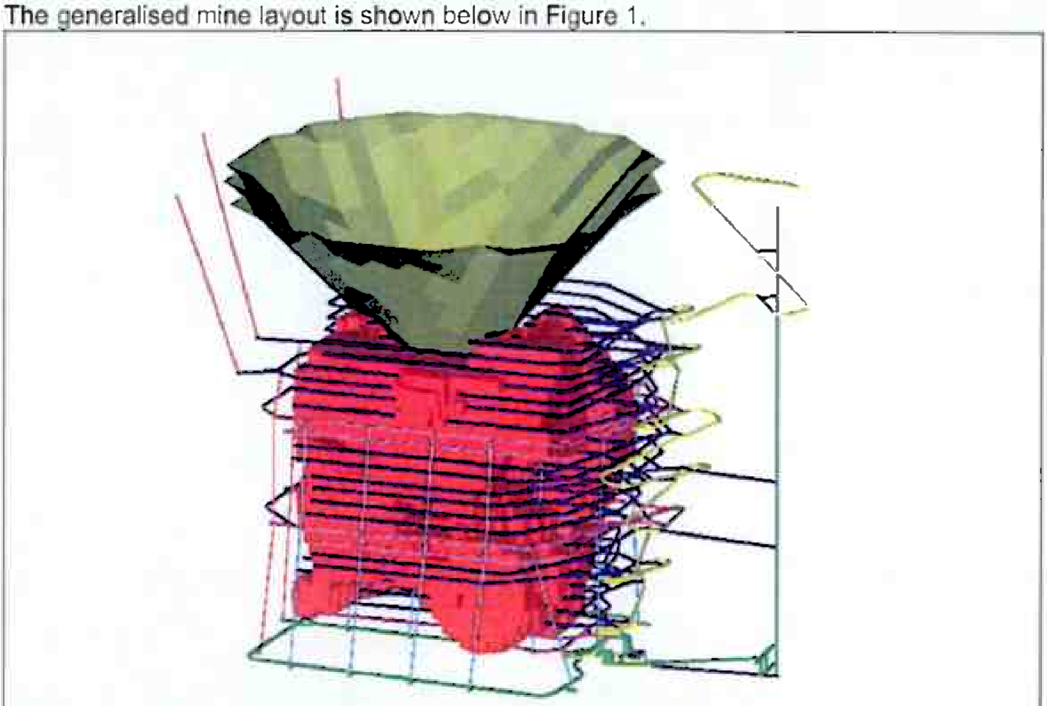


Figure 5 Overall Mining Layout

This section of the report focuses on the underground mine infrastructure.

4.2 *Mine Infrastructure*

The underground mine infrastructure presented in the study consists of a twin main mine access system, consisting of a vertical shaft approximately 830 m in depth and an inclined ramp, both commencing at a surface level of approximately 260 m RL.

The shaft is to be equipped as the main ore hoisting system, while the ramp will be used for man and equipment transportation.

Cost estimates for the various elements have been taken from recent studies carried out for similar sized mines and production requirements. Where applicable the rates have been amended to take into account Norwegian conditions.

CAPEX estimates have been developed for the following mine infrastructure elements:

(1) Shaft and station construction;
(2) In-shaft equipment and hoisting systems including the headframe;
(3) Mineral handling scheme from the crusher tip chamber to the shaft;
(4) Main ventilation fans;
(5) Main pumping scheme;
(6) Electrical distribution scheme.

OPEX has been estimated for the above fixed and mobile equipment in the mining section of the report.

Battery Limits

The battery limits for this section of the report are as follows: -

- For underground construction the limits are from the Crusher Chamber through to the shaft;
- Shaft limits include all shaft bottom arrangements, shaft stations plus 10 m lateral development on connecting levels;
- Rock handling will cover from the Crusher tipping point to the head frame discharge chutes;
- Electrical distribution from the out feed side of the suppliers' main sub station(s);
- Mine water handling from the intake to the settlers to a discharge point within 30 m of the shaft collar.

Shaft construction

The underground mine will be accessed by a combination of a vertical hoist shaft and a surface ramp system.

For the purposes of this study, the shaft has been located at a position outside of the known ore zone and at a relatively level site location. It may be that during the next stage of engineering the shaft and the ramp is re-located to another location if this is beneficial to the overall planning of the mine.
The shaft will be constructed from a shaft platform with a shaft collar level of +260 m RL.
The shaft system envisaged in this study has been based on the following parameters:

Shaft Output:	6.0 Mt/year
Operating hours:	6 day hoisting, 50 weeks per year
Shaft collar level:	260 m AOL
Skip discharge level at surface:	270 m AOL
Shaft bottom:	-600 m AOL (20 m below lowest production haulage level)
Hoisting distance:	825 m

In order to achieve the required mine output (6 Mtpa) and to meet ventilation requirements, a shaft with a finished internal diameter of 6.0 m is required.

For this study it has been assumed that there will not be any severe hydrological conditions to deal with in the shaft. Under these conditions, the shaft will typically be lined with cast in-situ concrete with a nominal thickness of 300 mm. This is not sufficient to provide a watertight lining, which would require a lining thickness greatly in excess of 300 mm but is adequate for "normal" ground and water conditions. Minor water inflow into the completed shaft and future water ingress, will be captured by water garlands included in the shaft lining.

The following shaft stations to the ramp and levels have been considered:

Ramp access:	175 m AOL
	90 m AOL
	-225 m AOL
	-380 m AOL
Access to top of shaft storage silos:	-525 m AOL
Access to shaft bottom for spillage clearance, pumping and inspections:	-600 m AOL

4.5 Hoisting Arrangement

The general topography of the Hurdal mine location is rugged and mountainous, with very steep valley sides and few level areas. Consequently there are very limited areas of flat ground available to site the shaft headframe and winder house.

To minimise the land-take a tower-mounted winder has been considered since this requires a smaller footprint for the shaft platform.

For the purpose of this report, a shaft head hoisting arrangement consisting of a concrete head tower with a multi-rope friction hoist mounted within it has been considered. During the next stage of study, the hoisting facility will need further consideration since the use of a tower mounted winder may be considered to have an unacceptable visual impact in a rural and ski-ing area.
The in-shaft arrangement considered, allows for a balanced twin skip system rope guides with provision of an independent man rider to act as the second means of egress through the shaft.

4.6 Hoist Capacity

The following parameters have been adopted for the purposes of the study:

Daily output:	20000 t per day
Hoist utilisation (allowing for hoist exams, shaft exams, rope exams etc):	20 hours
Hoisting distance:	825 m
Hoist Availability:	95%
Hoist Output per hour:	1053 tonnes per hour

Based on the above parameters a typical cycle time for a twin skip system is shown in

Table 6.

Operation	Speed m/s^2	Time (Seconds)	Travel (metres)
Forward Run			
Accelerate to full speed	0.8 m/s^2	20.00	160.00
Full speed	16 m/s	31.20	499.00
Decelerate to creep	0.8 m/s^2	18.12	158.59
Creep speed	1.5 m/s		
Creep distance	6 m	4.00	6.00
Decelerate to stop	0.8 m/s^2	1.87	1.41
Charging time	13 secs	13.00	0
Totals		**88.2**	**825.00**

Reverse Run			
Accelerate to full speed	0.8 m/s^2	20.00	160.00
Full speed	16 m/s	31.20	499.00
Decelerate to creep	0.8 m/s^2	18.12	158.59
Creep speed	1.5 m/s		
Creep distance	6 m	4.00	6.00
Decelerate to stop	0.8 m/s^2	1.87	1.41
Charging time	13 secs	13.00	0
Totals		**88.2**	**825.00**

Table 6 – Hoisting Cycle

From the above cycle times, 40 skips per hour is achievable, requiring a skip capacity of 27 tonne. For the purpose of the study, 30 tonne skips are adopted as a suitable size for the throughput required.

4.7 In-shaft arrangement

The layout of the conveyances within the shaft is based on utilising a balanced two skip system for the hoisting of ore with a small independent ("Mary-Anne") cage for carrying men and the second means of emergency egress.

The proposed twin 30 tonne skips will be accommodated in a 6.0 m diameter shaft, as shown in

Figure 6.

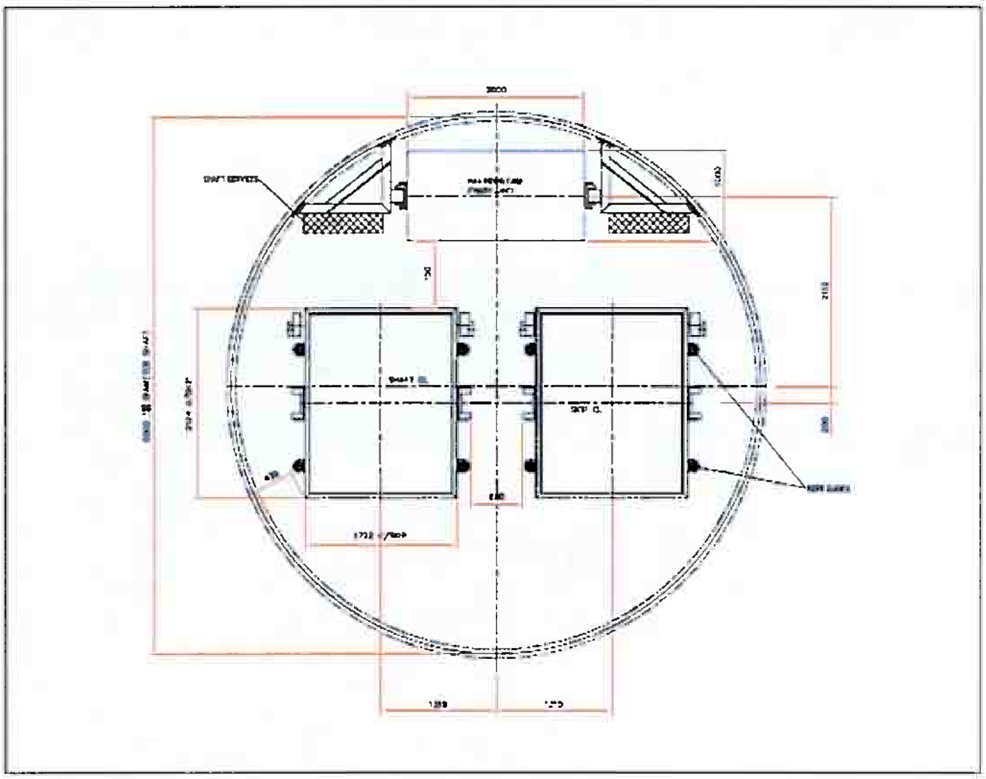


Figure 6: Typical Shaft Arrangement

4.8 Underground Ore Handling

The ore handling system is based on ore passes transferring ore and development rock from the production levels to the bottom of the mine where it is tipped into a primary crusher before being conveyed to the shaft and finally hoisted to surface.

The ore will be mucked from the stopes by LHD and dropped down to a haulage level at -580 m RL via ore passes, where loaders will charge 50 tonne trucks that then transport the ore to the primary cone crusher.

After crushing, the ore will be conveyed to surge silos adjacent to the shaft, and finally to the shaft skip loading facility for hoisting to surface.

A schematic of the arrangement is shown in Figure 7.

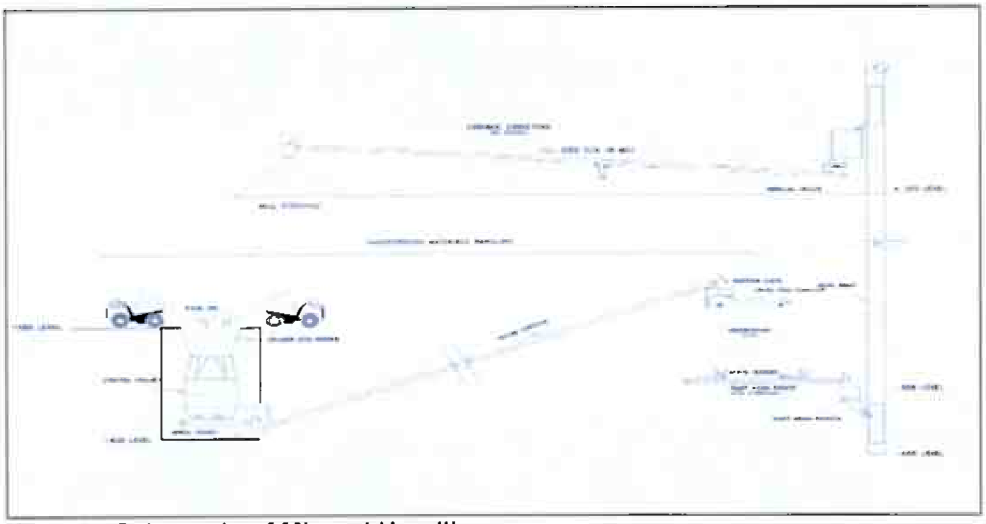


Figure 7 Schematic of Mineral Handling

A general arrangement of the shaft bottom area is shown in Figure 5 below.

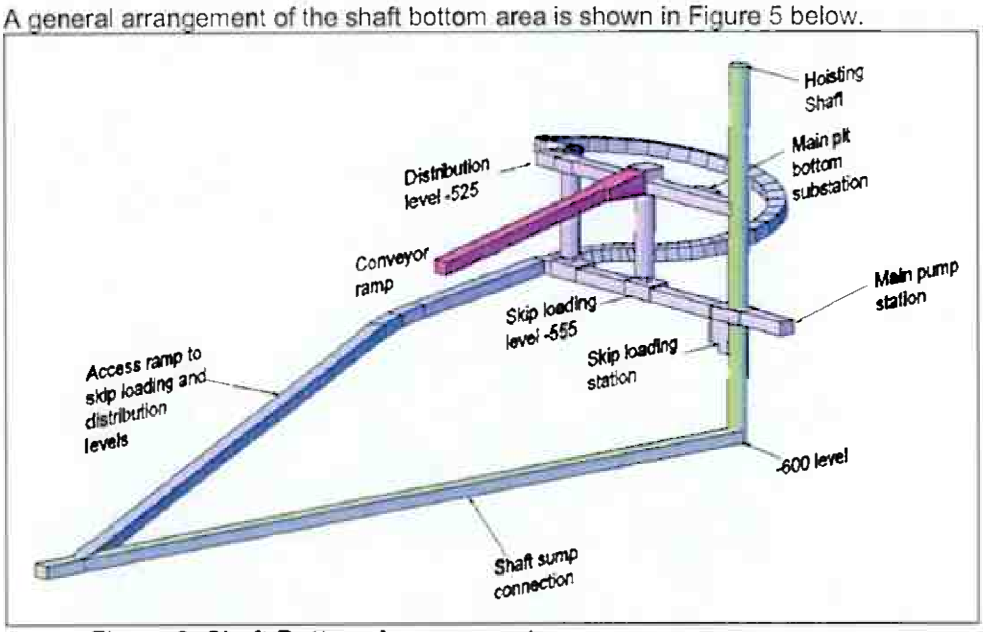


Figure 8: Shaft Bottom Arrangement

4.8.1 Crusher Chamber

The required throughput for the crusher is 20.000 tonnes per day.
Based on an utilisation of 18 hours per day and an availability of 90%, the installation must be sized for a throughput of 1300 tph.

A typical cone crusher for this duty will have a feed opening of 1300 mm (54") and a cone diameter of 1900 mm (75").

Additional equipment to be provided will include a rock breaker for oversize material and an overhead travelling crane for installation and maintenance. Other features will include a dust extraction system. Assuming that the crusher will be fed by 50 t dump trucks, the design must be able to take 26 truckloads per hour, i.e. 1 load per 2.3 minutes. In order to achieve this, a double-sided crusher chamber tip arrangement should be provided.

A typical general arrangement of the crusher chamber is shown in Figure 4.

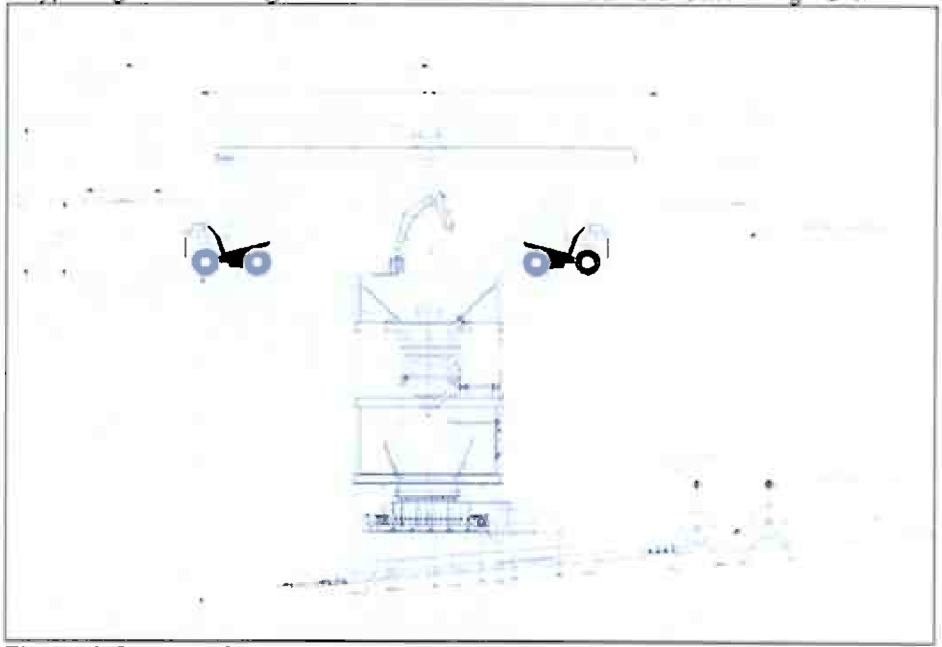


Figure 9 Crusher General Arrangement

4.8.2 Crusher Chamber Access Ramps

In order to construct the crusher chamber access ramps will be required to access the roof and the base (40 m has been estimated for the up ramp and 120 m for the down ramp). The roof of the chamber would be fully excavated and supported before the body of the chamber is created.
The down ramp will also access the Belt Change station and Tramp metal removal discharge point.

4.8.3 Conveyor Drift

A conveyor ramp will transport the crushed rock into the shaft surge silos.

The conveyor will convey crushed ore from the crusher bottom at –600 m to the top of the silos at approximately –525 m level, a lift height of 75 m. Assuming an inclination of 1:6, the conveyor will be some 450 m long.

4.8.4 Silo Distribution Level

Assuming a stand-off distance of 15 m between the shaft and silos and 15 m separation between silos, the Silo Distribution level will be some 50 m long.

4.8.5 Shaft side Storage Silos

On the basis that the shaft will operate on a 20 hour hoisting capability, and the crusher only 18 hours per day storage capacity must be provided. Clearly the greater the storage the better, however for practical purposes a total of 1000 t has been assumed.

Typically a deflector and conveyor will feed the rock into one or other of the two silos. The silos will be sized to store 500 t of ore discharging at the –555 m level and be equipped with out feeders discharging onto a belt which will in turn charge the skip measuring flasks.

4.8.6 Feed Level to the Measuring Flasks

Rock from the silos will be fed onto a conveyor which will transfer the rock into a pair of measuring flasks prior to discharge into the skips. This level will be some 50 m long.
An apron feeder or similar will be required to feed ore onto the belt to the shaft.

4.8.7 Main Conveyor to the Shaft Silos

The main conveyor below the crusher should have the capability of dealing with surges in feed and typically will consist of 1400 mm wide belt with a 600 kW drive motor rated at 1500 tph. The belt should be equipped with all the protection devices to control slack belt, belt rip, overload, overheat etc. Incorporated with the belt there will be a belt change station and an overhead magnet and belt to remove tramp iron.

An apron feeder or similar will be required to feed ore onto the main belt.

4.8.8 Shaft sump connection

The shaft sump connection drive will be driven from the crusher access drift to the shaft sump, a distance of approximately 450 m. This will allow spillage clearance from the shaft and to accommodate pumping requirements in the shaft sump.

4.8.9 Vehicle access to the skip loading level and silo distribution level.

A ramp will be driven from the shaft bottom driveage to connect with the skip loading level and silo distribution level. This will allow maintenance access without interfering with shaft operation. The ramp will be some 450 m long.

4.8.10 Underground Workshops

Due to large quantity of mobile mining equipment involved in the operation and the depth of the mine it is likely that vehicle maintenance and refuelling will be carried out in the workshops.
Provision should be made for 3 workshops serving each of the main mining blocks. Each workshop will have facilities for daily maintenance, repair, wheel change and refuelling. Major overhauls will be carried out on surface or offsite as required. A typical workshop layout is shown in Figure 6.

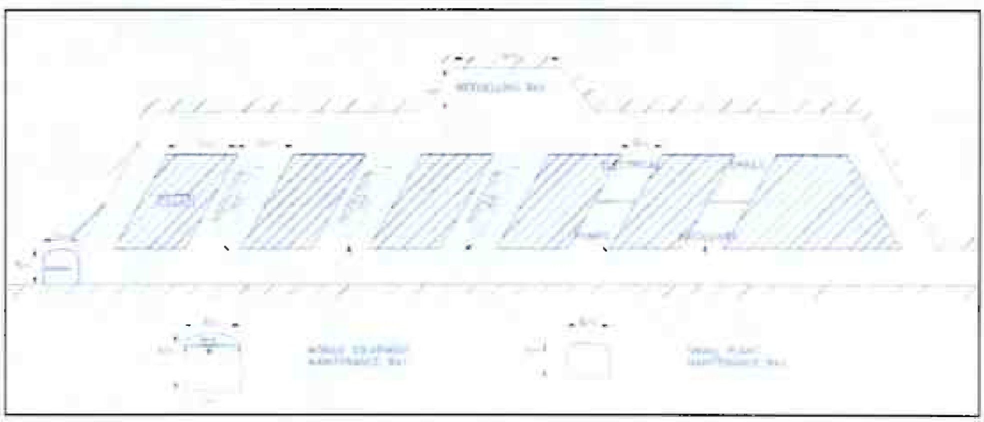


Figure 10 Underground Workshops

4.9 Water management

No information is currently available regarding expected water inflow. A hydrological assessment of the host rock and ore body will be required at the next level of study in order to derive an anticipated inflow figure for mine dewatering.

In the absence of any data an average life of mine inflow figure of 100 l/sec has been assumed, which together with a process water usage of say 20 l/sec gives a pumping requirement of 120 l/sec.
The basic layout for water management will consist of water settlement on the -580 m level, a decant system to a clear water tank on the –600 m level, low pressure pumping to the main pump station on the –525 m level and high pressure pumping up the shaft to surface.

Further surface treatment may be required, but is outside the scope of this report. A schematic is shown in Figure 7.

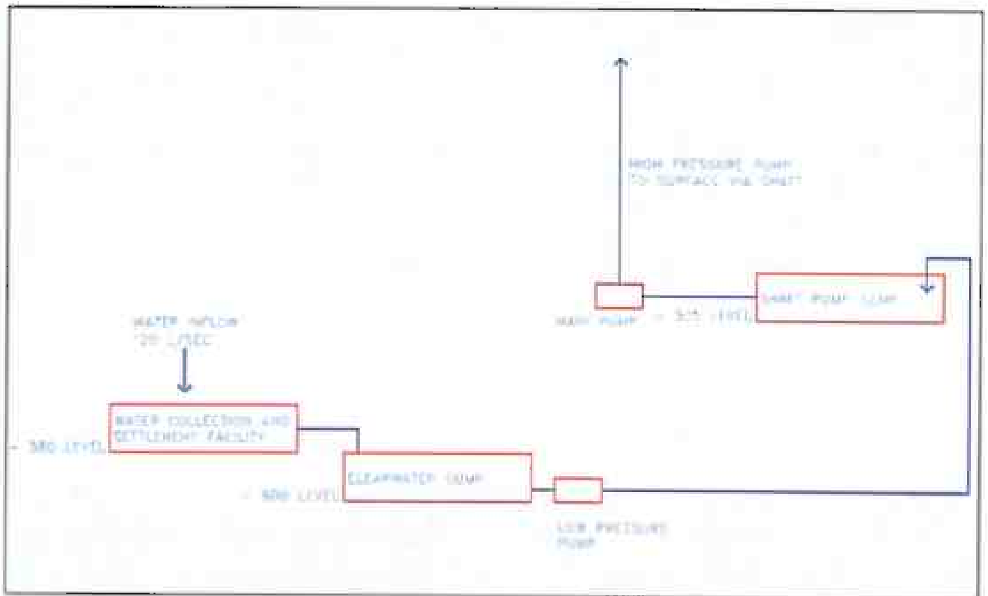


Figure 11 Pumping Schematic

4.10 Main Ventilation Fans

In order to properly specify the main ventilation fans a full ventilation network analysis is required but at this stage of project development this is beyond the scope of this level of study.

Typically the ventilation design will investigate the effect of the mining method on air circulation and contaminants and can be expected to fulfil the following criteria throughout the mine workings:

Air velocity:	0.5 m/s minimum at all places where personnel work
Dust removal:	95% compliance with 0.1mg of respirable quartz
Gas dilution:	Carbon monoxide (CO) [1]TWA= 30 ppm [2]STEL=200 ppm
	Carbon dioxide (CO_2) TWA= 5000 ppm STEL= 15000 ppm
	Nitric oxide (NO) TWA= 25 ppm STEL= 125 ppm
	Nitrous oxide (NO_2) TWA= 3 ppm STEL= 5 ppm
	Sulphur dioxide (SO_2) TWA= 2 ppm STEL= 5 ppm
Diesel fumes	A minimum air dilution rate of 0.04 m^3/s per kW of rated installed power and this must be at the point of use

The ventilation concept is based on a twin fresh air intakes (shaft and ramp) with return air being exhausted from the mine via surface raises. This may be modified in the next level of ventilation engineering.

[1] TWA is Time Waited Average
[2] STEL is Short Term Exposure Limit³

The conceptual mine layout includes for 3 exhaust ventilation raises, and based on studies of mines with comparable mining methods, equipment, and production rates, a total ventilation requirement of the order of 690m^3/sec could be anticipated.

This can be supplied by three 1000 kW axial flow fans located at the top of each surface ventilation raise.

4.11 Electrical Distribution Scheme

The design of the electrical distribution scheme in any major mine must be based on total security of supply. A review is required of the electrical capacity currently available in the Hurdal area and, necessary arrangements may need to be negotiated with the power supplier to provide sufficient capacity if not available. For the purposes of the study it is assumed that adequate electrical capacity is available in the locality.

Regardless of the security of supply, locally generated standby capacity must be provided such that in the event of a major power failure, operation of the mine fans and pumping system can be maintained. Further standby power arrangements must be made to operate the man riding facility in the shaft.

The typical distribution network will comprise the following main points of supply:

- Mine Winder
- Underground rock handling
- Mine developments in each of the 3 main blocks
- Underground pumping
- Mine ventilation

Each of the main activity areas will be equipped with a substation linked in a ring main system to allow for modifications to the system, while maintaining continuity of supply.

A mine of Hurdal's size typically will have a total connected load of 20 MW, with the major individual power users being the Mine Hoist (5 MW), Main Mine Fans (3 MW) and the Main Pumps (1.5 MW).

4.12 CAPEX ESTIMATE

CAPEX costs have been estimated for the main mine infrastructure using internal database information and major equipment prices obtained for recent feasibility studies.

CAPEX costs have been estimated for the main mine infrastructure using internal database information and major equipment prices obtained for recent feasibility studies.
The development costs shown in the following table have been used:

Table 7

Capital Development per metre	$2,600
Shaft sinking and equipping/metre	$22,750
Enlargement/m^3	$156

All costs are based on an exchange rate of 1.3 US$/Euro.

The accuracy level of the cost estimate is considered to be +/-50%.

4.12.1 Shaft Site Investigation costs

Once the shaft site has been identified, a detailed site investigation programme will be required to determine design parameters, geological and hydrological conditions for the shaft infrastructure.

The following allowance has been included for site investigation associated with the shaft site.

Table 8

Hydrological investigation	$325,000
Shaft centre line borehole	$416,000
Borehole testing and evaluation	$65,000
Shaft Site investigation Total	**$806,000**

4.12.2 Shaft sinking costs

The following costs have been estimated for the shaft sinking activities:

Table 9

Shaft design	$195,000
Shaft site clearance and levelling	$260,000
Shaft substation	$104,000
Construct shaft collar and fore shaft (30 m)	$520,000
Sink shaft 830 m @ $22,750/m	$18,882,500
Stations at 175 m, 90 m – 25 m and -380 m	$130,000
Loading pocket and pit bottom	$65,000
Total for shaft sinking	$20,156,500

4.12.3 Shaft Equipment

Shaft equipment costs are based on database and recent supplier information obtained for a similar project.

Table 10

Tower and foundations	$3,250,000
Winding engine and shaft furnishing, ropes, etc., including recommended spares	$12,350,000
Total for shaft installation	**$15,600,000**

4.12.4 Minerals Handling system

Mineral handling equipment costs are based on database and recent supplier information obtained for a similar project.

Table 11

Crusher chamber	$2,210,000
Crusher access drifts	$487,500
Conveyor drift	$1,170,000
Silo distribution level	$130,000
Silos	$780,000
Feed level to shaft skips	$130,000
Cone crusher – supply and install	$4,030,000
Conveyors and feeders	$2,080,000
Total Underground Mineral Handling	**$11,017,500**

Other Infrastructure

Estimates in this section cover the other main infrastructure items together with additional development costs not included elsewhere.

Table 12

Shaft bottom driveage	$1,170,000
Shaft access ramp	$1,170,000
Underground workshops excavation	$2,600,000
Underground workshops equipment	$650,000
Water management excavation	$520,000
Pumps and pipes	$1,625,000
Main Ventilation Fans	$1,950,000
Underground electrical distribution	$9,100,000
Communications & SCADA system	$780,000
Total Other Underground Infrastructure	**$19,565,000**

4.12.6 INFRASTRUCTURE CAPEX

The individual CAPEX elements are summarised below:

Table 13: CAPEX Summary for Underground Infrastructure

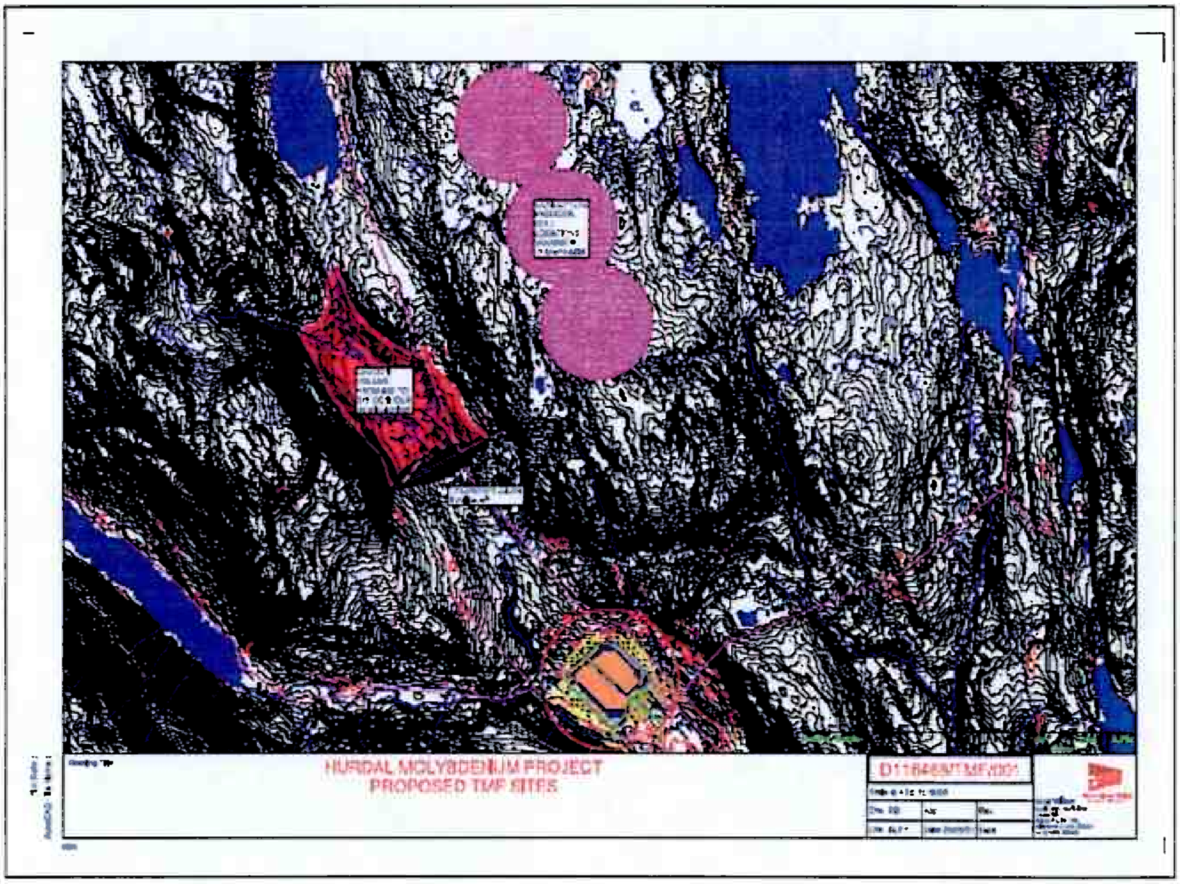
Shaft Site investigation	$806,000
Shaft sinking and installations	$35,756,500
Underground Mineral Handling	$11,017,500
Other Underground Infrastructure	$19,565,000
Total CAPEX	**$67,145,000**

5. Recommendations for further work

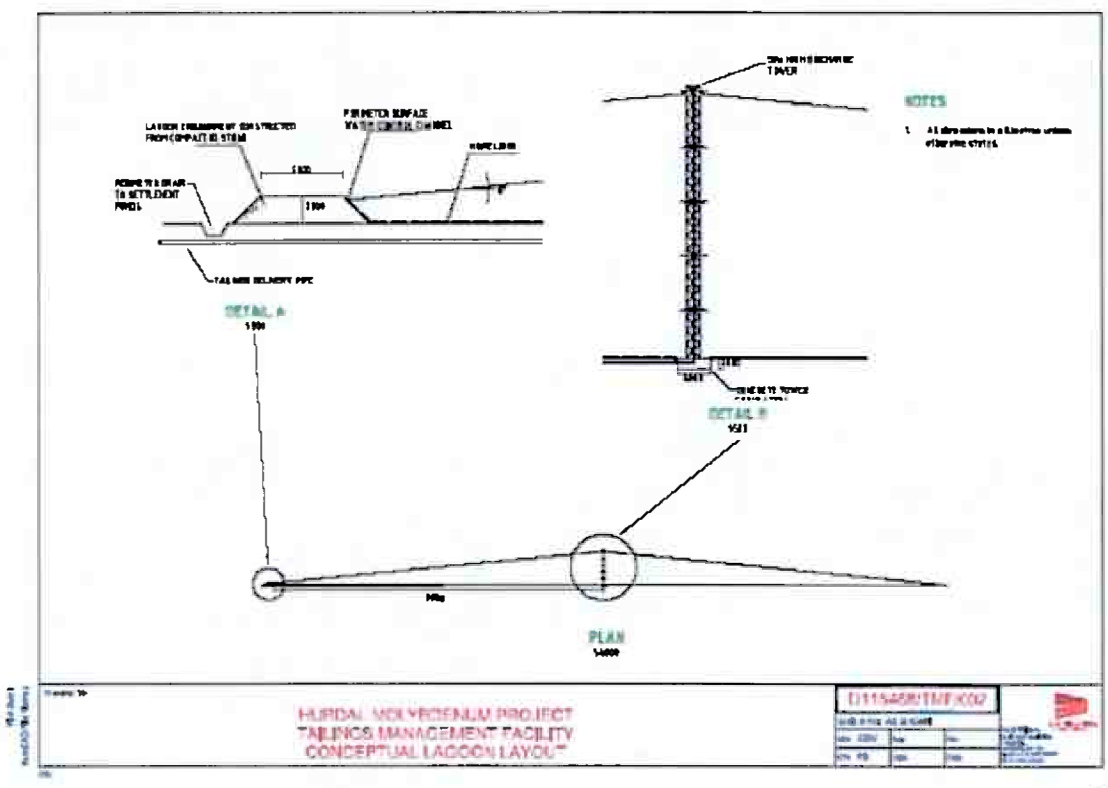
To enable the project to progress to the next stage of project development a key and fundamental issue to be determined is the location of the main project infrastructure. To properly assess the project location and reach some more definitive site selection, it is considered essential to carry out a series of further engineering and environmental studies as listed below;

- Detailed on-site topographical surveys to determine suitability of proposed tailings management facility sites, process plant and shaft site;
- Assessment of affected properties, and likely compensation costs.
- Site investigations (boreholes, trial pits, lab testing etc) to determine;
  - Geotechnical properties of materials to be used for and affecting construction (paste tailings, waste rock, overburden, shaft sinking etc.)
  - Quantities of overburden to be stripped at tailings management facility and plant sites;
  - Hydrology and hydrogeology of potential sites, including assessment of watercourses;
  - Meteorological studies (rainfall, snowfall, winds etc.);
- Determination of Norwegian environmental legislation applicable to mine and tailings management facility construction;
- A consultation process with stakeholders, initially government, to assess the accessibility of placing the mine and associated infrastructure in this area. It would be useful to use this opportunity to discuss possible locations for tailings disposal, including hill tops. A lack of acceptance of the project concept at an early stage could prove to be a show-stopper; and
- Ecological studies;
- Likely requirements for restoration of TMF sites.
- Once a clearer idea is obtained of how the project is to be progressed the opportunity to engage with other stakeholders, such as local communities, tourism organisations and regional authorities, should be pursued. This will need to be carefully timed and managed, as it appears that the Project will potentially require resettlement and will impact significantly on the existing landscape and land use.

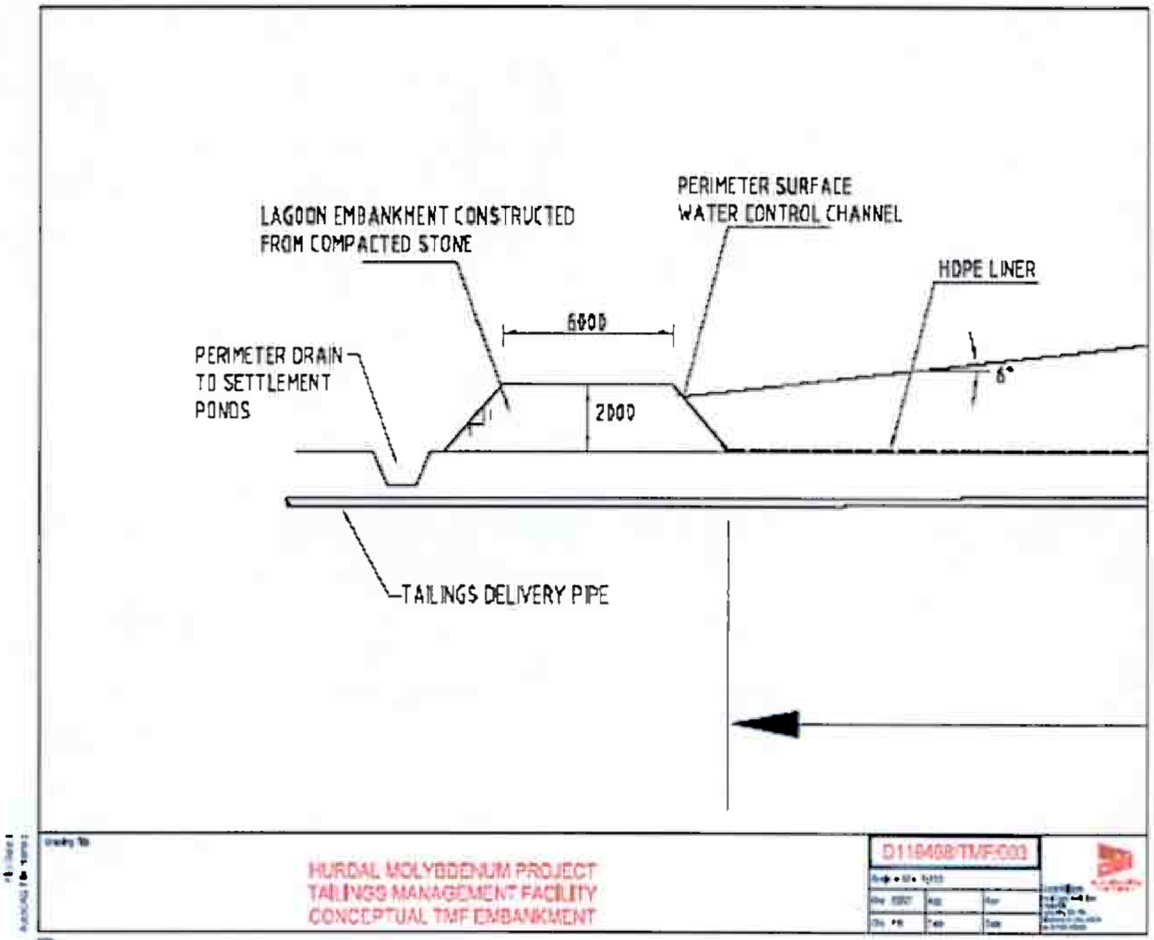
APPENDIX 1
DRG NO. D116468/TMF/001
PROPOSED TMF SITES



APPENDIX 2
D116468/TMF/002
PADDOCK TMF CONCEPTUAL LAYOUT



APPENDIX 3
D116468/TMF/003
CONCEPTUAL TMF EMBANKMENT



5.0 PROCESS PLANT DESIGN

5.1 Summary

No metallurgical data exists for Hurdal. Therefore it has been assumed that the Hurdal ore is a typical porphyry molybdenite deposit and a generic flotation plant has been defined for benchmark costing purposes. It has been assumed that the only floatable mineral in the ore is molybdenite and that the copper, lead and talc content is negligible.

The plant is a typical molybdenite flotation plant to produce a dried 50%Mo molybdenite concentrate in bulk bags. Roasting has not been included. However, it is understood that currently molybdenite roasters are operating at high capacity so customer's roaster capacity should be considered. It may be advantageous to consider the inclusion of a multiple hearth moly roaster. Crew's marketing consultants should advise on this.

The Plant is designed to treat 8,000,000 tpa of ore at a feed grade of around 0.1% MoS_2. For consistency with previous study work the recovery has been assumed to be 84% so the plant will produce nominally 1 t/h of concentrate. However the plant design and costing is insensitive to increases in feed grade. The tailings are thickened to paste at the plant and the paste pumped to the TMF in a valley about 4km to the northwest of the plant during the open pit phase and to either the open pit or, with cement addition, to underground as backfill during the underground mining phase. The estimated operating cost is 3.7 US$/t milled and the capital cost US$ Million 203. Both capital and operating costs have been derived from other projects by adjustment to the scope and factorisation.

A plant footprint has been developed by modification to the layout for another project of a similar processing tonnage and no plant engineering has been done. The footprint has been added to the overall site plan to the southeast of the open pit convenient for the shaft location currently envisaged. There is a stream crossing the plant site, but it is envisaged that this can be accommodated by modifications to the plant layout.

Consideration has been given to the effects on the plant should only the project be modified to have the underground mine only. In that case the plant capacity would decrease to around 6.1 Million tpa and it is envisaged that the shaft would be relocated to the north of the mine. In that case there is the opportunity to relocate the plant to the north of the mine and, because there will be no waste dumps, move the TMF further south. This would reduce the operating cost by a small amount and the indicative capital cost of the plant is expected to reduce to of the order of US$ million 170.

The 8 Mtpa plant will require about 6,000 m^3/day of water from an external source such as the lakes or rivers. The total installed power is estimated to be of the order of 36 MW.

5.2 Plant Design Criteria

The following have been assumed for the plant costed.

Plant throughput	t/annum	8,000,000
Plant feed grade (Nominal)	%Mo	0.06
	$\%MoS_2$	0.10
	%Cu	Negligible
	Pyrite	Negligible
	Talc	Negligible
Primary crusher throughput	t/h	1,235
Primary crusher operating time		360 days/annum. 75% availability
Crushed ore stockpile capacity	t	21,000t live, 88,000t total
Milling operating time		365 days/annum. 91% availability
Milling throughput	t/h	1,000
Ball Mill Work Index	kWh/t	14
Ore Abrasion Index		0.6
SAG Mill motor power	MW	10
Ball Mill motor power	MW	10
Flotation feed particle size	P_{80}	80% - 120µm
Flotation feed pulp density	%w/w	35%
Flotation Cells		
Rougher + Scavenger		7 off 160m^3 tank cells
Cleaner 1		4 off 10m^3 tank cells
Cleaner Scavenger		6 off 10m^3 tank cells
Cleaners 2, 3 and 4		2.5m diameter flotation columns
Regrind Mill		250kW Tower Mill on combined rougher and cleaner scav. conc.
Regrind size	P_{80}	80% -30µm
Concentrate Thickener		4m diameter High Rate.
Concentrate Filter		1 t/h Vacuum Drum or Disc Filter
Concentrate Dryer		1 t/h Diesel fired Holoflite dryer
Molybdenite Concentrate Moisture	%	4% moisture
Molybdenite Concentrate dispatch		1 t bulk bags
Final Flotation Concentrate grade	%Mo	Min 50% Mo, <0.5%Cu.
Molybdenum Recovery to concentrate	%	84
Molybdenite Concentrate production	t/day	21 nominal
Tailings Disposal		Paste to TMF or Open Pit
Tailings Thickeners		3 off 20m dia. Deep Cone Thickeners
Tailings Pulp Density to TMF	%w/w	70%
Tailings Pulp Density on TMF	%w/w	80%
Plant water makeup requirement	m3/day	6,000 (from mine or external source)

5.3 Plant Description

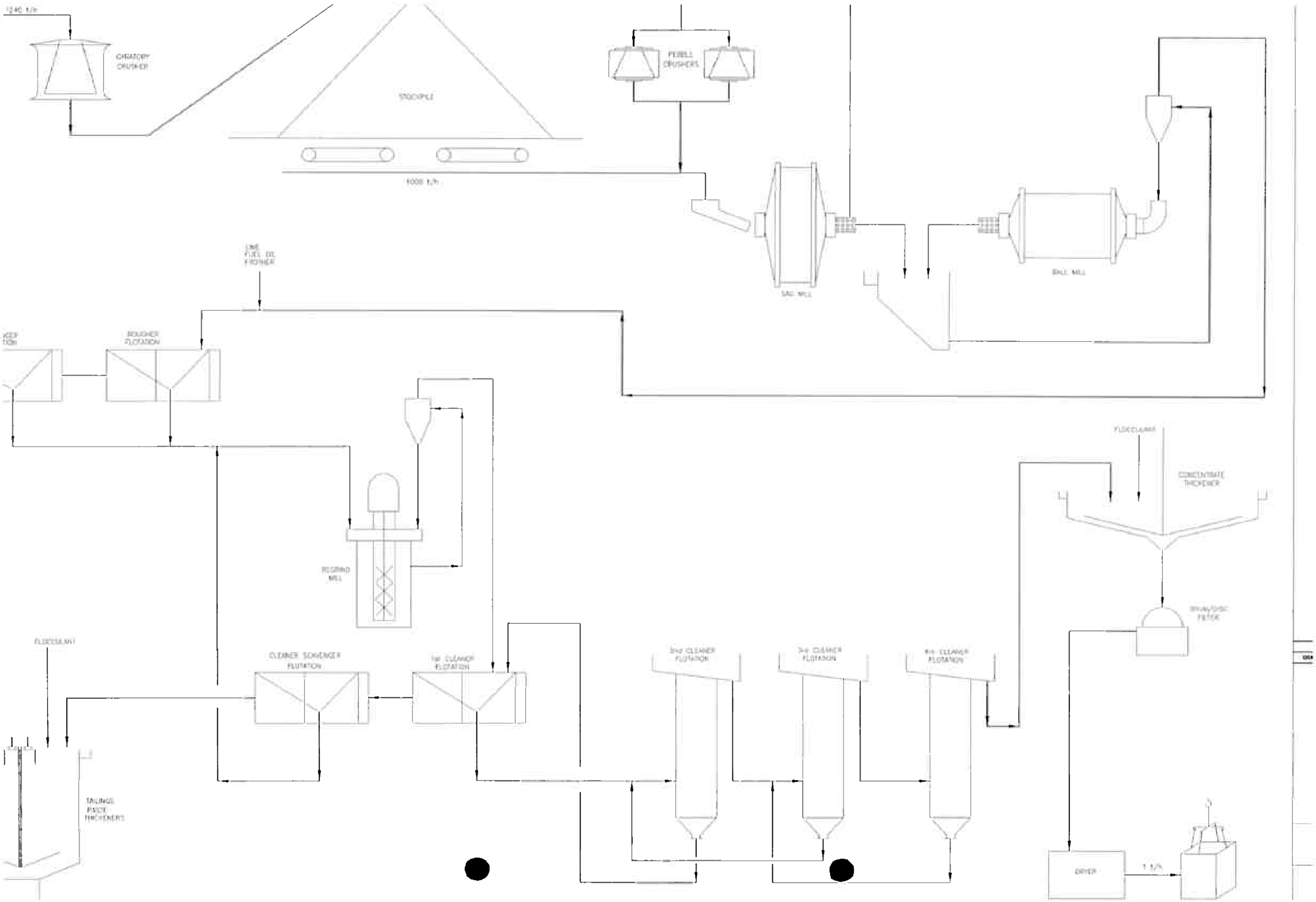
Please refer to the conceptual flowsheet.

The Run of Mine ore from open pit is crushed to about 180 – 250mm by a gyratory crusher at the plant. Crushing is envisaged to be performed on a 24h/day basis, 360 days per year. The crushed ore is conveyed from the crusher to a covered stockpile with a live capacity of about 21,000t from which it is fed to the milling circuit by two apron feeders. When the mining moves underground, the ore is crushed underground and hoisted to the surface and conveyed to the same stockpile. The open pit crusher can then be used to crush low grade stockpiled ore from the open pit stripping which, if economic to treat, could be used to supplement the plant feed because the underground mine produces at a maximum of 6,000,000 tpa and the plant capacity is 8,000,000 tpa.

The remainder of the plant would operate for 365 days per year with an availability of at least 91%. It has been assumed that the ore is moderately hard and abrasive. The milling circuit comprises a 10MW SAG Mill with a single pebble crushers and a 10MW Ball Mill. Generally the grind size for Molybdenum ores is fairly coarse and a grind size of about 80%-120 microns has been assumed. There may be an opportunity to reduce operating costs by using High Pressure Rolls instead to the SAG mill but this is ore specific and cannot be considered at this stage.

The grinding circuit cyclone overflow is pumped to rougher/scavenger flotation cells comprising 7 large (160m^3) tank cells. The combined rougher/scavenger concentrate is reground in a tower mill. The reground concentrate is cleaned initially in four 10m^3 first cleaner cells with the cleaner tailings processed by six 10m^3 scavenger cleaner cells. Molybdenite circuits differ from porphyry copper circuits in that, because of the much lower head grade, the concentrate processing equipment is much smaller although more cleaning stages are required to make a saleable product. Sometimes regrinding is done in stages within the cleaning circuit to maximise recovery while minimising overgrinding of the flaky molybdenite mineral. For this plant no further regrinding has been included, but 3 stages of cleaning in flotation columns has been included, to give 4 stages of cleaning overall. With a mill feed rate of 1000 t/h, only about 1 t/h of molybdenite concentrate is produced at nominally 50% MoS_2. This molybdenite concentrate is thickened in a small (4m) thickener. The thickener underflow is filtered on a vacuum disc or drum filter. This filter cake is dried, typically using a diesel or gas fired Holoflite dryer to produce a concentrate of about 4 – 5% moisture. This concentrate, bagged in bulk bags, is the final product of the plant and is sold typically FOB truck at the plant site.

The tailings at nearly 1000t/h, is thickened to a paste consistency with deep cone thickeners. During the open pit phase, all this tailings is pumped to a Tailings Management Facility (TMF) located in a valley about 4km to the north west of the plant. This area is also about 195m higher than the plant site so the paste tailings pumping duty is high. So installing tailings thickeners close to the TMF was considered, with tails thickened to about 60% solids being pumped from the plant. However the extra cost of the future paste thickening required at the plant for the underground phase, the cost of the multiple stage centrifugal pumping required plus the piping to return thickener overflow back to the plant was high and single stage paste pumping was more economic. For this study single stage pumping of the paste from the plant has been selected, but the pressure is approaching the limit for single stage paste pumping so optimisation of this would be needed. Once underground mining commences tailings is required underground as paste backfill with the surplus sent to the open pit. Therefore a cement storage and addition facility will need to be added at the plant in about year 5 -6. The tailings thickeners and paste pumps will then provide for both underground backfill and disposal of surplus tails to the open pit. It has been assumed that no water needs to be returned from the TMF to the plant because of the paste disposal. 1,700 m^3/h of water will be returned to the plant from the tailings thickeners. In addition to this recycled water it is expected that the plant will require roughly 6,000m^3/h of water makeup. This could come from mine drainage water, rainfall or from an external source. The area contains numerous lakes and rivers and it is assumed that a source is found close to the plant. No costs have been included for purchase of water.



GYRATORY CRUSHER
STOCKPILE
PEBBLE CRUSHERS
SAG MILL
BALL MILL
LIME
FUEL OIL
FROTHER
ROUGHER FLOTATION
REGRIND MILL
CLEANER SCAVENGER FLOTATION
1st CLEANER FLOTATION
2nd CLEANER FLOTATION
3rd CLEANER FLOTATION
4th CLEANER FLOTATION
FLOCCULANT
CONCENTRATE THICKENER
DRUM/DISC FILTER
DRYER
FLOCCULANT
TAILINGS PASTE THICKENERS

5.4 Plant Operating Costs

Operating costs have been developed by using typical consumption data and recent prices from other projects. The overall operating cost for the 8 million tonne per annum plant has been estimated at €2.7/tonne milled including a 10% contingency, but excluding overall G & A costs. A Dollar to Euro exchange rate of $1.36=Euro has been used in the operating costs

5.4.1 Summary of Plant and Infrastructure Operating Costs

	g/t	Cost '000 €/year	€/tonne	US$/tonne
Power				
Power Comminution		2,746	0.34	0.47
Power Other		1,038	0.13	0.18
Total Power		**3,784**	**0.47**	**0.64**
Reagents				
Fuel Oil Collector	300	529	0.07	0.09
Fuel Oil Emulsifier	20	400	0.05	0.07
Lime	260	169	0.02	0.03
MIBC - Frother	15	328	0.04	0.06
Dowfroth 250 - Frother	13	255	0.03	0.04
Flocculent	21	528	0.07	0.09
Dryer Diesel Fuel		39	0.005	0.01
Analysis Consumables		105	0.013	0.02
Total Reagents		**2,354**	**0.294**	**0.40**
Comminution Consumables				
Primary Crusher Liners		193	0.02	0.03
SAG Balls	240	1,368	0.17	0.23
Ball Mill Balls	560	3,102	0.39	0.53
Regrind Balls		50	0.01	0.01
SAG Liners		657	0.08	0.11
Ball Mill Liners		318	0.04	0.05
Regrind Liners		25	0.003	0.00
Pebble Crusher Liners		436	0.05	0.07
Total		**6,150**	**0.77**	**1.05**
Maintenance Spares		**2,155**	**0.27**	**0.37**
Misc Consumables				
Mobile Equipment Fuel		128	0.02	0.02
H&S plus G&A Consumables		120	0.02	0.02
Total		**248**	**0.03**	**0.04**
Labour				
Operating		2,502	0.31	0.43
Maintenance		1,144	0.14	0.19
Admin		1,611	0.20	0.27
Total		**5,257**	**0.66**	**0.89**
Contingency		**2,220**	**0.28**	**0.38**
Total Plant & Infrastructure		**22,170**	**2.80**	**3.80**

5.4.2 Plant Labour

The following have been included. Unit rates have been assumed for Norwegian staff. The manning structure has been derived from other projects adjusted for the Hurdal plant scope.

Direct Plant Staff

Description	Total	Cost to Employ €/year	Cost €/year
General - Management, Technical & Clerical:			
Plant Superintendant	1	99,694	99,694
General Mill Forman	1	74,771	74,771
Senior Plant Metallurgist	1	79,755	79,755
Plant Metallurgist	1	55,829	55,829
Metallurgical Technicians	1	31,122	31,122
ISA Optimisers (Met. Technician)	1	31,122	31,122
Technical Clerks & Secretaries	2	26,081	52,163
Concentrator - Day shift Workers:			
Reagent Preparation Operators	1	31,807	31,807
Concentrate Loadout FEL driver	1	27,036	27,036
General Operator	1	31,807	31,807
General Labourers	3	26,081	78,244
Concentrate Outloading Operators	1	31,807	31,807
Concentrator - Shift Operation Workers:			
Shift Foreman	4	48,550	194,201
Control Room Operator	4	39,582	158,327
Open Pit Crushing Station Operator	4	39,582	158,327
Mill Operator Circuit	4	39,582	158,327
Regrind & Flotation Operator	4	39,582	158,327
Concentrate Dewatering Operator	4	39,582	158,327
Tailings Dewatering Operator.	4	39,582	158,327
General Operator	4	39,582	158,327
Labourers	8	33,384	267,074
Gate House	4	33,384	133,537
TMF Maintenance:			
Foreman	1	43,695	43,695
General Labourers	5	26,081	130,407
M & E Maintenance			
Plant Maintenance Manager	1	78,220	78,220
Chief Mechanical Engineering	1	74,771	74,771
Chief Electrical Engineer	1	74,771	74,771
Chief Instrument Technician & Process Control Engineer	1	71,780	71,780
Instrument Technicians	1	35,272	35,272
IT Technicians	1	30,263	30,263
Fitters/Mechanics	2	35,272	70,543
Electricians	2	35,272	70,543
Apprentice Fitters/Electricians	2	27,512	55,024
Shift Mechanical Fitter	4	41,268	165,071
Shift Electrician/Inst. Tech	4	41,268	165,071
Overall Plant Total	**85**		**3,393,686**

Plant General Administration

Description	Number	Cost to Employ €/year	Cost €/year
General - Administration Mine & Surface Works			
Commercial Manager	1	78,220	78,220
Chief Accountant & Payroll Officer	1	62,576	62,576
Accounts & Wages Clerks	2	26,506	53,011
Chief Procurement Officer	1	62,576	62,576
Buying Clerks	1	26,506	26,506
Concentrate Sales & Shipping Manager	1	62,576	62,576
Chief Chemist	1	74,771	74,771
Analysts	8	31,122	248,976
Laboratory Assistants	4	39,582	158,327
Sales & Shipping Clerks	1	26,506	26,506
HR & Training Manager	1	62,576	62,576
Training Officer	1	43,803	43,803
Personnel & Training Clerks	1	28,690	26,506
Office Receptionist	1	31,298	28,690
General Typists & Secretaries	1	26,081	26,081
Cleaners & Janitors	3	23,473	70,420
Safety & Medical Staff			
Senior Safety Officer	1	62,576	62,576
Safety Engineers	1	43,803	43,803
Enviromental Officer	1	62,576	62,576
Enviromental Technicians	1	35,272	35,272
Medical Room Orderly/First Aider	1	27,512	27,512
Shift Workers			
Changehouse Attendant	4	33,384	133,537
Mobile Security Officer	4	33,384	133,537
M & E Maintenance (Concentrator Only):			
Storekeeper	1	48,550	48,550
Storeman	2	31,807	63,613
Vehicle Maintenance Manager	1	42,326	42,326
Vehicle Maintenance Filters	2	35,272	70,543
Auto Electricians	1	27,512	27,512
Overall Plant Total	49		1,863,477

5.4.3 Plant Power

A 2007 power cost of €0.02/kWh for a Norwegian power intense industry supplied with hydroelectric power was obtained from the Internet. The power consumptions were taken from another project, adjusted for the scope of the Hurdal plant.

Area Description	Installed kW	Absorbed MWh p.a.	Cost '000 €/year	Cost €/tonne
Ore Crushing & Conveying	1,907	10,220	204	0.03
Primary Grinding Circuit	20,908	124,298	2,486	0.31
Pebble Crushing Circuit	759	2,789	56	0.01
Flotation, Reagents & Conc Dewatering	2,098	12,862	257	0.03
Tailings Thickening	978	3,468	69	0.01
Compressed Air Services	957	4,751	95	0.01
Water Services	1,474	7,691	154	0.02
Tailings Disposal	700	3,764	75	0.01
General Surface Infrastructure	650	2,182	44	0.01
Total	30,643	172,024	3,440	0.43

5.4.4 Plant Consumables

A steel ball consumption typical of moderately hard ores has been used from a number of benchmark operations. Liner costs were based on other projects for which vendor recommended lives were obtained together with recent prices.

Moybdenite ores normally require only fuel oil as collector and either one or two frothers. A fuel oil emulsifier has been included, but this may not be required if the fuel oil is added to the mill. Likewise lime has been included for pH control, but if other sulphides are totally absent it may be possible to operate at natural pH thereby eliminating the lime. Flocculant has been included for the tailings, concentrate and also a provision for Mine Water treatment.

5.4.5 Maintenance Spares

Maintenance spares have been estimated at 5% p.a. of the direct mechanical equipment capital cost.

5.5 Plant Capital Cost Estimate

The capital costs have been estimated by factorisation from other recent projects, with the exception of the paste pumps for which a potential pump selection and an order of cost was obtained by discussion with Putzmeister. The estimate is order of cost.

Costs are included for the usual plant infrastructure including:
- Analytical and paste tails testing laboratories.
- Workshops.
- Store.
- Offices.
- Change house.
- Canteen.
- Gatehouse and security fencing.
- First Aid facility.
- Training.
- Plant mobile equipment and refuelling facility.

The estimated capital cost for the 8 Mtpa Process Plant and Plant Infrastructure is US$ 203 million.

6.0 COSTING AND PRELIMINARY FINANCIAL MODEL

The capital and operating costs for the project components were estimated for the various components as outlined below. The costs have been compiled into a single spreadsheet in which is also the financial model.

6.1 Open Pit Costs.

The costs were estimated by Edgar Urbaez year by year over the life of the Open Pit with Year 1 defined as the plant start up year. Costs start in year -3 with purchase of the mining fleet, construction of mine facilities such as offices, workshops etc., construction of haul roads and electrical facilities and the engineering costs of developing the Open Pit.

Operating costs for waste and/or ore mining commence from Year -2 and continue to Year 7 or 8.

For using these costs for the financial model, the open pit capital cost was adjusted by removal of the costs for management and for working capital. The management cost was included in the overall costs under G&A incurred in one year at the start of open pit development, although probably this would be spread over both the development years. All working capital has been excluded from the model as it examines only the cash flow assuming 100% equity funding.

An adjustment was made to the operating costs for waste mining to adjust for tramming distance. These included waste tramming for 0.5km and the waste dump is so large that tramming distances of up to 2km with an average of 1.5km is expected. An allowance of US$0.12 per t.km was added to the waste mining costs to allow for this. No adjustment to the mine fleet capital has been made although this may increase somewhat for longer distances.

The waste mining cost per tonne also included US$ 0.25/t for waste dump rehabilitation. This has been stripped out of the waste mining operating cost and the amount removed inserted over the last years of the open pit operation as it will not be possible to rehabilitate much of the dump in the early years. Thus this cost becomes deferred which helps the project economics.

6.2 Underground Mining Costs

The costs were estimated by Diogo Caupers year by year over the life of the Underground Mine with Year 1 defined as the year in which substantial amounts of ore are mined from underground and therefore in which the Underground Mine takes over from the Open Pit in feeding the plant. The underground mine commences development 6 years before this. A further major expenditure is scheduled in the last year of development for underground mining equipment. Replacement capital is scheduled over the life of the Underground Mine.

For using these costs for the financial model, the cost of shaft sinking and the excavation and equipping of the crushing, conveying and water handling facilities have been deleted because these costs have been included by Scott Wilson Mining.

6.3 Underground Mine Infrastructure

The capital costs were estimated by Scott Wilson Mining (SWM) for:

- Shaft site investigations
- Shaft sinking

- Shaft Equipment
- Minerals Handling system
- Miscellaneous Underground infrastructure

These costs have been scheduled in the financial model to be incurred in the first year of Underground Mine development.

For using these costs for the financial model, the cost of shaft site investigations have been deleted because these are included in Crew's Owners Costs.

6.4 Tailings Management Facility

The capital costs were estimated by Scott Wilson Mining (SWM) for:

- Valley impoundment or as paddock impoundments. The valley option has been included as the cheaper option.
- River diversion around the Open Pit either as a tunnel or channel. The channel option has been included as the cheaper option.

The costs of the TMF construction have been scheduled as incurred in the year before plant start up. It has been assumed that all the rock required is sourced from the open pit stripping. The river diversion costs have been scheduled to occur in the same year as the purchase of the Open Pit mining fleet, the year before Open Pit development commences.

6.5 Environmental and Social

This covers the future use of the Open Pit after closure. No costs or revenues from this have been included in the model as they are insufficiently developed and also too far in the future to radically affect the NPV.

6.6 Plant Costs

The Plant capital and operating costs have been included. To enable the operating costs to reflect the reduction in tonnage processed in the Underground phase, these were split into fixed and variable components on the Plant sheet of the financial model workbook. The labour costs were taken as fixed with all other costs variable.

The plant capital cost has been split over the 3 years prior to plant start up as shown on the workbook to indicate an appropriate spending scenario.

6.7 Owners Costs and G&A

Owner's costs were provided by Crew to cover:

- Exploration drilling
- Geotech drilling
- BFS
- Land acquisition.

These have been scheduled to take place over a 3 year period prior to purchase of the Open Pit equipment.

G&A costs have been provided as:

- A US$3.8 million lump sum for the open Pit development from Edgar Urbaez's report, scheduled for the year in which the Open Pit equipment is purchased.

- A fixed annual sum of US$ 3.0 million for each year from plant start up. This is purely a provisional sum. It is intended to cover corporate overheads, marketing travel, insurances, local taxes, public relations and the like.

6.8 Overall Cost Schedule and Financial Model

6.8.1 Model

The costs have been compiled year by year into a spreadsheet for each of the Options studied. The tonnage and grades mined and plant recoveries are included so that a revenue stream can be calculated for each option. The price basis is clarified on each model sheet to avoid confusion between costs per tonne metal and costs per tonne MoS_2. The metal costs in US$/lb Mo metal contained is used as the prime figure. Roasting, concentrate transport and the conversion loss have been allowed for as variables. The figures are generic figures by reference to other projects. The roasting charge in particular is not small and is apparently a function of the Mo price and hence supply and demand. As Mo prices increase, the roasting charge can also increase. This needs to be checked by Crew when performing a detailed market study. It is also observed from the literature that there is a shortage of roasting capacity at present and an assessment of the likelihood of this persisting in the future should be considered.

For each Option, the cash flow as Total Revenue - (Capex+Opex) has been calculated and the NPV and IRR of the cash flows calculated. The model is pre tax and assumes no loans and hence no interest payments. No royalties have been included. If these are known, they could be added into the spare rows of the Owners Costs.

Pertinent data have been extracted from the model of each option and collated onto a single page. The overall results have been extracted onto the overall summary page.

6.8.2 Options

The Options considered were:

		Mo $/lb
OPTION 1	325m pit, COG 0.061% MoS2	30
OPTION 2	275m pit, COG 0.061% MoS2	30
OPTION 3	325m pit, COG 0.091% MoS2	22
OPTION 4	275m pit, COG 0.091% MoS2	22
OPTION 2a	275m pit, COG 0.061% MoS2, No open Pit	30

The Options 1 – 4 correspond to those generated by Diogo Caupers with two sizes of pit based on different cut off grades (COG) and two Molybdenum contained metal prices. In all cases no low grade (waste) has been fed into the plant and therefore the milling rate drops once underground mining starts. The model allows for the addition of waste to maintain an 8Mtpa plant throughput, although no capital or operating costs are added for the reclaim trucks. When run with waste treatment, the NPV is not significantly changed.

Option 2a is a variant of Option 2 with Open Pit mining deleted. This entails:

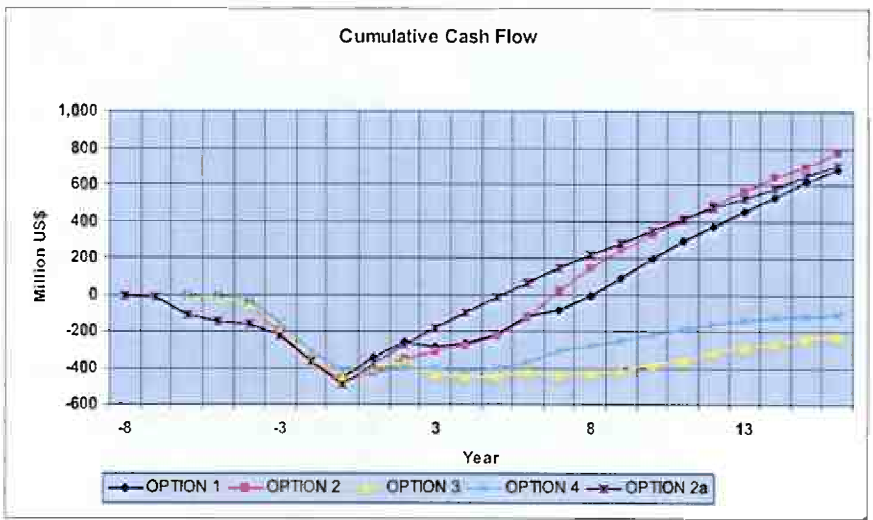
- 2 years more delay between commencement of the project and plant start up.
- Mining of higher grades from start up.
- No open pit or waste dumps required, which may facilitate permitting.
- More compact site.
- Smaller plant (6.1 Mtpa).

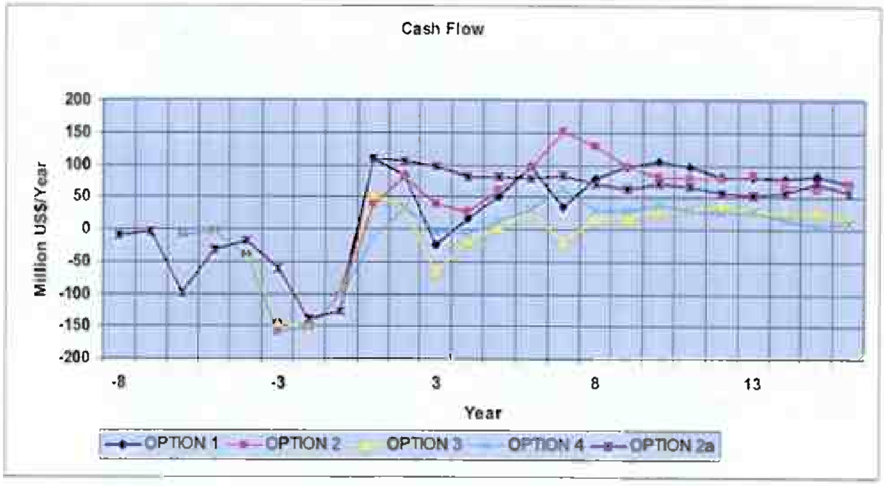
6.8.3 Results

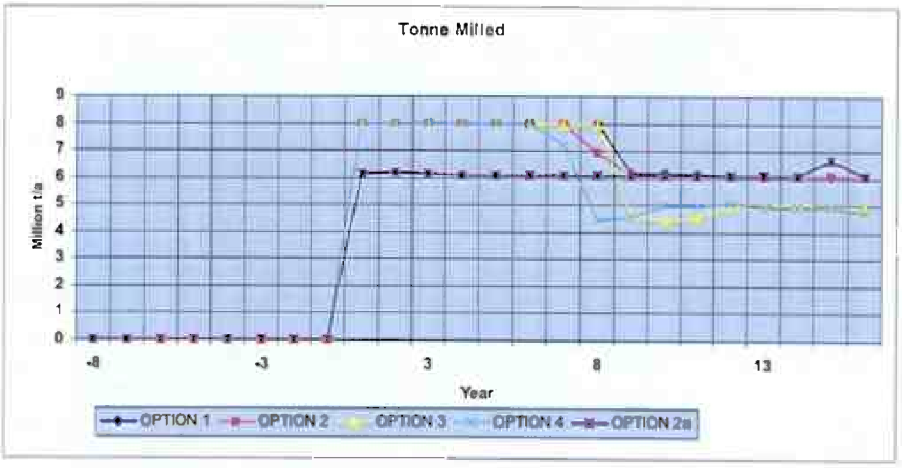
(NPV of Cash Flow over Project Life)		NPV $MM	@Rate %	IRR %	Life Years Project	Life Years Mill	Mo $/lb
OPTION 1	325m pit, COG 0.061% MoS2	471	5%	11.9%	41	35	30
OPTION 2	275m pit, COG 0.061% MoS2	515	5%	12.4%	42	36	30
OPTION 3	325m pit, COG 0.091% MoS2	-235	5%		39	33	22
OPTION 4	275m pit, COG 0.091% MoS2	-204	5%		39	33	22
OPTION 2a	275m pit, COG 0.061% MoS2, No open Pit	385	5%	12.0%	38	30	30

Assumed Offsite Costs for ALL OPTIONS

Roasting	4.0	$/lb Mo
Concentrate Transport	0.40	$/lb Mo
Conversion loss	1.0%	Mo







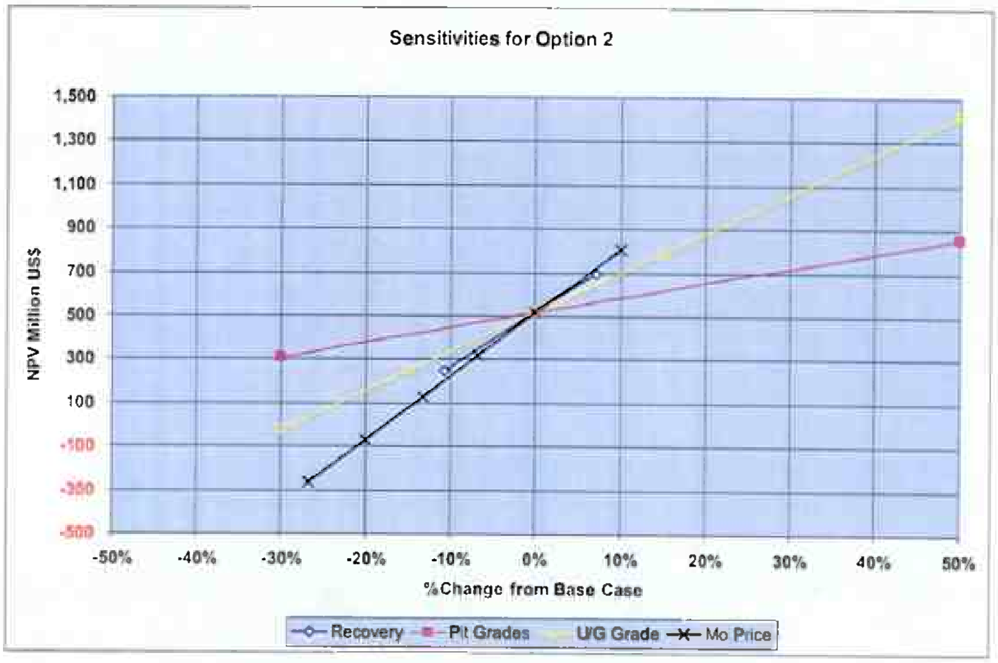
The best result is Option 2 due mainly to the high Mo price.

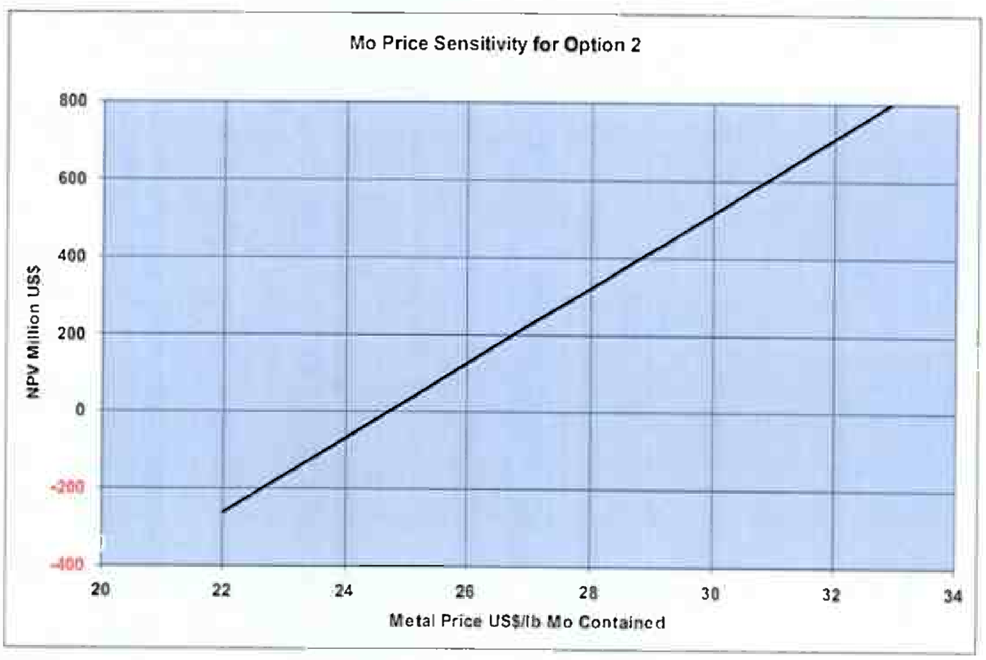
6.8.4 Sensitivities

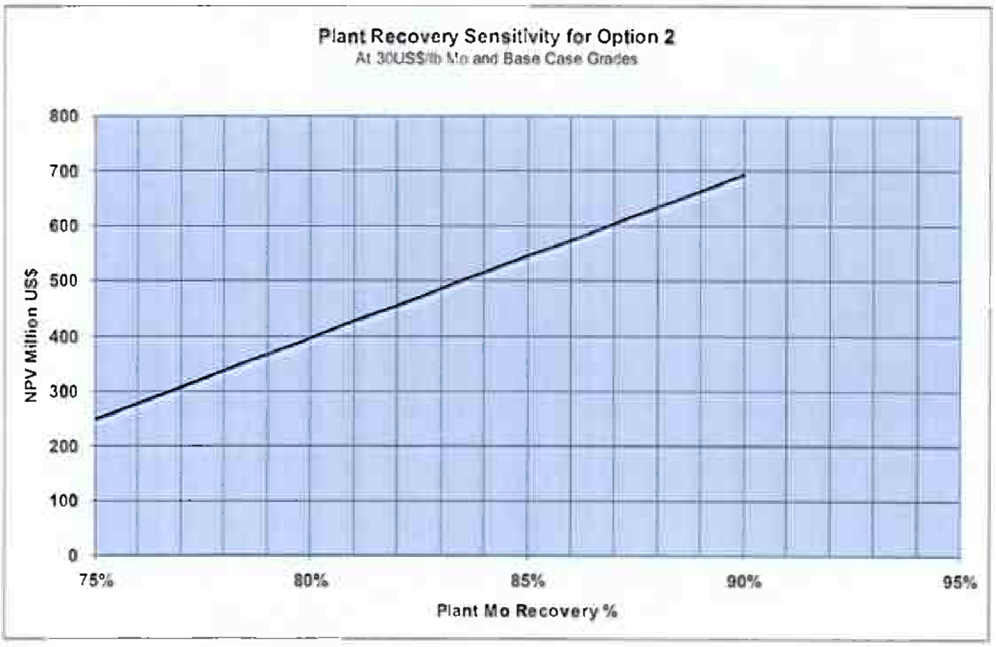
A series of sensitivities were run for the best option, Option 2. The results are shown graphically below.

The model is most sensitive to Mo price and plant recovery and least to Open Pit grades. The effect of Mo price is also shown against actual Mo metal price. The break even Mo price is about US$25/t with the present model.

An increase in plant recovery from 84% to say 90% would increase the NPV by about US$177 million at 30 US$/lb Mo price.


Sensitivities for Option 2
NPV Million US$
1,500
1,300
1,100
900
700
500
300
100
-100
-300
-500
-50%
-40%
-30%
-20%
-10%
0%
10%
20%
30%
40%
50%
%Change from Base Case
Recovery
Pit Grades
U/G Grade
Mo Price


Mo Price Sensitivity for Option 2
NPV Million US$
800
600
400
200
0
-200
-400
20
22
24
26
28
30
32
34
Metal Price US$/lb Mo Contained


Plant Recovery Sensitivity for Option 2
At 30US$/lb Mo and Base Case Grades
NPV Million US$
800
700
600
500
400
300
200
100
0
75%
80%
85%
90%
95%
Plant Mo Recovery %

7.0 CONCLUSIONS AND RECOMMENDATIONS

Based on the available information and the data generated during the course of this scoping study the following conclusions and recommendations for the Hurdal Project have been generated by the study team:

1. There is an urgent need for consistency when referring to the grades of molybdenum in the data available for the project. Currently many of the study references refer to MoS_2 and there is confusion between this and Mo, which can lead to significant under or over reporting of grades. It is recommended that all future reference should be to Mo only.

2. There is a need for much more mineralogical and assay data on the ore, than is currently available to the study team. Specifically this is required to evaluate likely penalties and / or credits from other elements in a concentrate which would be produced from this deposit. Currently there is no mention of copper as an associated element in the ore, if present in sufficient quantity; it could provide an economic benefit to the project.

3. Currently drill spacing is very wide; it is recommended that infill drilling is carried out as soon as possible. This will not only add significantly to the knowledge of the deposit, but will also give much greater confidence in the assay data and will delineate ore and waste with much greater confidence.

4. Currently the resource calculation is very preliminary; however, infill drilling results will almost certainly alter the resource calculations significantly and has the potential to increase the amount and grade of the ore.

5. Intuitively the obvious way to exploit a deposit such as Hurdal is to concentrate on development of an open pit in the early years of production while developing an underground mine with production transferred underground once the pit reaches its economic mining limit . However preliminary findings based on the current data indicates that the opposite may be true in this case. Hurdal will have a high strip ratio for the open pit which will result in significant waste dumps close to the mine and hence local communities, as such this may make permitting more difficult. In addition the current information indicates that the better ore grades would only be accessible by an underground mine.

6. The open pit development would generate approximately 100Mt of waste which would have to be stored in large dumps close to the mine. In addition development of the pit will require diversion of a stream outside the mine site to allow access to the ore. The open pit ore is generally quite low, with a grade of >0.1% MoS_2 being found at levels below 200m from the surface.

7. It is felt that environmental permitting for an open pit may be quite difficult as there will be significant visual impact from the pit itself and the associated waste dumps, together with the dust and noise an open pit operation would produce to affect the surrounding inhabited area.

8. Currently there appears to be little environmental data available. It is recommended that environmental baseline work should commence as soon as possible to ensure that as much data as possible is obtained to support a permitting application and to establish whether there are any significant areas of environmental concern which may require attention or mitigation prior to submission of a permit application.

9. Although the initial conclusions suggest that an open pit may not be the preferred option, there are several advantages of an open pit which should be considered. Firstly development of a pit will allow exploitation of a greater part of the Hurdal resource than by underground mine alone. Secondly there is a possibility for future reuse of the pit as a dump for municipal waste from the Oslo conurbation which is relatively close to the site; this in turn could generate a future revenue stream. The re-use of the open pit as a waste dump does offer a significant opportunity but this does need to be examined in more detail to confirm its viability.

10. The underground mine has been designed using backfill as an integral part of the mine method to maximise ore extraction and reduce the volume of tailings to be disposed of on surface thereby minimising the environmental impact of the operation

11. Underground operational development will generate "waste" from approximately 64 km of tunnels that will be located just outside the stope design cut off grade. Re-evaluation of the resource model following the infill drilling may almost certainly prove that this rock could be classified as marginal ore. Currently waste is classified as material which is <0.019% MoS_2 which has been used in the study as the marginal mining cutoff grade.

12. The underground design is based on the assumption that there are FAIR to GOOD ground conditions which will be reasonably consistent throughout the ore body and will allow development of stopes 12m wide x 25m high. Geotechnical information is required to support these basic assumptions.

13. At an underground stope design cut-off grade of 0.091% MoS_2 there will be approximately 100Mt of ore with a grade of 0.12% MoS_2. A further exercise should be undertaken to establish what would happen to the mine plan and economics if the cut off grade was set at 0.12% MoS_2. When the infill drilling is completed, a new resource model should be generated. To optimise Net Present Value, it may be worth while re-evaluating the underground production with a fixed mine life of, say 15 years, to establish a new higher stope design cut off grade.

14. Four mining options were considered and evaluated economically:

Pitshell	US$ Mo	Cut Off %MoS_2	Preliminary Economical Evaluation
325m	30	0.61	Positive NPV
275m	30	0.61	Positive NPV-Best Option
325m	22	0.91	Negative NPV – Worst Option
275	22	0.91	Negative NPV

he returns are sensitive particularly to Molybdenum price and plant recovery. With the current model a price of US$25/lb contained Mo metal is required to achieve even a zero NPV. This is high in historical terms although much lower than the current price. There have been two historical high price spikes in the Mo price. In both cases these were followed by a slump to prolonged periods of low prices. Hurdal will be vulnerable to such volatility unless the ore contains another value such as copper.

15. Currently the TMF location is proposed as a valley to the NW of the site. At the next stage of study it will be worth investigating moving the TMF embankment downstream closer to the

mine site , where the valley is wider, thereby reducing the overall embankment height which would increase the capacity/embankment ratio and so reduce the construction costs.

16. The TMF will contain approximately the same quantity of tailings (ie 65Mt) irrespective of whether the deposit is exploited by a combination of open pit and underground or only as an underground mine . The current strategy is that the open pit is mined first and the tailings from open pit ore would go to the TMF, and that when the underground mine is mined the balance of the tailings (not placed as backfill) would go to the old open pit for disposal.

17. The TMF will need approximately 6Mt of material for the embankment, some or all will have to be from a quarry. It is currently proposed that most of this could come from the underground mine waste development, however if the development waste is reclassified as ore in the future then a quarry for the full 6Mt will be required in the area.

18. If there was no open pit in the project, a major reassessment of the land take requirements should be undertaken as it is very likely that a much more compact and cheaper site will result with the processing plant and possibly the mine shaft being located to the north of the orebody in the open pit waste dump position, close to the TMF. This in turn would significantly reduce pumping costs and would have less impact on the current habitation as it will be further away form the majority of the existing housing.

19. Current land take is 600 hectares, a reasonable cost would be $10-15,000 per hectare for non developed land. Currently we have an allowance of $80M to cover land purchase, relocation of housing, roads and power lines. This is probably valid if an open pit option is considered, however if an underground mine only is considered it may be very high.

20. Currently the processing plant is sized at 8Mtpy throughput. This was sized on the open pit production rate, however it now looks as if it will be difficult to maintain this rate for underground which is likely to be closer to 6Mtpy, therefore there will be excess capacity of 2Mtpy on the plant. This level of excess capacity does not seem to be acceptable from economic or operational reasons therefore consideration should be given to installation of a plant with a throughput of 6Mtpy to treat both ore types.

21. If a 6Mtpy plant was installed it may be possible to maintain the higher open pit mining production level of 8Mtpy from the open pit, with 6Mtpy of "high grade" processed each year and 2Mtpy low grade ore being stockpiled which could be processed at a later date should prices improve to allow its processing economically. This would give the opportunity for "high grading" the open pit to a more economic grade.

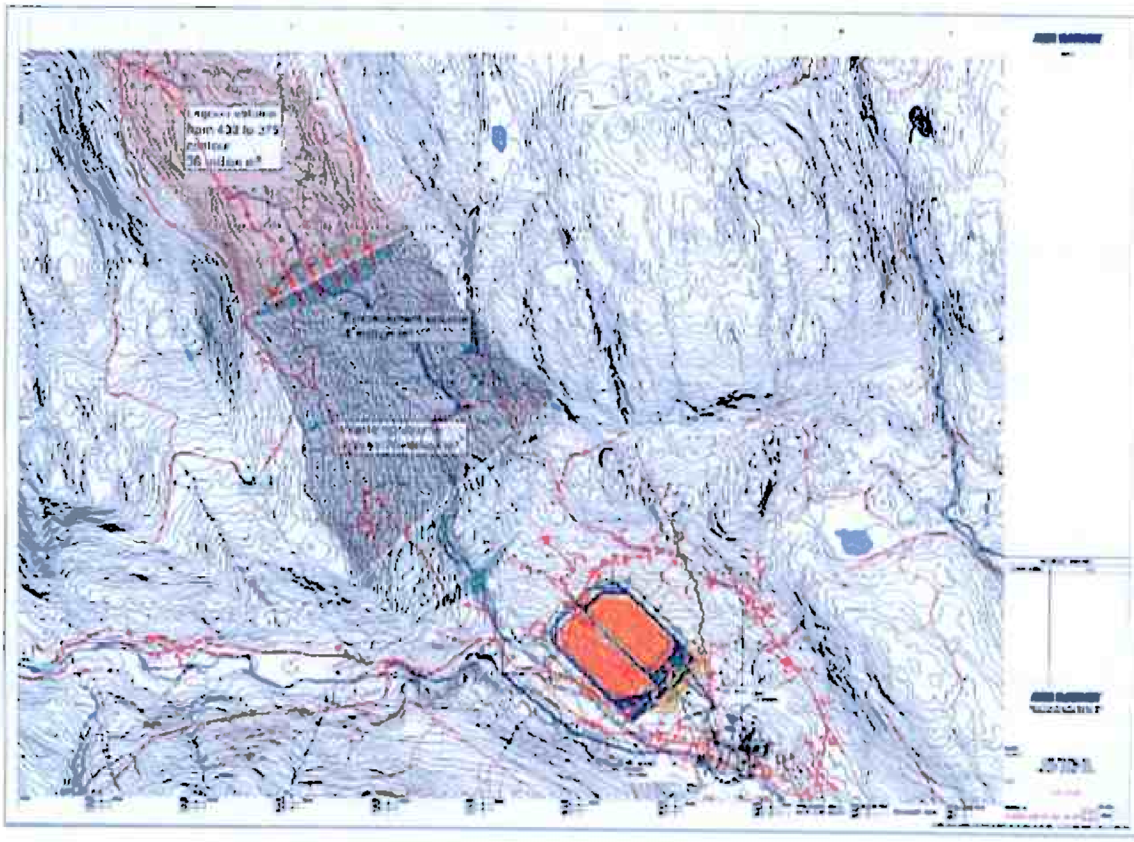
22. Current operating costs have used a power cost of $0.03/kwh. This needs to confirmed and re-evaluated in the future. Likewise a figure for sales costs of $6.00/ lb Mo was proposed by SRK, we have changed this to $4 /lb Mo for roasting, £0.4 /lb Mo for concentrate transport and 1% for conversion loss, these factors are used in the preliminary financial model in this Scoping Study and will need future investigation and justification.

23. It is recommended that metallurgical testwork is carried out as soon as possible on representative samples of ore to confirm basic ore processing characteristics. This can be done at the same time as the infilling drilling to produce sufficient core for bench scale or mini pilot plant testing.

24. Overall, the preliminary financial calculations carried out as part of this study suggest that the project is worthy of progressing to a further stage of development. This should be done by addressing many of the items and recommendations given in this report which will generate additional information on which future investment decisions could be made.

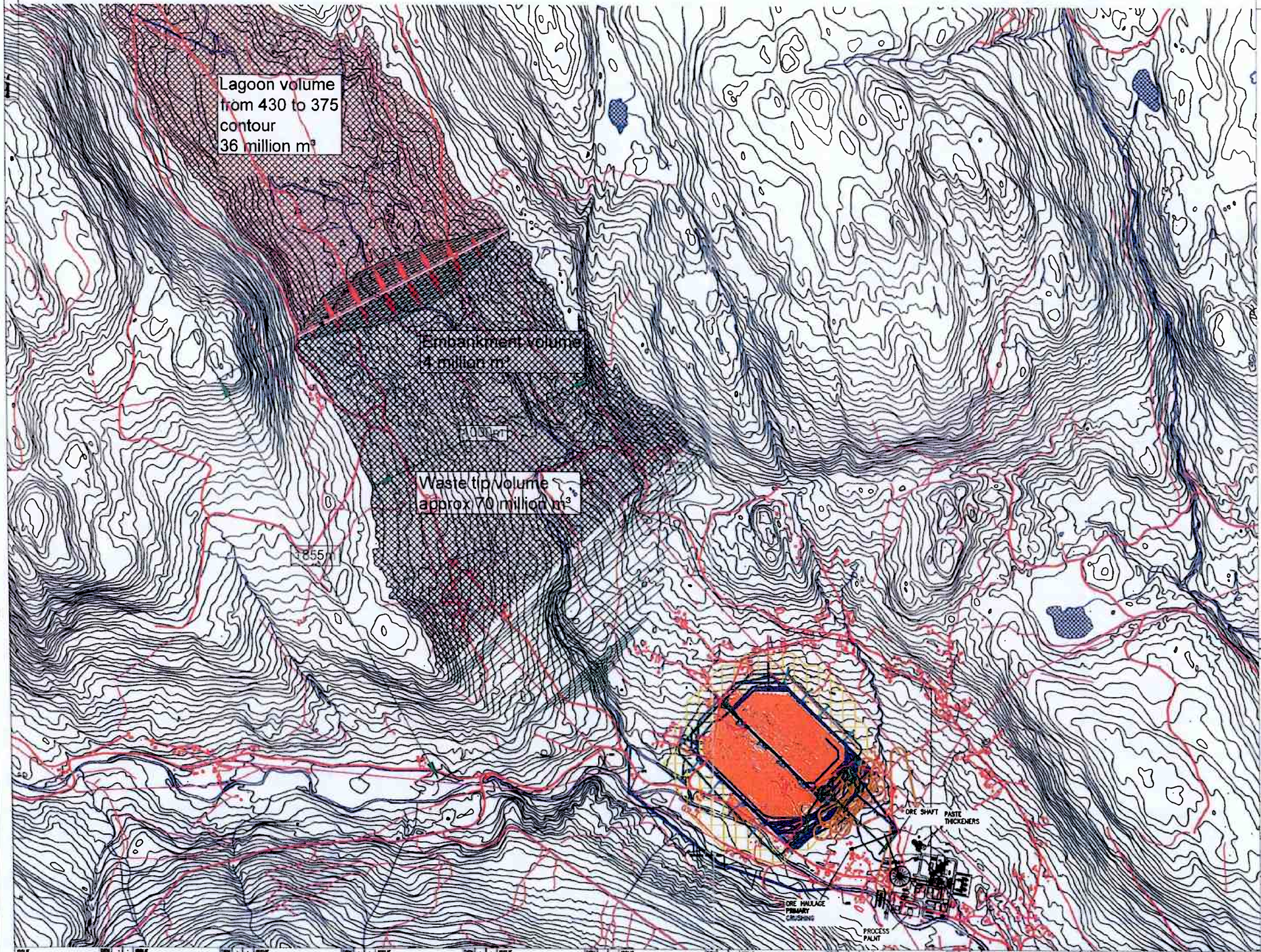
APPENDIX 1

OVERALL PROJECT SITE PLAN



APPENDIX 2

PRELIMINARY FINANCIAL MODEL



AKER KVAERNER

NOTES

REFERENCE DRAWINGS

DRAWING NUMBER	TITLE

AKER KVAERNER

PROJECT:
CREW MINERALS ASA
HURDAL MOLYBDENUM
PROJECT SCOPING STUDY

TITLE:
SITE PLAN

DRAWING No. 81060X04-000-00-150-LAY-001

P2 PRELIMINARY ISSUE WASTE DUMP ADDED

P1 PRELIMINARY ISSUE