Lappland Goldminers AB

An Introduction to Lappland Goldminers AB

Lappland Goldminers AB is a Swedish prospecting company with gold projects in Sweden and gold projects and a gold mine in Finland. The company was founded in 1997 with the focus on prospecting on the so called Gold Line in the province of Lappland in Northern Sweden. The Gold Line is a gold province, a gold province, which was discovered in the late 1980's and 1990's. The area is located southwest of the well known Skellefte District, which has seen 30 base metal and gold mines in operation since the 1920's. The Gold Line is very under prospected compared to the Skellefte district. One gold mine has opened so far on the Gold Line, the Svartliden mine and it is operated by Dragon Mining, Perth, Western Australia.

Already in 1998 it was decided that the very fragmented ownership structure of the different prospects on the Gold Line was a hindrance for effective development of the region. Since then the company has worked on getting control of the key projects in the area so that effective and economical plans for the area could be worked out. Today the Company's Swedish projects are 100% owned by the Company with no royalty contracts existing.

In 2001 the strategy of the company was reformulated and it was decided that the Company should become a mining company. At that time it was clear that the company had one large low grade deposit in Fäboliden and that there existed smaller, higher grade deposits on the Gold Line as well as a number of good exploration targets. The strategy then became to build one central processing plant at Fäboliden where ore from the whole Gold Line could be processed to gold ingots. Fäboliden has a central location on the Gold Line with access to good infrastructure; roads, power supply, railway, two airports and available personnel from three community centres in the area. The organisation of the company is reflected in this ambition and today the company is staffed like a mining company with a combined experience from leading positions in nine mines and two smelters.

This feasibility study concerns the development of the mine and mill at Fäboliden. The mill is designed to process 4.6 million tonnes of ore/year and the mine will start as an open pit and then go underground. The operating cash flow will be around 500 million Swedish Krona (80 million US $) per year the first 8 years of production and will then gradually decrease as mining goes deeper. Management will however aim for increasing cash flow over time with deliveries of higher grade ore from other mines on the Gold Line. The known ore reserve at Fäboliden as of today is sufficient for 12 years production. Drilling at Fäboliden is however ongoing which will increase reserves and mine life.

Test mining at Fäboliden in 2005 produced a nice surprise. The grade of the ore was estimated to be close to 1.5 gram Au/ton but turned out to be 3.4 gram Au. The company then carried out an extensive grade verification program, including two new trenches over the deposit and a number of large blast holes were drilled and analyzed. These tests also indicated considerably higher grades than results based on assays from diamond drill holes. The hanging wall was then cleared of overburden and channel samples over 20 meters width were taken. The grade of these samples was 1.0 gram Au instead of the expected 0.2 gram Au. After that a number of bigger tests of the diamond drill cores have been made both from the ore body and from the gold mineralised rock on the hanging wall. All these tests seem to confirm the assumption that the true grade is higher than the grade interpreted from diamond drilling. There is 33 million tonnes of gold mineralized material in the open pit that has the potential of becoming marginal ore. The tests did raise the question whether the marginal material really is marginal or if parts of it are low grade ore. The testing done has this far not been sufficient to give a scientific explanation for the higher grades found in bigger samples but management is of the belief that the true grade is not reflected in the drill cores which all calculations in this feasibility are based on. Extensive test mining could test this assumption but then access is needed to a nearby mill. So far such an arrangement hasn't been possible to make.

The processing plant has a technical and economical life which far exceeds 25 years. The company plans for a very long-lasting operation. The tailings area is sufficient for a yearly production of 5 million tonnes of sand for 50 years. With some additional work it can be made to last for another 50 years. When it comes to permitting, the tailings dam is the most difficult part of a mining operation to get a permit for. This tailings dam is designed to hold tailings from all the future goldmines of the Gold Line, even though permits are only granted for 15 years at a time.
With a processing plant and tailings dam in place it will be comparably easy to open up other mines in the region, which will then be served by the Fäboliden facility.

The Company is today prospecting in 14 areas on the Gold Line. The main projects are Stortjärnhobben, Knaften, Gubbträsk and Tjålmträsk. In January 2008 the company was granted an exploitation concession for Stortjärnhobben. In Knaften the biggest gold anomaly in bottom till in Sweden is found. In the Tjålmträsk area the Mother Lode of the richest boulder on the Gold Line so far has not been found, containing 350 gram Au/ton. In Gubbträsk, both gold and base metal mineralizations have been discovered.

In 2007 the company expanded into Finland through the acquisition of Northern Lion Gold Corporation's exploration portfolio there. The main deposit is the Haveri deposit in Southern Finland. The deposit has an historic resource of 742.000 ounces from 1999 and has been drilled with very promising results since that time by Northern Lion Gold. The Haveri deposit is a very interesting gold, copper and cobalt project and the strategy for this project is similar to the Gold Line strategy in Sweden i.e. to establish a central processing plant in Haveri which can serve that deposit but also neighbouring deposits. In March 2008 Lappland Goldminers acquired the Pahtavaara mine and mill operation from the receiver of the Scan Mining bankruptcy. The Pahtavaara mine is located in Finnish Lapland and was in operation up to December 2007. The mill which has a good standard has a capacity of 500 000 tonnes/year. Lappland Goldminers estimates that the identified mineralization is sufficient for approximately one year of production with good potentials for additional reserves as the mineralization is largely open at depth and under explored. Lappland Goldminers will start with prospecting in the mine with the aim of increasing mine life, while manning up the mine during the spring and summer with the purpose of taking up production in the autumn. The Pahtavaara mine has historically produced 30.000 ounces/year at a cash cost of approximately 500 dollar/ounce

With its position as the dominant company on the Gold Line in southern Lappland and its new Finnish acquisitions, Lappland Goldminers is very well positioned to become a major European gold producer active in two politically stable countries with a friendly mining culture.

Lycksele, April 2008

From the Board of Lappland Goldminers AB

Lappland Goldminers AB

Fäboliden Gold Project

-Mine and Process Plant-

Feasibility Study

April 2008

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Background – The Company

Lappland Goldminers AB is a mining and exploration company. The company is listed on the market place "First North" and on the Norwegian OTC list.

Lappland Goldminers has secured a number of gold deposits along the so-called Guldlinjen ("The Gold Line") in Västerbotten, Sweden. The company's strategy is to develop a profitable, producing mining company with centrally located processing plants, in Fäboliden in Sweden and in the Haveri area in Finland, which are supplied with ore from one or several mines through the company's own exploration, or alternatively through acquisitions.

Lappland Goldminers is a member of SveMin (the Association for Mining, Mineral and Metal Producers in Sweden - formerly called "Svenska Gruvföreningen").

For further information about the company, please visit; www.lapplandgoldminers.se

Executive Summary

Key features for Fäboliden Gold project:

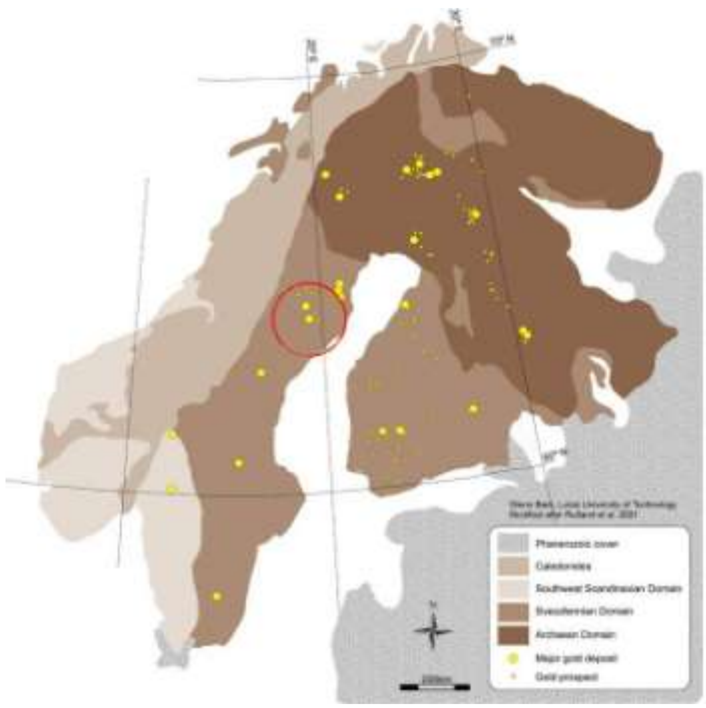
- *Measured and indicated mineral resources are estimated to approx. 60 Mt and contained Au at 2.4 M.tr.oz, corresponding to proven and probable reserves of 1.94 M.tr.oz. In addition there are inferred resources amounting to approx. 10 Mt (not included in this evaluation).*
- *Average Au grade after dilution in the open pit: 1.2 g/t.*
- *Average Au grade after dilution for the underground production: 1.07 g/t.*
- *Cash cost open pit, dollar/oz: 385*
- *Cash cost underground, dollar/oz: 485*
- *Mining and processing are scheduled to start 2011.*
- *Annual processing capacity: 4.6 Mt.*
- *Accumulated cash flow is estimated to approx. SEK 3,500 million (excluding interest costs) after 100% of the investment is paid off.*
- *Pay-off time: 4 years.*
- *Capital Expenditure (Capex) is estimated to SEK 1,770 million for the open pit and SEK 280 million for the underground mine.*

Introduction

Lappland Goldminers AB (SE556544-3339) (Lappland Goldminers) has commissioned Outotec Sweden AB (Outotec) to coordinate the work with this Feasibility Study for the Fäboliden Gold Project and to carry out engineering design and capital cost estimates for a process facility. Companies and key persons responsible for sections in the study are: Sweco (Civil works and related cost estimates), GeoVista AB (Mineral reserves and modelling), SRK Consulting UK (Open pit optimization), Ingemar Marklund, Mining Engineer, previously LKAB (Underground production and operational costs).

This Feasibility Study examines the:

- Development of a large mine with open pit and underground production in Fäboliden in Northern Sweden.
- Engineering and construction of a new gold and silver recovery facility (process plant) including all supporting facilities.



Gold deposits in Fennoscandia, Rutland 2001

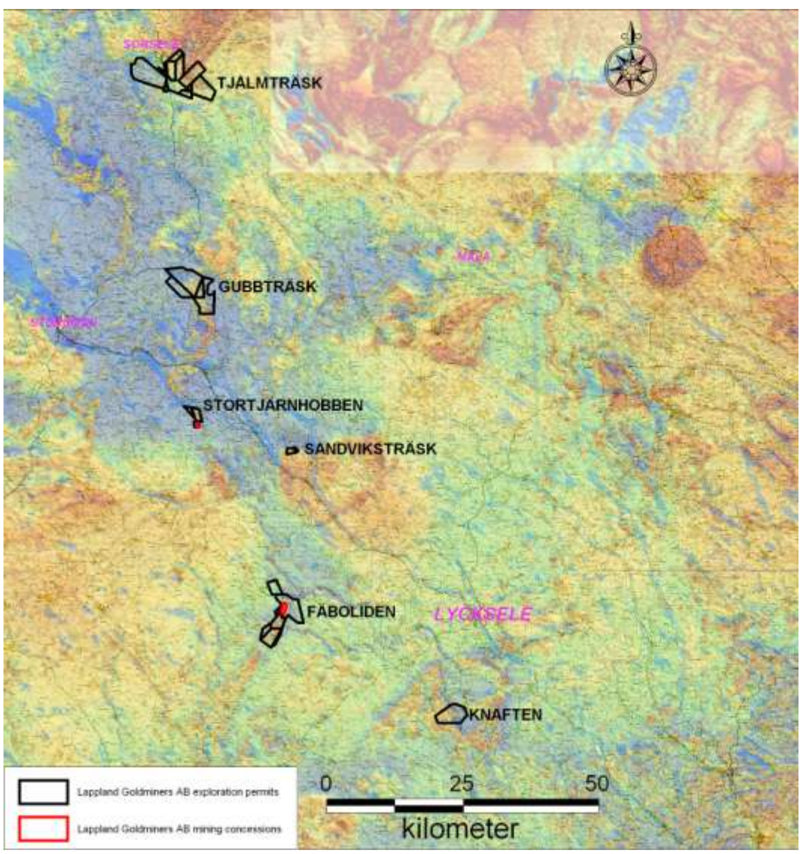
Background

Lappland Goldminers is holding 100 % of the exploitation concession Fäboliden K nr 1, covering 122 hectares. Fäboliden is situated about 40 km west of Lycksele in the county of Västerbotten. The Fäboliden gold deposit is situated in the centre of the “Gold Line”, a regional geological structure that reaches from northwest to southeast over Västerbotten from the Caledonian mountains to the Gulf of Bothnia. Lappland Goldminers AB and its subsidiary Lappland Guldprospektering AB currently hold approx. 42,000 hectares of exploration permits for metals including gold, subdivided among several areas, including Fäbolidens´ surroundings.

The intention of Lappland Goldminers is to mine the Fäboliden deposit, initially in an open pit down to a depth of approx. 200 metres and to build a processing facility, in order to extract gold and silver from the ore. The production from the open pit is planned to continue during 5 years. Mining will then continue with large scale, low cost underground mining using methods such as Sub Level Caving as used at the LKAB mines in Kiruna and Malmberget, Sweden.

The concentrator is designed to be used to process similar types of ore from other mines within a large surrounding area. Such concept is regarded appropriate from a cost and environmental view, especially since the concentrator is located on the Gold Line in an area with several known deposits and a high potential for further discoveries. Some 10 exploration and mining companies are currently actively exploring for gold along the Gold Line. Examples of projects are Barsele gold deposit (Northland Resources) and the Svartliden gold mine (in production since 2004 by Dragon Mining).

In addition there are a number of gold deposits held by Lappland Goldminers nearby to Fäboliden where the company is exploring the feasibilities for future gold ore production, implying future possibilities to support the process plant in Fäboliden with additional ore.



Location of gold deposits where Lappland Goldminers is carrying out exploration works

Resources and Reserves

The Fäboliden project comprises measured and indicated mineral resources estimated to approx. 60 Mt and contained Au at 2.4 M.tr.oz. In addition there are inferred resources amounting to approx. 10 Mt. The estimate shown below has been elaborated by GeoVista AB in full compliance with SveMin´s (The Swedish Mining Association) regulations, using a lower cut-off at 0.4 g/t Au.

		Tonnage (t)	Au g/t	Ag g/t	Cont. Au (g)	Cont. Ag (g)	Cont. Au kOz	Cont. Ag kOz
+250 m	Measured	24 961 000	1.26	3.27	31 409 000	81 688 000	1 010	2 626
	Indicated	1 435 000	0.97	1.95	1 398 000	2 798 000	45	90
250 m to -5 m	Indicated	30 500 000	1.23	3.69	37 376 000	112 604 000	1 202	3 620
-5 m to -150 m	Indicated	4 120 000	0.88	3.06	3 609 000	12 606 000	116	405
	Inferred	7 057 000	1.42	4.19	9 991 000	29 576 000	321	951
Footwall/Hanging wall	Inferred	2 470 000	0.61	1.49	1 524 000	3 670 000	49	118
Total		70 542 000			85 306 000	242 943 000	2 743	7 811

Resource Estimate 16th of March 2008

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

As for the inferred resource of approx. 10 Mt it is intersected with a number of drill holes but still not many enough to classify it as indicated. Additional drilling is ongoing and the inferred resource is expected to be upgraded to indicated during summer 2008.

The resource estimate has been converted to reserves according to the table below.

	Ore Mt	Au g/t	Ag g/t
Open Pit (Proven reserve)	22.9	1.20	3.12
Underground production (Probable reserve)	30.5	1.07	3.29

Mineral Reserve Statement 16th of March 2008

Production

Processing is planned to be at a level of 4.6 million tonnes ore per year assuming an average Au grade of 1.13 g/t (dilution included) and an average recovery of 84% then this will provide approx. 4.4 tonnes of gold annually (140,000 tr.oz per year). As for silver the annual average production is estimated to 7.5 tonnes (240,000 tr.oz per year).

Financial Evaluation

Required investments in the process plant, the supporting infrastructure and environmental costs are estimated to SEK 1,532 millions. Costs for mining equipment are assessed to SEK 240 millions. The cost estimates are based on budget quotations and a predominant portion of new equipment.

Operating costs for the open pit down to approx. 200 meters are estimated to an annual average of approx. SEK 400 millions with a unit cost for mining of 10.5 SEK/tonne and 41 SEK/tonne for processing. Consumables and energy costs constitutes a mayor part of the operating costs.

Investment cost for underground production is assessed to SEK 280 million. Operating cost for underground production is estimated to 50 SEK/tonne.

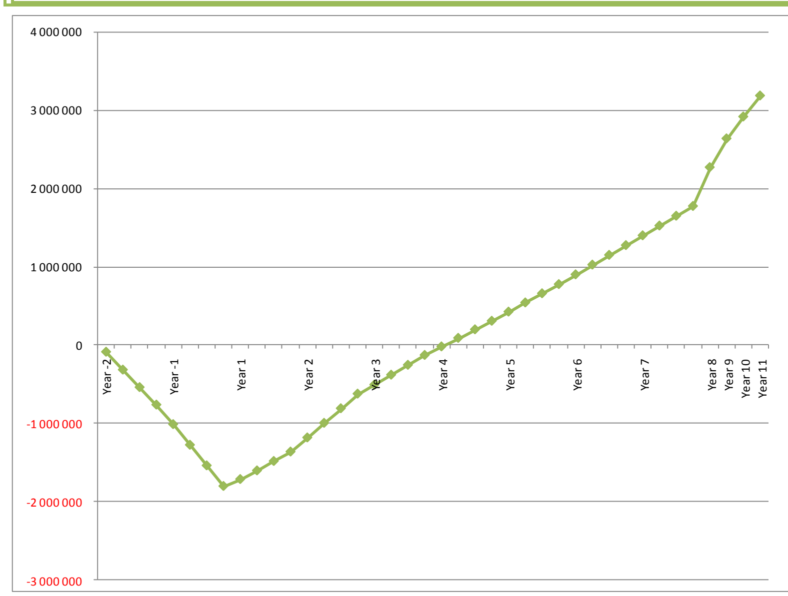
The Grand Total for *Capital Costs adds up to approx. SEK 2,000 million.*

The price for Au has been set to USD 933 per tr.oz and for Ag USD 17 per tr. oz.

Accumulated *cash flow*, including reclamation costs, but excluding interest and financing costs, is estimated to approx. *SEK 3,500 million*. *Pay-off time* is estimated to 4 years. The *net present value* of the accumulated cash flow is estimated to approx. *SEK 2,000 millions* at 5% interest rate.

Basic Assumptions

Capital cost Process Plant	SEK 1,533 millions
Capital cost mining open pit	SEK 240 millions
Closure cost	SEK 56 millions
Residual value after 11 years	15% of capital cost
Required working capital	SEK 100 millions
Annual process capacity	4.6 Mt
Environmental permit	5.0 Mt per year
Ore open pit	23 million tonnes
Number of production years open pit	5
Ore underground	30.5 million tonnes
Number of production years underground	6.5
Average Au grade open pit (dilution incl.)	1.2 g/t
Average Au grade underground (dil. incl.)	1.07 g/t
Average Au grade open pit + underground (dilution incl.)	1.13 g/t
Average Ag grade (dilution incl.)	3.2 g/t
Average recovery grade Au	84%
Average recovery grade Ag	50%
Gold price, Au (1st of February 2008)	USD 933 per tr. oz
Silver price, Ag (1st of February 2008)	USD 17 per tr. oz
USD (1st of February 2008)	6.44 SEK
Operating cost process	41 SEK/tonne ore
Operating cost mining open pit	10.5 SEK/tonne
Strip Ratio	1:4
Investment cost underground	SEK 278 millions
Operating cost mining underground	50 SEK/tonne
Interest rate (Net present value)	5%

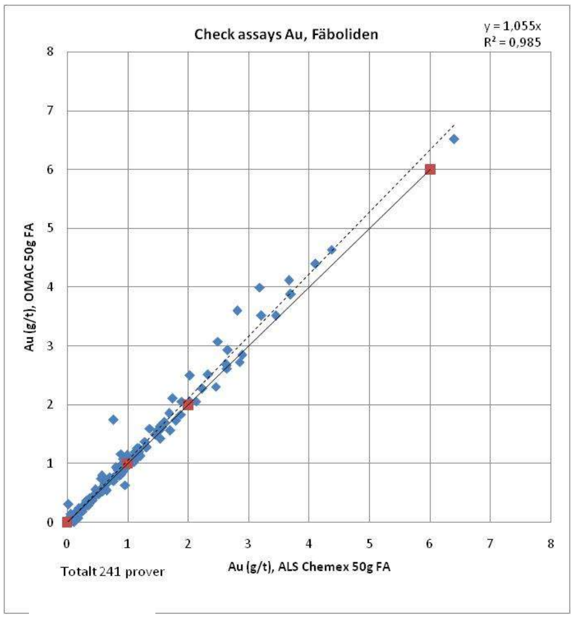


Accumulated Cash Flow KSEK

		Accumulated	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mine production																
Overburden stripping	Tonnes	12 093 163	0	3 448 717	4 923 761	1 436 977	2 283 708	0	0	0	0	0	0	0	0	0
Waste rock	Tonnes	84 973 058	0	9 567 992	14 641 009	15 019 153	15 250 408	15 411 536	15 082 960	0	0	0	0	0	0	0
Ore	Tonnes	53 397 048	0	14 902	4 625 743	4 580 796	4 585 056	4 596 866	4 493 685	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	2 900 000
	W/O	1.59														
Processing																
Ore	ton	53 397 048	0	14 902	4 625 743	4 580 796	4 585 056	4 596 866	4 493 685	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	2 900 000
Au grade nominal				0.55	1.11	1.40	1.15	1.12	1.26	1.25	1.25	1.24	1.06	0.93	0.89	0.86
Gold (recovery considered)	kg	51 020	0	6	4 293	5 696	4 427	4 323	4 752	4 793	4 793	4 755	4 079	3 590	3 440	2 074
	Oz	1 640 321	0	199	138 007	183 145	142 327	138 987	152 776	154 094	154 094	152 886	131 131	115 420	110 586	66 669
Recovery Au	%	84.2%	0%	75%	84%	89%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%
Ag grade nominal			0	0.37	2.51	2.48	2.34	2.54	2.68	2.35	3.32	3.30	3.30	3.30	3.30	3.30
Silver (recovery considered)	kg	86 050	0	3	7 193	6 230	7 336	6 987	7 976	7 590	7 590	7 590	7 590	7 590	7 590	4 785
	Oz	2 766 579	0	89	231 261	200 295	235 861	224 645	256 444	244 024	244 024	244 024	244 024	244 024	244 024	153 841
Recovery Ag	%	50.0%	0%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%
Revenue																
Gold in Doré	Oz	1 640 321	0	199	138 007	183 145	142 327	138 987	152 776	154 094	154 094	152 886	131 131	115 420	110 586	66 669
Gold revenue	k kr	9 855 901	0	1 198	829 219	1 100 430	855 172	835 103	917 959	925 878	925 878	918 616	787 905	693 503	664 456	400 584
Silver in Doré	Oz	2 766 535	0	44	231 261	200 295	235 861	224 645	256 444	244 024	244 024	244 024	244 024	244 024	244 024	153 841
Silver Revenue	k kr	302 880	0	5	25 318	21 928	25 822	24 594	28 075	26 716	26 716	26 716	26 716	26 716	26 716	16 843
Total revenue	**k kr**	**10 158 782**	**0**	**1 203**	**854 537**	**1 122 358**	**880 994**	**859 698**	**946 035**	**952 594**	**952 594**	945 332	814 621	720 219	691 172	417 427
Total operating costs	**k kr**	**4 724 623**	0	102 370	386 392	373 801	378 184	369 796	362 363	415 881	415 881	415 830	414 915	414 255	414 051	260 904
	kr/t ore	88.48														
	USD/oz Gold	447.25														
Operational results	**k kr**	**5 434 158**	**0**	**-101 167**	**468 145**	**748 558**	**502 810**	**489 902**	**583 672**	**536 712**	**536 712**	**529 502**	**399 706**	**305 964**	**277 121**	**156 523**
Capital costs (total)	**k kr**	**2 026 189**	**803 331**	**945 370**	**0**	**0**	**0**	**45 000**	**106 860**	**30 000**	**28 000**	**25 000**	**24 000**	**18 000**	**628**	**0**
Open Pit	**k kr**	**1 748 701**	**803 331**	**945 370**	**0**	**0**	**0**									
Underground	**k kr**	**277 488**	**0**	**0**	**0**	**0**	**0**	**45 000**	**106 860**	**30 000**	**28 000**	**25 000**	**24 000**	**18 000**	**628**	**0**
Equipment Spares	**k kr**	*24 169*	0	24 169	0	0	0	0	0	0	0	0	0	0	0	0
Sustainable capital	**k kr**	*18 000*		0	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500
Closure cost	**k kr**	*31 000*	0	0	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583
Reclamation bond	**k kr**	*25 000*	5 000	20 000	0	0	0	0	0	0	0	0	0	0	0	0
Overhead cost	**k kr**	*52 000*	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	
Residual value	**k kr**	*-300 000*	0	0	0	0	0	0	0	0	0	0	0	0	0	-300 000
Working Capital	**k kr**	0	40 000	40 000	20 000	0	0	0	0	0	0	0	0	0	0	-100 000
Cash flow	**k kr**	**3 553 801**	-852 331	-1 134 705	440 062	740 474	494 727	436 818	468 728	498 629	500 629	496 418	367 622	279 881	268 409	548 440
Accumulated	**k kr**		**-852 331**	**-1 987 037**	**-1 546 975**	**-806 501**	**-311 774**	**125 044**	**593 772**	**1 092 402**	**1 593 031**	**2 089 449**	**2 457 071**	**2 736 952**	**3 005 361**	**3 553 801**
Present Value of Expected Cash Flow		**4 179 722**														
Net Present Value (NPV)		**1 950 176**														
Internal Rate of Return (IRR)		**20.5%**														

Sensitivity Analysis and Discussion

The gold assays for cores from Fäboliden have been analyzed at ALS Chemex Laboratories in Vancouver and the results from these analyses have been used when defining the Au grade of the mineral reserves and resources. The same cores have systematically and for comparative reasons been checked at OMAC Laboratories during the last two years. In total more than 300 tests have been performed indicating a strong linear deviation between the results where *OMAC shows approx. 6 % higher values compared to ALS Chemex*. In addition, the results from the *test mining that was carried out in March 2005, clearly indicates Au grades that are significantly higher* compared to exploration drilling results. The Au grades that are applied for the financial evaluation in this Feasibility Study are therefore regarded as *under estimates*.

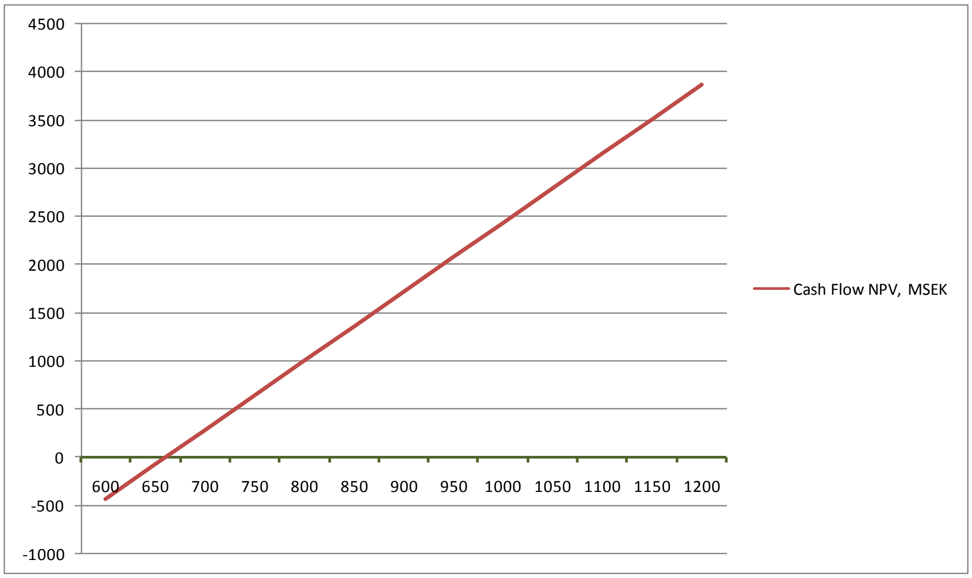


Comparison of test results (up to autumn 2007) from ALS Chemex and OMAC Laboratories. There is a strong linear deviation between the results where OMAC shows approx. 6 % higher Au grades.

Results from sensivity analysis calculations are outlined below:

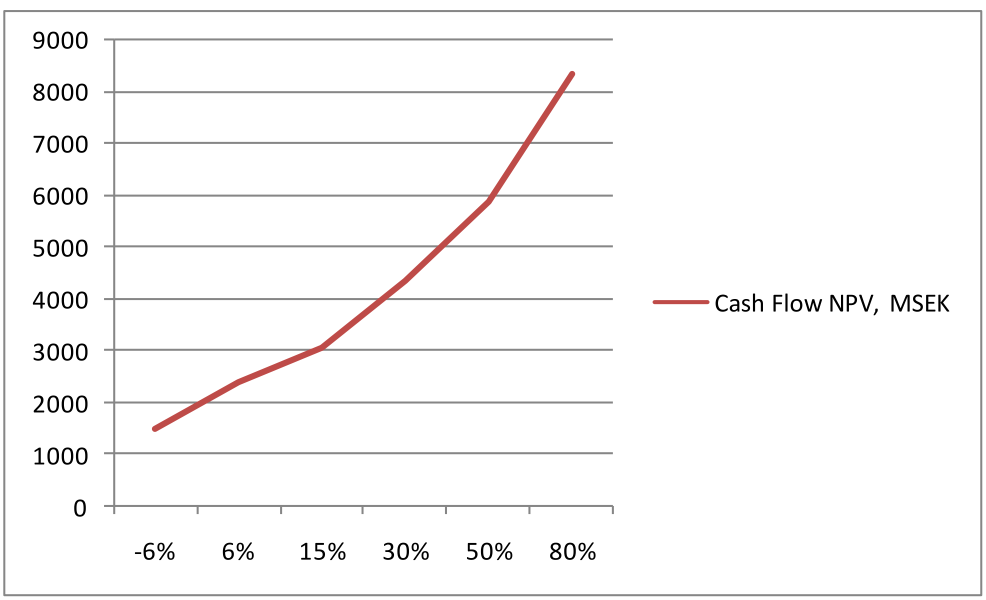
- Applying the *6% higher average Au grade* in accordance with the results from OMAC would mean a cash flow of SEK 4,249 million (NPV SEK 2,421 million) and a pay off time of 3.5 years.
- Applying the *30% higher average Au grade* would mean a cash flow of approx. SEK 7,052 million (NPV SEK 4,367 million) and a pay off time of 2.3 years.
- *Increasing the gold price with 10%* would mean cash flow of SEK 4,655 million (NPV SEK 2,705 million).
- *Decreasing the gold price with 10%* would mean cash flow of SEK 2,578 million (NPV SEK 1,281 million).
- *Increasing the total investment cost with 10%* would mean cash flow of SEK 3,405 million (NPV SEK 1,801 million).
- *Decreasing the total investment cost with 10%* would mean cash flow of SEK 3,702 million (NPV SEK 2,099 million).
- *Increasing the annual processing of ore* from 4.6 Mt to 5.0 Mt would mean a cash flow of SEK 3,791 million (NPV SEK 2,158 million) and 3.5 years pay-off time.
- Applying an *interest rate of 8%* would mean NPV cash flow of SEK 1,320 millions.

As shown in the figure below a gold price of approx. 700 USD/oz means a discounted cash flow of +/- 0 (assuming that the general conditions of today are unchanged). Assuming a gold price of 1,200 USD would imply a cash flow (NPV) of approx. SEK 4,000 million.



Cash Flow (Net Present Value MSEK, red line) with different Au prices (USD/oz, horizontal line)

Changes in Au grades, compared to the average grade of 1.13 g/t, are analyzed in the figure below. It shows that only smaller increases in the Au grade imply tangible increases in cash flow. For example 15% higher average Au grade (i.e. 1.3 g/t instead of 1.13 g/t) would mean a NPV cash flow of approx. SEK 3,081 million while 50% higher average Au grade (i.e. 1.7 g/t instead of 1.13 g/t) would mean approx. SEK 5,873 million.



Cash Flow Net Present Value MSEK with different Au grades (changes in percent compared to the average grade of 1.13 g/t)

Implementation

Lappland Goldminers received the required Environmental Permit from the Environmental Court in Umeå by the 21st of December 2007. The permit has been appealed by the Swedish Environmental Protection Agency and by some private persons and the Supreme Environmental Court has set the time for hearing of the appeals to May 2008. However, since the Swedish Environmental Protection Agency is questioning technical and economical details, not the project itself, Lappland Goldminers is confident that the permit can be finally approved during the summer of 2008. By this the time schedule for the implementation of the project won´t be affected on the condition that the project receive a permit to start preparatory construction works during summer 2008.

Major construction works are then expected to commence early 2009 and *production is scheduled to commence in the beginning of 2011*. The company has an ongoing dialogue with several possible stakeholders with the intention to get a favourable financing packet for the project and required operational capital.

Overwiew Fäboliden Gold Project

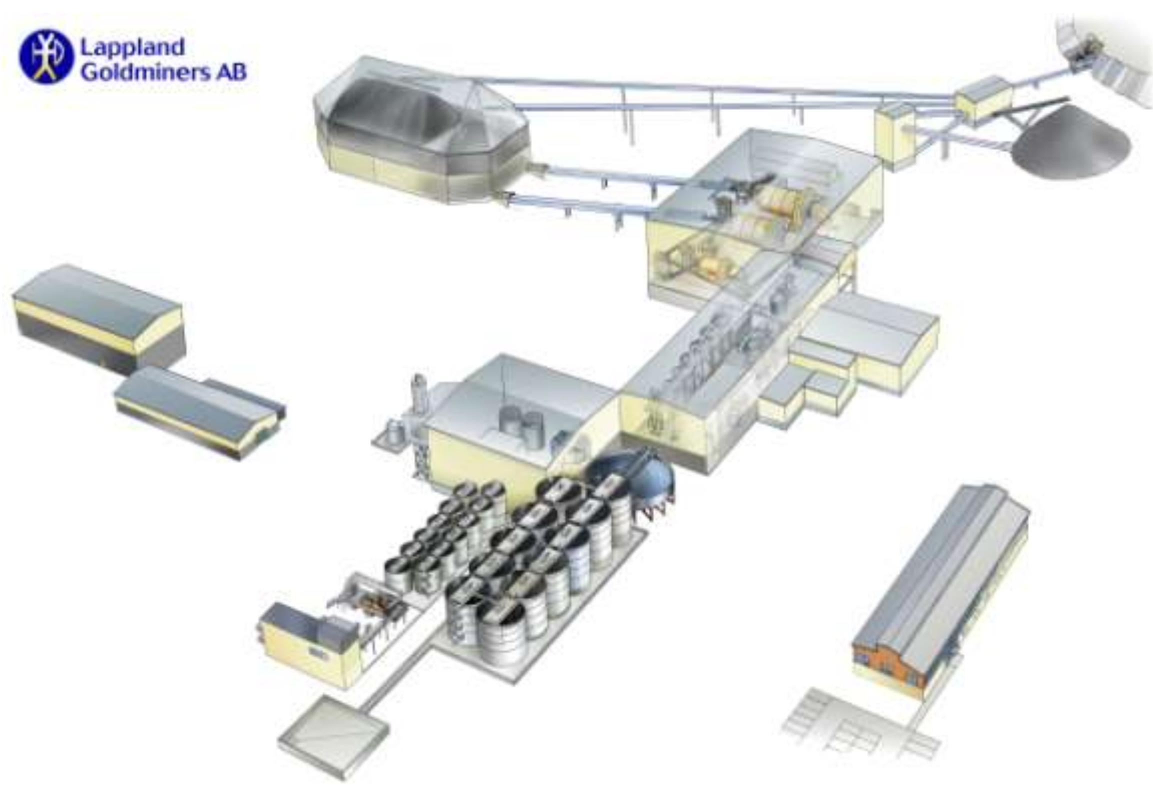
Risk Management

A number of risks/possibilities possibly affecting the financial results have to be considered and managed when carrying out the project:

- *Major future changes in the gold price*: The project means processing of large volumes of low grade ore which makes it sensitive for the development of the gold price. Hedging of parts of the future gold sales has to be considered.
- *Exchange rate changes/fluctuations*: The present exchange rate of approx. 6.4 SEK per USD means that SEK is relatively strong in relation to USD. This is favorable when buying equipment where the payment is in USD. As for the future when production starts the project would be favored by a stronger USD since the selling of gold will be in USD. On the other hand the price for tires, fuel and explosives and other commodities are often depending on the exchange rate and a stronger USD means higher prices. Dealing with financial risks, among other risks related to the exchange rate and gold prices, requires a financial strategy. Such a strategy will be developed by Lappland Goldminers with the aim to deal with financial uncertainties in a proactive and efficient way.
- *Variations in Au grades*: The average Au grade (dilution included) for the ore body is assessed to approx. 1.13 g/t. Recent checking of laboratory results and the test mining carried out back in 2005 clearly indicates that the actual grades will be higher. If this is the case the financial outcome of the project will be affected in a most positive way. Further testing on this will be carried out during the coming years and before production starts.

Drilling blast holes for test mining in Fäboliden

- *Investment cost increases*: Costs for mining and process equipment has increased in a tangible way during the last years and further increases can be expected. During 2008 detailed design will be carried out for the Fäboliden process plant. The design work will focus on developing and improving the preliminary design accounted for in this study, aiming at cost-efficient production, environmental concern and reductions of the investment cost. The procurement of the process plant will be carried out through competition between different possible suppliers.
- *Delays in the delivery of required equipment*: The immediate procurement of a mill to the process plant is of crucial importance for the time schedule of the project since the delivery time (including in site installation) is estimated to approx. three years. Lappland Goldminers is therefore presently evaluating a number of mill tenders and is planning to sign a contract on this in the nearby future. As for other equipment delivery times are less than 18 months.
- *Major future increases in the energy price*: Energy prices have increased during the last years and further increases can be expected. Lappland Goldminers has the intention to establish a long term agreement for energy supply with one major supplier, avoiding future fluctuations in the energy price. In addition, Lappland Goldminers is involved in a Wind Mill Project located in Fäboliden close to the process plant, which in a number of years might turn out to be an alternative solution supplying the process plant with environmentally friendly energy to a long term steady price. As for future mining activities the production will require large amounts of diesel fuel to the trucks and also to other mining equipment. However, the price for diesel fuel is usually not negotiable to any larger degree which will mean that current market prices will have to be paid.



Process Plant Fäboliden Gold Project

Site Map

- *Changes in EU and Swedish legislation*: Legislation concerning environment, taxes, work safety etc is continuously changing/developing. The Environmental Permit Lappland Goldminers has received takes foreseeable coming changes initiated by EU and Sweden into consideration within the legislative fields of health, environment and safety. This is a good thing since it reduces the uncertainty for future operations in the mine and in the process plant. Over and above this other legislative changes will take place not foreseeable today. These will mean uncertainties in different aspects which have to be dealt with when occurring.
- *Unexpected environmental impact*: Lappland Goldminers has within the frames for a comprehensive Environmental Impact Assessment worked with high ambitions to identify potential environmental risks and to implement risk eliminating measures and mitigation measures into the project. The environmental issues has to stay in focus, during the construction phase as well as when the mining and processing is getting operational, with the objective to avoid any future unexpected environmental impact.
- *Disruption to operations due to labor disputes or health and safety issues*: Different kind of personnel policies will be elaborated in good time before operations start. These will guide the management staff and help operations to run in a smooth, efficient and safe way. However, policies don't mean that all kind of problems can be avoided and there is no doubt that unexpected and unwanted events will occur. Such events have to be dealt with in a proactive way and with an ambition to avoid any future operational disruptions.

1. Introduction

1.1 Participants

Lappland Goldminers AB (SE556544-3339) (Lappland Goldminers) has commissioned Outotec Sweden AB (Outotec) to coordinate the work with this Feasibility Study for the Fäboliden Gold Project and to carry out engineering design and capital cost estimates for a process facility. Companies and key persons responsible for sections in the study are: Sweco (Civil works and related cost estimates), GeoVista AB (Mineral reserves and modelling), SRK Consulting UK (Open pit optimization), Ingemar Marklund, Mining Engineer, previously LKAB (Underground production and operational costs). The financial evaluation has been compiled by Lappland Goldminers based on the results brought forward by the different companies involved.

This Feasibility Study is based on a Pre-Feasibility Study (23rd of December 2005) elaborated by Outotec.

1.2 Objectives

This study focuses on proven and probable gold ore reserves in Fäboliden, Lycksele municipality of which Lappland Goldminers is holding 100 % of the exploitation concession (Fäboliden K nr 1, 122 hectares).

The intention is to mine the deposit, initially as an open pit down to a depth of approx. 200 metres and to build a processing facility, in order to extract gold and silver from the ore. The production from the open pit is planned to continue during some 5 years and will be followed by underground production.

The objective for this Feasibility Study is to:

- Examine the economical, technical and environmental feasibilities for a large mine (open pit and underground) in Fäboliden including a process plant and all supporting activities for gold and silver production.

1.3 Report Structure

The structure for the Feasibility Study is outlined below:

Executive Summary
1. Introduction
2. Geology
3. Exploration
4. Mineral Reserves and Modelling
5. Mining-Open Pit
6. Mining-Underground
7. Plant and Process
8. Infrastructure
9. Environment and Tailings Storage Facilities
10. Capital Costs
11. Operating Costs
12. Gold Price
13. Financial Evaluation
14. Time Schedule
15. Human Resources – Staffing and Operations
16. Site Map

Tables and Figures
Date and Signature Page

1.4 Appendices

Enclosed to this Feasibility Study are:

- Appendix 1(5) Appendixes to the Main Report
- Appendix 2(5) Process drawings
- Appendix 3(5) Equipment list
- Appendix 4(5) Civil works drawings
- Appendix 5(5) Civil works drawings

1.5 Disclaimer

This Feasibility Study contains assessments and statements concerning the future financial, operational and environmental outcome of Fäboliden Gold Project that include risks and uncertainties. Several factors can cause the assessments and statements to differ from those indicated here, for example the future price of gold, future exchange rates, the actual grade of gold in the ore, difficulties in mining operations, changes in government regulations etc. As a consequence of this Lappland Goldminers AB undertakes no responsibility for any future deviations from the assessments and results accounted for in this Feasibility Study.

2. Geology

2.1 Regional Geology

Fäboliden gold deposit is situated in the northern part of the Bothnian Basin SW of the Skellefte District in the Fennoscandian Shield and the western part of the East European Craton. The Paleoproterozoic "Gold Line" in Västerbotten is running for about 250 km and strikes from NW to SE from the Caledonian Border to the Bothnian Gulf.
The bedrock geology as chronostratigraphical units are made up of:

- Dolerite, 1.27-1.00 Ga
- Granitoids, Skellefte- and Revsund type, 1.86-1.75 Ga
- Mafic metavolcanic rocks, 1.88-1.86 Ga
- Metavolcanic and metaigneous rocks, 1.91-1.88 Ga
- Metagreywackes, 1.95-1.87 Ga
- Metavolcanic and metaigneous rocks, 1.96-1.91 Ga

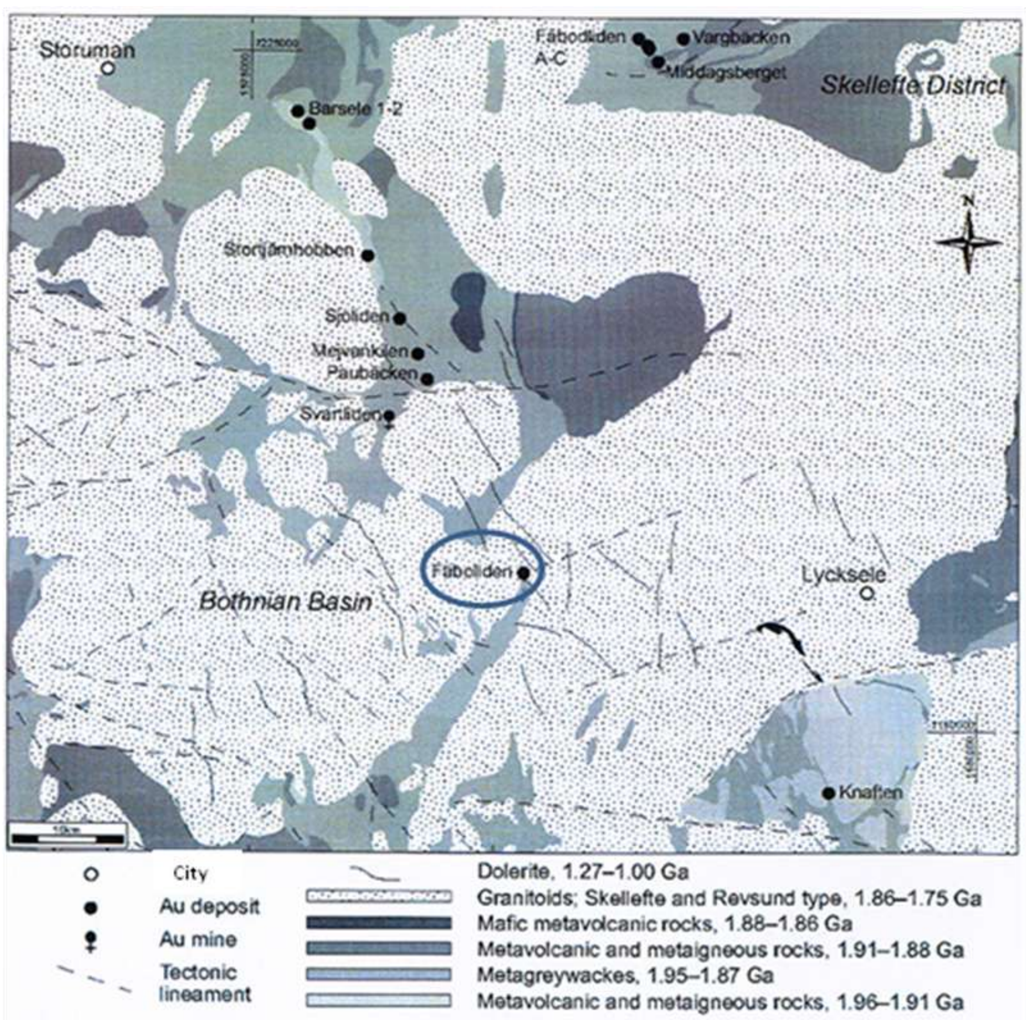


Figure 2.1. The Gold Line in Västerbotten, from Geological Survey of Sweden.

The Fäboliden area is poorly exposed, except in the most northern parts where basic volcanics dominate the outcrops with possible pillowed lavas and banded thin layers of exhalative chert with magnetite and garnets, banded iron formation (BIF). In this area intrusive granites of Revsund type are present. Further to the west and to the north strongly sheared metagrey-wackes are present with a N-S strike and mostly vertical dip. These greywackes are weakly anomalous in gold. The setting of "Fäboliden orogenic gold deposit" and another 11 known deposits on the Gold Line are described by Bark and Weihed 2003.

2.2 Geological Information

The geology of Fäboliden is defined through 307 diamond drill holes and 56 705 meter of cores. They have been drilled in sections in a systematical pattern 50-70 meters between the sections and approximately 50 meters between the holes in 24 sections from surface to 200 meters level and in a broader pattern to 550 meters level. The bedrock has been exposed in a test pit and also in two trenches about 600 meters apart. In addition, the area around Fäboliden has been drilled systematically by bed rock chip sampling programs as part of geochemical surveys. Drilling north and south of the open pit position has been carried out in 12 additional sections with various spacing.

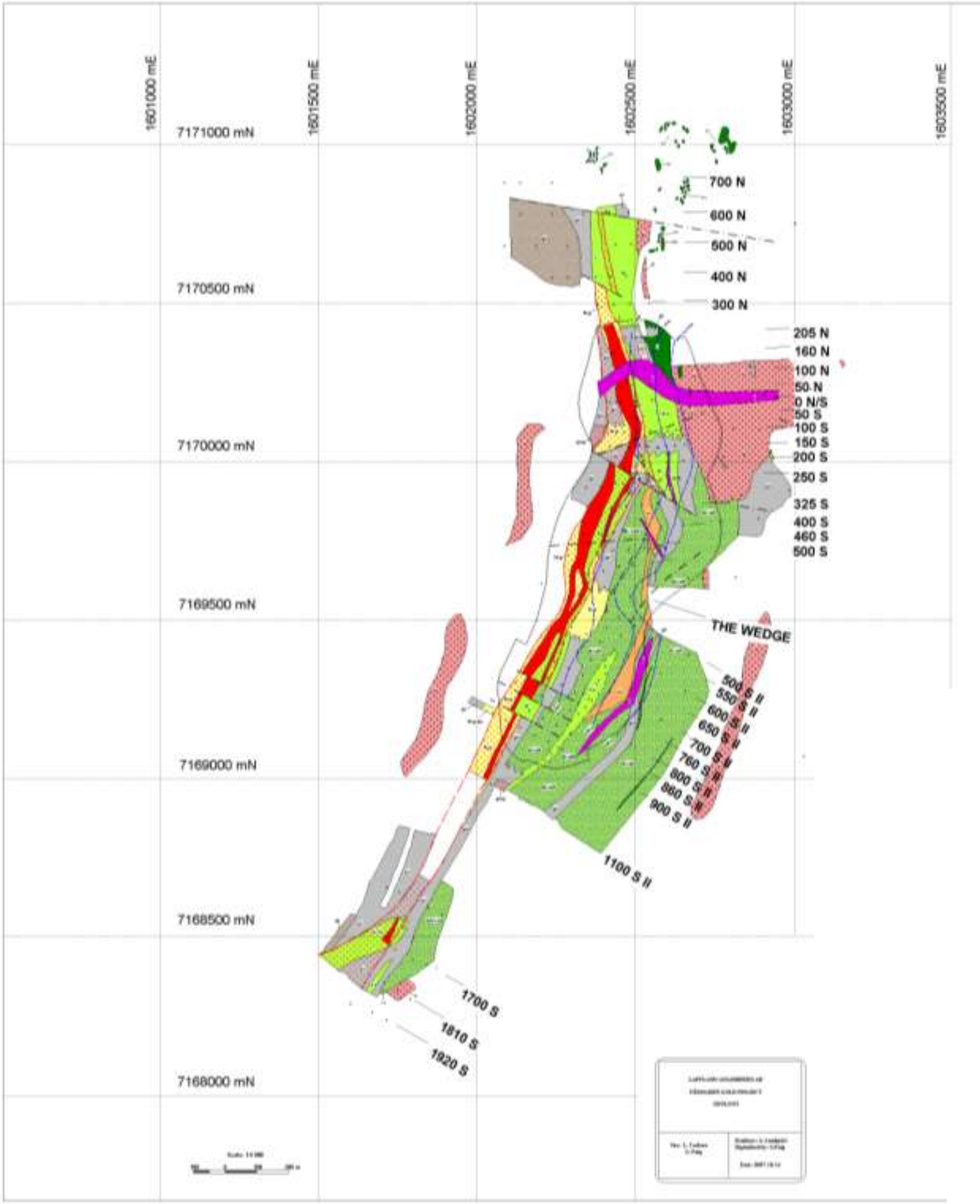
The geology of Fäboliden is therefore well known today, see references (Carlson 2000, 2001, 2002, 2005 and 2006). A Technical Report on Fäboliden Gold Project by SRK/J-F Couture 2006 states that the work carried out for the Fäboliden Gold Project follows the guidelines of the Canadian Securities Administrators National Instrument 43-101 and Form 43-101F1.

From the massive geological information from 24 sections on the open pit situation, and the additional 12 sections on North and South Fäboliden and sterilisation drilling for the Process Plant at Fäboliden a digitized map, "Fäboliden Geology" has been produced at a 1:5 000 scale.

The geological map of Fäboliden shows an extended zone striking N-S and NE-SW for 2 700 m and with a width of 300-400 meters with dominating intermediate volcanics and greywackes. This zone is squeezed between Revsund granites and paragneisses after greywackes. The total length of the zone drilled is 4 200 meters of which the open pit mineralization makes up 1 300 m along strike.

The geological information is present on 24 vertical sections and horizontal sections on every 50 meters level from 450 m to 300 meters above sea level. Length sections are showing the mineral intensity for gold and silver (m x Au g/t and m x Ag g/t) and in addition there are ground magnetic maps showing the magnetic trends characterizing the bedrock below the till cover.

A geological and a technical evaluation of rock stability properties has been carried out in vertical and horizontal sections (Carlson 2007), see below.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100 N
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THE WEDGE
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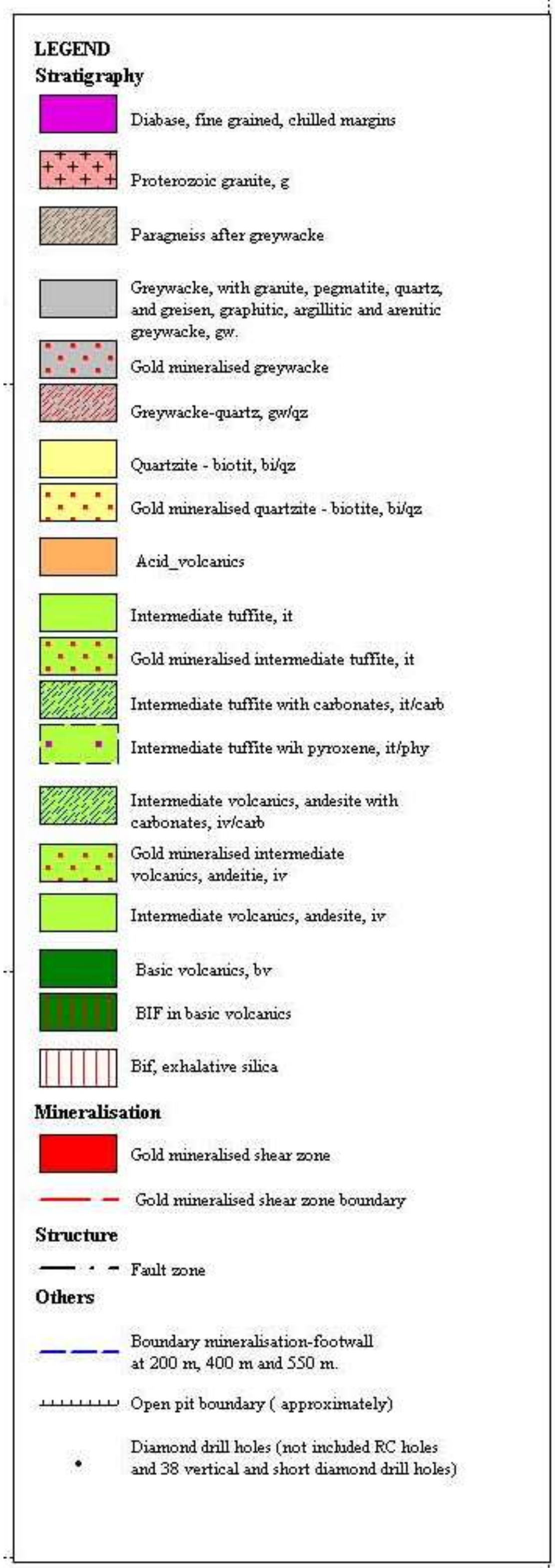

Figure 2.2. Fäboliden Geology, (Geology by L. Carlson, drafting by A. Lundquist and digitization by G. Puig).

Overburden		
	Δ	**Till**
Stratigraphy		
Intrusive rocks		
	d	**Diabase, sulphide free, fine grained close to contacts**
	uv	**Ultrabasic dike, serpentinized peridotite.**
	pyx	**Ultrabasic dike, pyroxenite**
	g	**Granite of Revsund type**
	grn	**Greisen veins**
	p	**Pegmatite veins**
	q	**Quartz veins**
Metamorphosed sedimentary- and volcanic rocks.		
	gw	**Greywacke, arenitic, argillitic or/ and graphitic.**
	av	**Acid volcanic, at = acid tuffite**
	ch	**Chert, exhalative SiO_2, BIF**
	iv	**Intermediate volcanics**
	an	**Intermediate volcanics, massive andesite**
	it	**Intermediate tuffite, carb. = carbonate banded.**
	itb	**Intermediate tuff breccia**
	itp	**Intermediate feldspar porphyritic volcanics**
	bv	**Basic volcanics**
	bt	**Basic tuffite**
Metamorphic alterations		
	gn	**Paragneiss after greywacke**
	bi-qz	**Biotite quartzite, biotite, quartzite alteration.**
	bi-sk	**Biotite alteration, partly intensive brown sheared structure**
	sk	**Skarn alteration**
	pyx	**Pyroxen alteration of intermediate volcanics- and tuffites**
	fsp	**Feldspar alteration of intermediate volcanics- and tuffites or greywackes**
	gt	**Garnet alteration**
Impact of tectonic elements (rock mecanic information.)		
	vvv	**Brecciated zone**
	ΔΔΔ	**Mylonite**
	KF	**Core loss**
	~	**Shear zone, graphitic and chloritic**
	—·—	**Fault**
	A	**Artesian water in drillhole**
Mineralizations		
		Gold diagrams in sections, 1 cm equal to 1 g/t Au (cut at 3 g/t Au).
		Ore- boundary in resource estimations at 0.4 g/t Au cut-off
		Gold mineralized waste < 0.4 g/t Au
Tectonic symbols		
	45°	**Strike and dip of layering and shearing**
	—·—	**Fault**
	—○45 °	**Plunge**
Various symbols		
	—○	**Diamond drill hole**
	/	**Lithological boundary**
	---	**Open pit boundary by B. Bärnman, 2004.**

Table 2.1. Fäboliden, legend for geological and technical rock evaluation, (Carlson 2007)

2.3 Geological Interpretation

The stratigraphy at Fäboliden is made up from metavolcanics, basic, intermediate and acid volcanics and metasediments, greywackes, which are intensively sheared and surrounded by granites and also intruded by thin granite-, pegmatite- and quartz veins. The greywackes are altered at the contacts to the granites into gneisses. They are foremost known from sterilization drilling for the beneficiation plant, the deep exploration program and exploration drilling on Fäboliden South. Relict sedimentary and volcanic structures are rare. Limited areas with chert (banded iron formation, bif) are interpreted as rhythmic exhalation of silica and represents centres of volcanic activities. In the area north of Fäboliden is an area of outcrops with remains of pillow structures and bif. This area is interpreted to be a volcanic centre. In section 400S, 460S, 500S and the “Wedge” massive andesite is present which is interpreted to be intrusions or feeder pipes to the intermediate volcanics and tuffites. The intrusions are relatively unaffected by shearing and have been outlined at 350 meters level and downwards. The massive andesite is partly surrounded by feldspar porphyritic intermediate volcanics and lapilli tuffs.

Correlation of the stratigraphy is complex between the sections. This is due to the fact that the stratigraphy is crossing over the schistosity. This can be seen from both vertical and horizontal sections.

The gold mineralized shear zone and weakly gold bearing waste on the hanging wall and foot wall is consistent for about 1 300 meters in N-S direction on a horizontal width of 50-100 meters. However in section 150-200S the horizontal width of the gold mineralization is 150 meters. This is due to late tectonic repetitions of the gold mineralised zone. The boundaries for resource estimations (Carlson 2006) can be seen from vertical and horizontal sections (Carlson 2007). The dip of the mineralized zone is plotted on the footwall of each section. The dip varies from 40 to 80 degrees E and is in average 60 degrees E.

The diabase, which is sulphide free intrudes the stratigraphy and is not sheared and cuts through all the stratigraphic units with a flat dip from section 100N towards south. The diabase is in general 25 to 30 meters thick in most sections in the open pit situation.

2.4 Mineralogy

The mineralogy of the Fäboliden gold mineralization is summarized as follows by Ekström 2001, based on five polished thin sections. The mineralization is in greywackes, which are penetrated by quartz veins (three samples). The gangue in the samples consists mainly of quartz, feldspar, biotite, muscovite and turmaline. Further there are sericite, amphibol, titanite, rutile, apatite and zircon.

Arsenopyrite, pyrrhotite and ilmenite are the dominating ore minerals. Chalcopyrite, sphalerite, galena, gold, pyrite, Au- or Pb- tellurides, as hessite and aguilarite or Ag-sulfosalt are present in minor amounts.

The gold is present partly as intergrowth with arsenopyrite and partly as inclusions or on fissures in arsenopyrite. Some gold grains were observed in the matrix. The grain size of gold is 0.002 to 0.040 mm.
Au- or Pb-tellurides are present mainly in arsenopyrite and in some extent in galena. In polished sections a blue mineral (aguilarite or Ag-sulfosalt) can be seen, which together with gold is very fine grained and is therefore difficult to control by optical means.

A mineralogical summary can also be seen from Bark and Weihed.

2.5 Breccia Zones

All breccia zones have been logged and are noted in core logs. The brecciated zones have been marked on vertical sections and where it is possible to make connections they have also been joined up on horizontal sections (Carlson 2007). For that purpose, brecciations, core losses and tectonic breccia zones have been utilized.

Breccia zones are most prominent on the hanging wall from section 400S to 900S. From section 500S-II to 860S-II the breccia zones are many and strike parallel with the schistosity, the mineralised zone, in NNE-SSW and dip 45 to 62 degrees SE. In section 400S there is a wide breccia zone on the hanging wall of the east boundary of the planned pit. This breccia zone will possibly affect the stability of the upper part of the open pit.

2.6 Brecciations and Mylonitic Zones

Major tectonic brecciations are present on the hanging wall and on the foot-wall in section 800S and in 860S. The tectonic breccias are open with coarse fragmentation and tight fine grained mylonitic zones. In section 860S the breccia zone in the foot-wall of the ore zone has a horizontal width of about 20 meters.

2.7 Core Losses

All core losses are notified from the drill cores in the core logs. For most zones the widths of the core losses are also notified. The biggest core loss, 1.5 meter, is in section 250S in connection with breccia zones, which have been interpreted to be a steep normal fault. In section 800S, drill hole 200144 and 200145, there are tectonic breccias and core losses, which are interpreted to constitute a fault zone with a displacement of 70 meters for the east block. The fault zone is on the western boundary of the open pit. Part of the ore zone has been down faulted and can have resulted in a repetition of the ore. This position has not been included in the resource estimation of 2006, see Carlson 2006.

2.8 Shear Zones

Shear zones with graphitic faces and striations show fault structures. Some clayey and chloritic shears are present but they are few.

2.9 Schistosity

Schistosity is a general feature of the ore and the wall rock of Fäboliden gold deposit with a strong cohesion between the schistosity planes, which affects the fragmentation of the rock at blasting and the grinding of the ore. Between the A-B-zone in certain sections there is a schistose biotite schist with low cohesion. The character of this rock with low gold content can give a special character at blasting, which means that loading of waste rock will be facilitated.

2.10 Normal Faults

Normal faults can foremost be seen from dislocations of the diabase and the mineralised zone itself and shears, brecciations and brecciated zones. At the rectification of drill hole information to sections with surveyed data the normal faults have been reduced or even disappeared (Carlson 2007). Earlier interpreted faults can be seen from the geological map of Fäboliden. This means that there is a reason to continue with the interpretation of faults, which can affect mining at the Fäboliden open pit. The ground magnetic survey map can thereby become useful.

2.11 Artesian Water

Artesian water is present in drill hole 200111 in section 900S. This hole was abandoned at 93,2 meters depth due to rock fall in the upper part of the hole. The hole gives artesian water all the year round and also precipitation of iron-oxides.

2.12 Additional Technical Information

In section 800S the western most drill hole 200143 intersects 28 m of till before it hits the bed rock but shortly afterwards the hole again intersects moraine before it finally hits the bed rock at 35 meters depth. This means that there is a canyon in the south- west of the open pit boundary. The massive andesite, which is present in section 400S to the "Wedge" from 425-300 meters level can give a specific fragmentation which can facilitate to distinguish between ore and waste at loading. The andesite has only low gold values.

2.13 Photographic Documentation of Drill Cores

All drill cores are systematically photographed and are thereby a source of information for further rock mechanic evaluations/studies.

2.14 References

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(2001) Fäboliden Gold Project 2001.
(2002) Fäbolidens guldprojekt, Lycksele. Resultat av diamantborrningar 2001.
(2005) Fäbolidens guldprojekt, Lycksele. Resultat av diamantborrningar 2004.
(2006) Fäbolidens guldprojekt, Lycksele. Resultat av diamantborrningar 2005.
(2006) Fäboliden, haltverifiering november 2005.
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Ekström, M., (2001) Fäboliden, Mineralogisk undersökning av fem malmprover. SGAB Analytica

3. Exploration

3.1 Background

Fäboliden Gold Project was developed at the same time as several other projects were going on along the Gold Line. The Gold Line was by then a geochemical gold anomaly trend discovered by till sampling in Västerbotten County by NSG/SGU during the 1980´s. On this trend among others the following gold deposits and projects were known; Ersmarksberget in Sorsele, Barsele, Stortjärnhobben, Svartliden gold project in Storuman, Fäboliden and Knaften in Lycksele and Yxsjö in Åsele and various other gold projects at different stages.

Fäboliden, in the Lycksele area, was first drawn attention to by three enthusiastic prospectors, Jan-Åke Unée, Eric Sjölund and Torbjörn Grahn. They used the classical way of boulder hunting to locate magnetic arsenopyrite boulders containing gold when assayed. The boulders were found at Norrån in 1988 and the prospectors formed the company Lappland Mineral HB with the aim to carry out exploration and to locate the Mother Lode to the gold mineralized boulders. In 1992 Fäboliden was located in the ice direction from Norrån to Fäboliden about 17 km to the NW and a joint venture was formed with International Gold Exploration AB, which acquired 50 % in Fäboliden from Lappland Mineral HB. A grid system was constructed, geochemical humus sampling and geological mapping were carried out together with ground magnetic surveys.

The first diamond drillholes at Fäboliden were drilled in 1993. The results were not convincing in any way. In 1997 a trench was cut across the strike of Fäboliden, Perdahl´s trench, at 205 N and low- grade gold mineralization was proved in the bed- rock. Another five holes were drilled in 1998 by Lappland Goldminers AB, the new company name of Lappland Mineral HB. Low grade gold mineralisation was intersected on 25 meters width in one of the holes.

In 1999 the grid system was enlarged and GeoVista AB extended the ground magnetic survey to the south. In 1999 Lappland Goldminers AB acquired 40 % of International Gold´s part in Fäboliden and a new drilling program comprising 17 diamond drill holes was carried out. Lappland Goldminers now had 90 % interest in Fäboliden. The 14th of September 2002 Lappland Goldminers acquired the remaining 10% of Fäboliden and is since then the sole owner of the property.

In April 2000 Lappland Goldminers AB had four exploration concessions in Fäboliden covering an area 8 x 2 km covering an aeromagnetic anomaly bounded to the south of a concession by Equinox Resources NL, Perth of Western Australia. In this area for 15 x 4 km Lappland Goldminers had a 10% share.

After the drill program in 1999, total diamond drilling was 25 holes and 2 387 meters. Assaying had been carried out at SGS Laboratories in England and France and American Assay Laboratories in Reno, USA by Fire Assay method and Cyanide Leaching for gold and ICP assays for multi-element-assays.

Gold mineralized boulders from Fäboliden assay 0.4 - 33 g/t gold. Out of 28 boulders 9 assays contained more than 5 g/t gold. In year 2000 geophysical surveys had delineated the magnetic anomaly in the bedrock and geochemical humus sampling had outlined a gold anomali within the northern part of the concession area. A trench could then be located and was dug for 50 meters. All samples in this trench had anomalous gold values.

Figure 3.1 Drilling operations in Fäboliden

3.2 Evaluation of Drilling Results in April 2000

Drilling results in 10 sections from 800N to 250S were compiled and evaluated (Carlson 2000). The most exciting result could be seen in section 250S and drill hole 99016 from 29,10- 61,8 m, a width of 32,7 meters, having an average grade of 1,16 g/t gold. In section 250S it is evident that the horizontal width of the gold mineralization is 155 metres at the surface and that the same width also is indicated at 50 and 100 meters level.

In section 250S the gold mineralization´s width is 155 m at surface. The same width is indicated at 100 meters depth. In this zone drill hole 99016 averages 1.16 g/t gold on 32.7 m close to the surface. This intersection and its location on an extensive magnetic anomaly is the base for the interpretation that a deposit with a large tonnage is present at Fäboliden.

As the highest gold value in drill intersection was 7.1 g/t gold and with a number of mineralized boulders assay from 5 to 33 g/t gold it was evident that the origin of these had not been found. There were accordingly indications for higher grades.

The gold potential magnetic structure is widening to the south from the drilled area and can be followed 3.5 kilometres in the strike direction to the south within the concession area. A gold mineralized boulder with 5 g/t gold was found at 3500S and an outcrop with arsenopyrite was located on 4200S.

The evaluation showed that Fäboliden had developed from a prospect of gold mineralized boulders to a gold potential project with an inferred resource of "High tonnage, low grade, bulk mineable and heap leachable deposit". This means that the deposit has a big potential for a large tonnage with low grade which can be beneficiated with open pit mining and leaching (CIL/CIP).

3.3 Exploration Strategy

With the aim to indicate a large low- grade gold deposit with more than 1.5 g/t gold, 1.5 million troy ounces, the following recommendations were made.

Phase 1. Infill drilling and exploration drilling,
Diamond drilling, complementary drilling and additional drilling, to prolong existing holes in the north part of Fäboliden from section 200N to 250S, a total of 1 250 meters. Surveying of the concession area of Fäboliden from 300S-200S with the following activities, geophysical survey on 2.1 km^2 and bottom till and bed rock chip sampling in 45 points with a spacing of 100 m between the holes and 200 m between the section from 400S to 1400S. A total of 2 x 450 meter of drilling followed by assaying of deep till and bedrock chip sampling on gold and multi-elements by ICP, geological and mineralogical evaluation of chip samples, diamond drilling of 37 drill holes and assaying of mineralised sections and evaluation and estimation of "Indicated Resources".

Phase 2. Infill drilling and exploration drilling,
Repetition of phase´s 1 exploration methods within the concession area from 2000-4000S and infill drilling from 400S-2000S.

Phase 3. Infill drilling and resource estimation from 2000-4000S
Drilling and resource estimation from 2000-4000S.

3.4 Exploration Work and Results at Fäboliden 2000-2004

The geochemical and geophysical surveys were carried out as planned in the autumn 2000. Bottom till and bedrock chip samples were sampled in 94 points with 100 m between the samples and 200 meters between the lines by Dala Prospektering AB. Samples were assayed by Swedish Geochemical Services AB in Piteå for gold and multi element assays by ICP.

Gold grades in till were from 1-27 ppb Au per tonne. Arsenic values in the till were up to 428 g/t As and copper and zinc values were at max 448 g/t Cu and 530 g/t Zn. The gold values in bed-rock chip samples ranged from 0.01-1.17 g/t Au. The highest gold value at 1.17 grams per tonne was in sample point 500S/200E.

The survey gave an average depth of the till cover to 7-10 metres with exception of the SW area where a deep canyon with up to 35 meters of till seems to be present. The examination of bed-rock chip samples gave an insight in the geology and became the base for a preliminary geological map. This information together with the magnetic survey map gave the most excellent information for the guidance of further exploration.

Drilling was carried out from October 2000 with 14 holes and 1 659 meters to the end of the year. In 1999 the best result of drilling was in hole 99016 in section 250S with 32 m x 1.16 g/t Au in a 150 meter gold anomalous zone. With further drilling in 2000 and assay results being available the “Inferred Resources” rose steadily from February 2001 to May from 3.4 million tonnes at 1.36 g/t Au to 10.3 million tons at 1.51 g/t Au.

Drilling continued during 2001 with 54 drill holes and 8 946 meters and in October 2001 “Indicated mineral resources” were estimated to be at 15 million tonnes at 1.43 g/t Au based on 93 diamond drill holes, 12 988 meters of drilling and more than 5 710 gold assays.

In January 2002 “Indicated Resources” were at 16.6 million tonnes at 1.38 g/t Au and 2.7 g/t Ag. The resource estimations were based on the section method and a lower cut-off at 0.4 g/t Au and an upper cut-off at 10 g/t Au for exceptional gold values.

Only minor drilling was carried out in 2003 due to the lack of funding. In November 2004 the number of drill holes were 126 with a total of 19 131 meters of drilling and 11 000 consecutive gold- and silver assays.

3.5 Statement November 2004

In a press release on the 15th of November 2004 the following resources were released: “Measured Resources” were 16.1 million tonnes at 1.33 g/t Au in an open pit situation and the “Indicated Resources” were 9.5 million tonnes at 1,5 g/t Au in the same position from the surface to 200 meters level. In an underground position below the open pit the estimated “Indicated Resource” was 15 million tonnes at 1.5 g/t Au from 200-350 m. In addition 15 million tonnes at 1.5 g/t Au was estimated to be present from 350-450 meters level.

Total “Measured and Indicated Resources” were accordingly 25.6 million tonnes at 1.39 g/t Au. This qualified estimation was based on a very good knowledge about the gold mineralised shear zone to 200 meters level and on additional four drill holes intersecting the zone between 200 and 350 meters level.

3.6 Statement February 2007

Based on drilling results from 2005 and 2006 Lappland Goldminers AB presented (February 2007) a proven gold reserve of 20.75 millions of tonnes gold ore with an average grade of 1.43 g/t corresponding to 29.6 tonnes of gold (0.95 M tr Oz.). In addition the company presented an indicated mineral resource on the level below 200 meters. This resource was estimated to 15 millions tonnes gold ore and with an average grade of gold of 1.5 g/t corresponding to 22.5 tonnes of gold.

3.7 Statement October 2007

Based on further drillings down to 600 metres an updated estimate was made in October 2007, comprising mineral reserves and mineral resources estimated to 54 million tonnes. The average grade of gold is estimated to 1.17 g/t (dilution included) and the average grade of silver is estimated to 4.5 g/t. This corresponds to 70 tonnes of gold (2.25 M.tr.oz) and 247 tonnes of silver (7.94 M.tr.oz).

3.8 Test mining

In March 2005 test mining of 1000 tons of rock from the gold mineralized zone was carried out at Fäboliden in section 400S to get a representative sample for beneficiation tests. The average grade for the sample was calculated to be 1.53 grams of gold per ton based on diamond drilling results from nearby drill holes. However the average grade for the blast holes was almost twice as high or 2.91 gram of gold per tonne. The beneficiation test of minerals of 67 tonnes of rock was carried out at Stråssa. The average grade of 16 days beneficiation tests averaged 3.35 gram of gold per tonne, which was contradictive (in a positive way) to the results based on core drilling. Thus, the results from the test mining clearly indicate Au grades that are significantly higher compared to exploration core drilling results.

Figure 3.2 Test Mining Trench at Fäboliden in section 400 S during 2005.

3.9 Comparative analyses

The gold assays for cores from Fäboliden have been analyzed at ALS Chemex Laboratories in Toronto and the results from these analyses have been used when defining the Au grade of the mineral reserves and resources. The same cores have systematically and for comparative reasons been checked at OMAC Laboratories during the last two years. In total more than 300 tests have been performed indicating a strong linear deviation between the results where OMAC shows approx. 6 % higher values compared to ALS. With regard to this and to the results from the test mining the Au grades that are applied in this Feasibility Study are regarded as under estimates.

3.10 International Geological Consultants

In May-June 2001, Ilmar Gemuts, Mahopac, New York, evaluated Lappland Goldminers AB exploration program and strategy and provided new data and ideas for future gold, platinoid and diamond exploration in Lappland. Gemuts assessed the data available for Fäboliden and gave advice on drilling on the total length of 4 200 meters on the Fäboliden trend, geophysical IP-survey, engineering studies for open pit mining, metallurgical drilling and tests, monitoring of groundwater baseline studies and advice on purchasing of computers, large map printers and software.

Professor Peter Laznicka, Adelaide, South Australia, visited Lappland Goldminers in July 2006 to assess and discuss the geology of all active projects and to propose geological models for Fäboliden, Stortjärnhobben, Gubbträsk, Tjålmträsk and Knaften.

During 2004-2006 SRK Consulting carried out a major assessment in cooperation with Lappland Goldminers staff. The aim was to make the Fäboliden Gold Project in compliance with the regulations for mineral resource estimations according to NI 43-101. The assessment was in 2005 also involving grade verification at Fäboliden. The program involved an assessment of the following aspects of the project:

- Topography, landscape, access
- Regional and local geology
- History of exploration work in the area
- Audit of exploration work carried out by Lappland Goldminers
- Geological modelling
- Mineral resource estimation
- Validation
- Exploration potential and recommendations for additional work

The assessment was carried out by J-F Couture a Principal Geologist and Qualified Person of SRK Consulting, and others.

3.11 Specific Projects on Fäboliden

Bark (2005) produced a Licentiate Thesis on Fäboliden to be developed into a Doctorial Thesis in 2008. Fäboliden gold deposit is classified by Bark as a "Hypozonal Gold Deposit of Orogenic Type". The knowledge of this fact helps the geologists of Lappland Goldminers to discuss the Fäboliden gold deposit in seminars at home and abroad at international conferences.

Under the guidance of Dr. Lennart Malmkvist/Rantech AB a "Geogas trial" was carried out at Fäboliden in 2005. Isaksson initiated in 2006 a project to build a tectonic model for gold mineralizations in the Fäboliden region. This work was continued during 2007 with the aim to build a regional structural framework of importance for the setting of gold mineralizations on the Gold Line.

3.12 Exploration work at other gold deposits

There are a number of gold deposits held by Lappland Goldminers nearby to Fäboliden where the company is exploring the feasibilities for future gold ore production, implying possibilities to support the process plant in Fäboliden with additional ore. The present status for the exploration work is outlined below.

- In the *Knaften* project there are 49 diamond drill holes with a total of 7,800 meter. Additional drilling and shorter distance between profiles will be required to create a mineral resource. The best sections so far are 11 meters with 3.17 g/t Au in drill hole 96009, 7 m with 3.11 g/t Au or 5 m with 4.27 g/t Au in 200714 and 10 m with 3.20 g/t Au in 200707. The potential gold mineralization is open in all directions.
- In the *Gubbträsk* project there are 64 diamond drill holes and 8,400 meters drilled. The distance between the drill profiles is around 200 meters and more. There is an estimated “inferred mineral resource” for the project, which was completed last June. The inferred mineral resource for silver, zinc and lead is around 2.1 million tons with 10.93 g/t Ag, 1.21 % Zn and 1.08 % Pb. The inferred mineral resource for gold is about 0.7 million tons with 1.64 g/t Au. The mineralizations are two separate bodies which are open for continuation downwards and towards northeast and southwest. The estimation is performed from the surface and down to 100 meters below the surface.
- In *Tjålmträsk* there are 19 diamond drill holes with a total of 2,500 meters of drilling. Additional drilling and shorter spacing between profiles is required to estimate an inferred mineral resource.
- In *Stortjärnhobben* there are 52 diamond drill holes and 5,950 meters drilled. Additional drilling will be required to reach an understanding of the mineralization and the grades in the project as well as estimate the inferred mineral resources. Examples of interesting sections are:
  - Profile 110N, drill hole 200001 has 10 meters with 2.56 g/t Au.
  - Profile 240N, drill hole 97004 has 4 meters with 7.27 g/t Au, drill hole. 200508 in the same profile has 49 meters with 2.04 g/t Au.
  - Profile 340N, drill hole 97002 has 6 meters with 3.77 g/t Au.
  - Profile 456N, drill hole 200415 has 9 meters with 3.38 g/t Au.
  - Profile 500N, drill hole 200412 has 7 meters with 9.77 g/t Au.
- In *Sandviksträsk* there are 11 diamond drill holes with 1,491 meters drilled. Additional drilling is planned and with those results it may be possible to accomplish a closer interpretation of the mineralization.

In addition to the properties mentioned above there are a number of exploration projects held by other companies within a trucking distance that could be processed at the Fäboliden process plant.

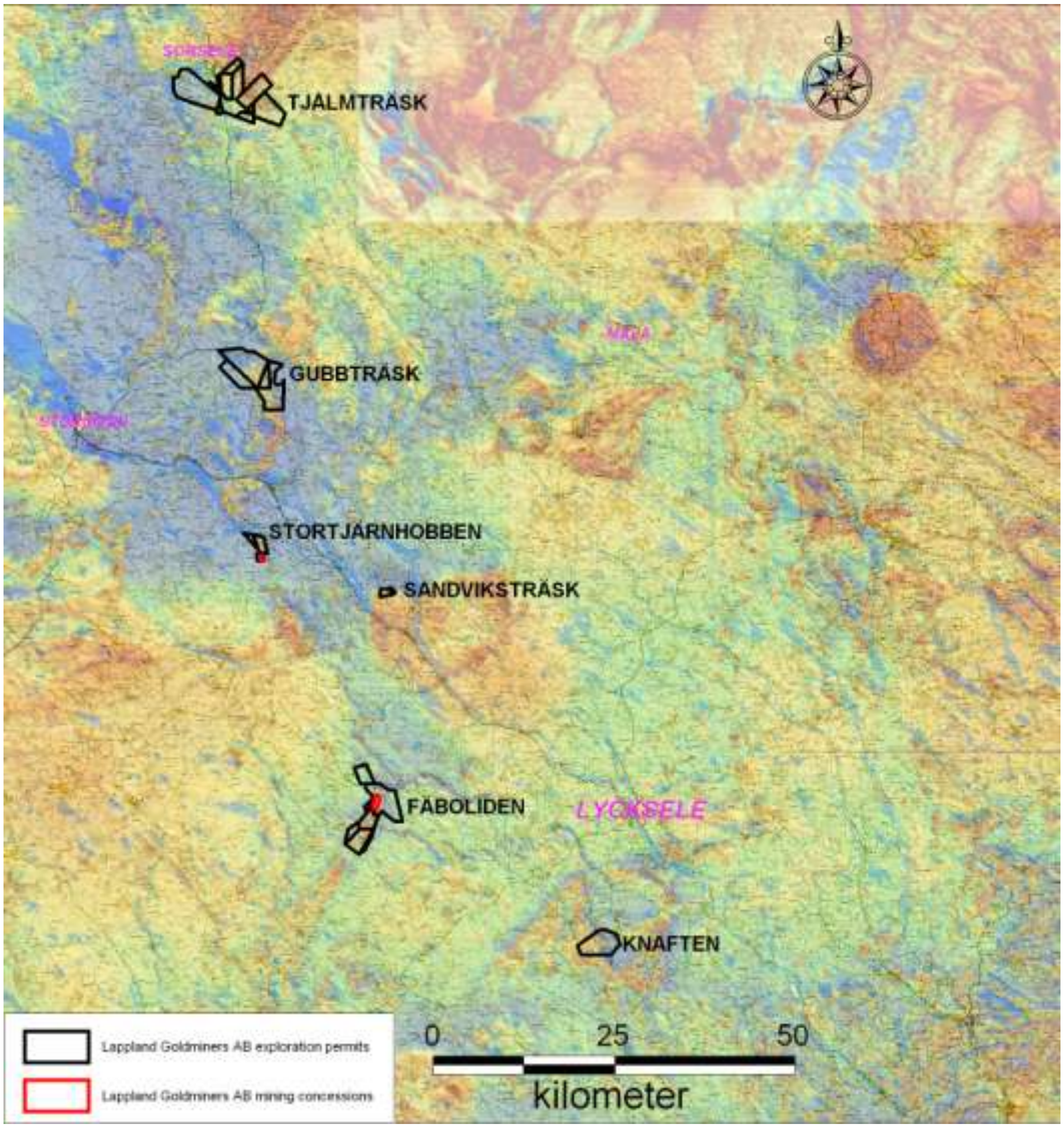

Figure 3.3 Location of gold deposits where Lappland Goldminers is carrying out exploration works

3.13 References

Bark, G., (2005)	Genesis and tectonic setting of the hypozonal Fäboliden orogenic gold deposit, northern Sweden. Luleå University of Technology
Carlson, L., (2005)	Fäboliden, haltverifiering, November 2005.
Gemuts, I., (2000)	Evaluation of Lappland Goldminers AB Exploration Program by Gemuts Exploration
Isaksson, H., (2007)	Fäboliden region, a tectonic model for gold mineralization- a working concept- Preliminary results from 2006.
Laznicka, P., (2006)	Brief geological report and suggestions to Lappland Goldminers, based on observations gained during Peter Laznicka´s stay in Lycksele (July 27 to August 10, 2006)

4. Mineral Reserves and Resource Modelling

4.1 Introduction

Several resource estimates, internal as well as external, have been prepared by Lappland Goldminers AB throughout the course of developing the Fäboliden gold and silver deposit. The principal purposes have been to guide further exploration work and to justify further investments in exploration drilling.

Based on the results from the diamond drilling as of February 2007 a new block model was elaborated. Since the feasibility study at this stage only contemplated the open pit part of the deposit, the limit for that model was set to include only material above the 250 m elevation.

In September 2007 the block model was up-dated, and included results from all drill-holes, reaching its maximum depth at the -150 m elevation which corresponds to roughly 600 m below surface.

4.2 Database Validation

The Fäboliden exploration database has been verified against paper records and the present internal quality assurance and quality control protocols have been investigated and found to be reliable for resource estimation purposes. Data representing drilling pre-2000 have not been subject to the same protocols, but since this comprise only a limited part of the entire database, it is considered to be of little concern.

In addition to what is mentioned above, the database has been subject to a standard check that includes:

- Check for duplicate collars.
- Check for twin holes.
- Check of collar location against topographic surface.
- Check for anomalous down-hole surveys.
- Check for overlapping assays.
- Check for zero-length assays.
- Checks for voids in assay sequences.
- Checks for holes bottomed in mineralized rock.
- Check for assay values successively the same.
- Check for assay spikes.

A few anomalies were noted, and forwarded to Lappland Goldminers, but the number and type of anomalies were within industry norms for databases of this size, and even if some anomalies turn out to be errors, they would have no material effect on the overall resource estimate.

The author believes that the exploration database has been prepared according to industry norms and is suitable for the development of geological and grade models on the basis of these statistical checks, and the checks of data entry discussed previously.

4.3 Resource Estimation

Database
The current resource model is based on a total number of 301 diamond drill holes, with an accumulated length of just under 56,000. Of those holes, 250 holes, comprising just less than 49,500 m hit the mineralized zone. In addition to the core holes, there are 11 reverse circulation (RC) drill holes, but due to uncertainties of sampling quality they were not considered in the resource calculation. The grade of gold however, was often slightly higher in the RC-holes than in the corresponding adjacent core drill holes. This difference should be addressed since sampling volume can be of importance to properly determine the grade distribution in deposits of varying grade such as Fäboliden. A specially designed drill program was executed to investigate suspected local variability near surface along three drill sections, 42 holes totaling 2,550 m were drilled and assayed at the time of data freeze (included in the above total).

Older drill holes generally have smaller diameter drill core, typically 36-39 mm, while more recent holes have larger diameter core, typically 42-49 mm. Old cores were sometimes assayed without splitting which means that no reference core is left for further investigations (certain holes 2001 and 2002). The sample rejects and assay pulps are still in storage if there should be need for further assays.

All assays, with exception for those from the vertical surface near holes, used for this resource estimation are made with Fire Assay, carried out by independent and most often certified laboratories including Filab in France and ALS Chemex in Vancouver. Check samples have been sent on a regular basis to SGS in the UK, XRAL in Mississauga and others. The vertical holes were assayed by accelerated cyanide leach using “Leachwell” reagent with AAS finish on 2.5 m sections at the ALS Chemex lab in Piteå.

All but a handful of the drillhole locations, as well as their starting azimuths and dips, have been surveyed relative to the Swedish RT90 system by a qualified surveyor contracted from Tyréns AB.

Down hole surveys have generally not been carried out on older holes (pre 2005). Since most of those holes are 100 m or shorter, and therefore typically don’t deviate much, this is not considered a problem. All holes drilled after 2005 have been surveyed for deviations in azimuth as well as in dip with an instrument that is not sensitive to changes in the magnetic field (Reflex Maxibor).

All in all, the exploration database for Fäboliden is found to be reasonably good for resource estimation at this stage, the only observation being the lack of down hole survey data for older drill holes, but since they generally are short this is of little or no consequence.

Solid body modelling
The basic assumption for modeling of the mineralized zone was that it would be mined with bulk mining in fairly large mining units. Outlines for the mineralized zone were interpreted on each drill section, following the 0.4 g/t Au outline. The outlines were later connected by triangulation to form a solid body. This body was cut against the bedrock surface, the dolerite dyke (diabase) was modeled separately and cut out from the solid.

The gold mineralized zone forms a very continuous corridor between drill sections. The main zone is split by a fairly thin (~20 m) dolerite dike, dipping gently towards the south.

There are often elevated grades of gold, occurring in narrow veins outside the mineralized envelope along both hanging wall and footwall. While the pit designs have only been based on the core mineralization, the footwall and hanging wall mineralization has been separately modeled to give indication as to those zones that will be mined and are mineralized and these have been included as Inferred Resources in the resource statement given below.

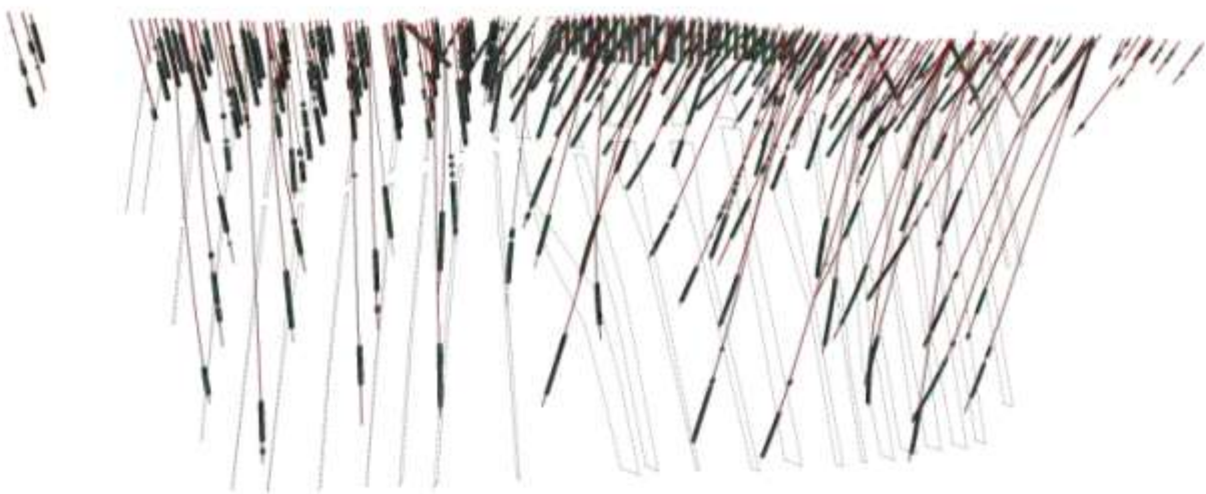

Figure 4.1 Fäboliden wireframe, overall length of mineralization approximately 1350m (view looking west).

Specific gravity data
The specific gravity database contains more than 500 density determinations from drill cores. These samples represent 11 different drill holes located in sections S400, S300, 0N, 50N and 250N. The density varies from 2.54 to 3.25 t/m^3, with an average of 2.83 t/m^3. The variations in density are due to the type of mineralization, which is a stockwork of quartz-sulphide veins, enclosed in a hydrothermal zone of alteration containing disseminated sulphides. The density thus varies depending on the rock type, vein density as well as the amount of sulphides.

The distribution of density determinations is by far too limited to create a varying density block model, and so is the actual number of determinations. It is recommended to carry out such determinations on a routine basis.

The average density of 2.8 t/m^3 was used globally and assigned to all blocks below the overburden surface in the absence of a variable density.

A density of 1.80 t/m^3 has been used for the overburden above the fresh rock interface. This estimate is commonly used as an average for Northwest Sweden.

Statistics

The normal assay section length for Fäboliden is 1 m, approximately 95% are exactly 1 m, 1.4% are shorter and the remaining 3.6% are longer than 1 m. For the purpose of resource evaluations, all sections from within the interpreted mineralization must be composited to have equal weight in the interpolation of block values.

Assays from within the mineralized envelope were flagged and composited to different lengths to evaluate a proper compositing regime. To better honor the variations within the zone it is preferable to have shorter composites and thus avoid "over smoothing" the original data. After testing 2.0 m, 2.5 m and 4.0 m composite lengths respectively, the 2.5 m length was selected.

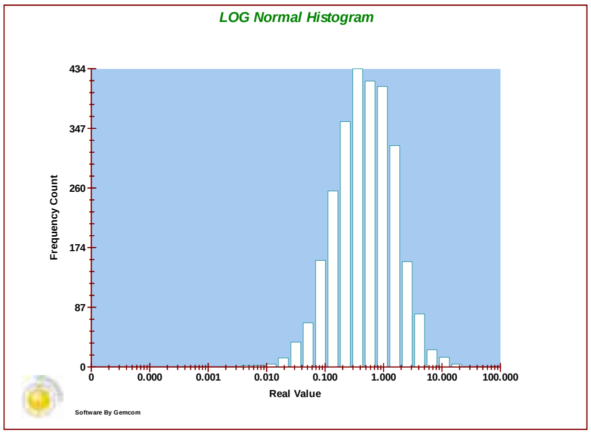

Figure 4.2 Histogram of 2.5m composites of Au-assays, showing a typical log-normal distribution.

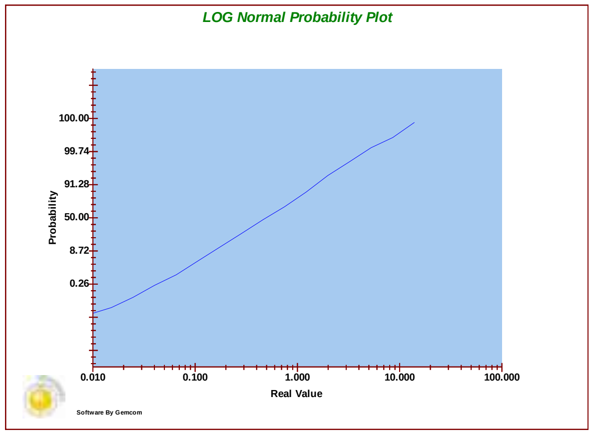

Figure 4.3 Cumulative probability plot of 2.5 m composites of Au-assays showing that 2.5 m is sufficient to normalize data.

Precious metals top cutting

The previously used top-cut (5.5 g/t) was considered to be overly conservative since a +10 g/t population was observed spread around the southern sectors of the mineralization. Following this analysis the top cut was removed for the first two grade interpolation runs where close spaced high grade could interact. On the last run, which used a much larger search ellipse, the data was cut to 10 g/t.

No top-cutting was applied to silver.

Block modelling and grade estimation

The block model consists of 15*15*15 m (width*length*height) blocks, oriented along the Swedish co-ordinate system. The selection of block size was principally based on average drill but also considered the selective mining unit in the planned open pit.

Since the mineralized zone is slightly curved it was not possible to adopt one single set of search directions for the interpolation, instead the orebody was split up into three different domains or areas, each with its own set of search directions.

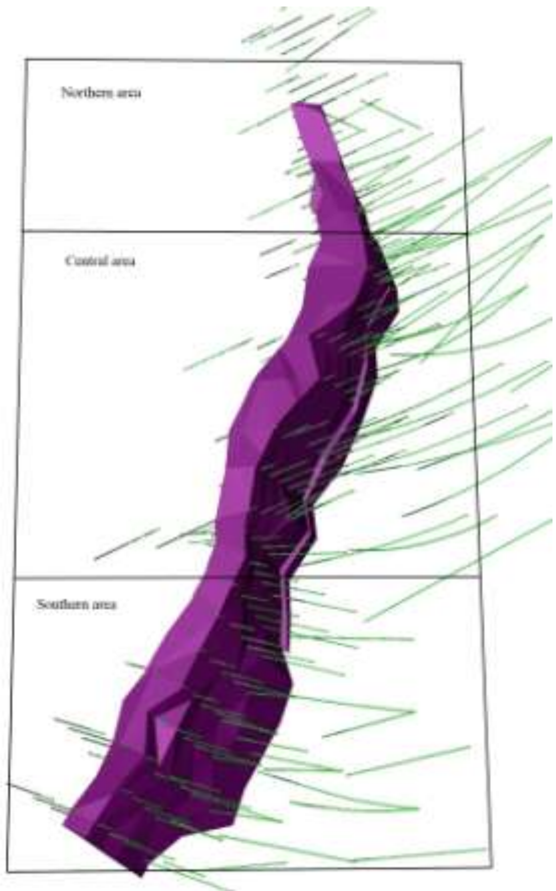

Figure 4.4 The interpreted mineralization at Fäboliden, showing drillholes and interpolation domains.

The search directions for the three areas are:

Northern area: strike 15º, dip -15º
Central area: strike -22º, dip -25º
Southern area: strike -15º, dip -30º

Grade estimation

Grades have been estimated into the block model using the inverse distance weighting to the power of one method (IDW1) with the use of three increasing search ellipses. These ellipses were orientated to the strike and dip of the three estimation domains and increase in overall size as follows:

Run	Min. numb. of holes	Maximum numb. of hits	Search ellipse size X	Y	Z
1	2	12	5	50	50
2	2	12	10	80	80
3	1	12	40	200	200

Table 4.1 Grade estimation

The three runs were completed separately for gold and silver and for the three geological domains.

Mineral resource classification
Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories respectively.

A resource must have reasonable potential to be economically extracted at a profit to be reported as such. A resource increasing in confidence from inferred through indicated to measured as the grade, tonnage and other factors are known with more reliability.

The global mineral resource has been classified into the above mentioned categories. The classifications are based on interpreted geological continuity seen in the drillhole plots as well as on geostatistical evaluations. A nominal drill section spacing of 50 m has been followed throughout the property; this is considered sufficient considering the size and continuity demonstrated for the type of mineralization.

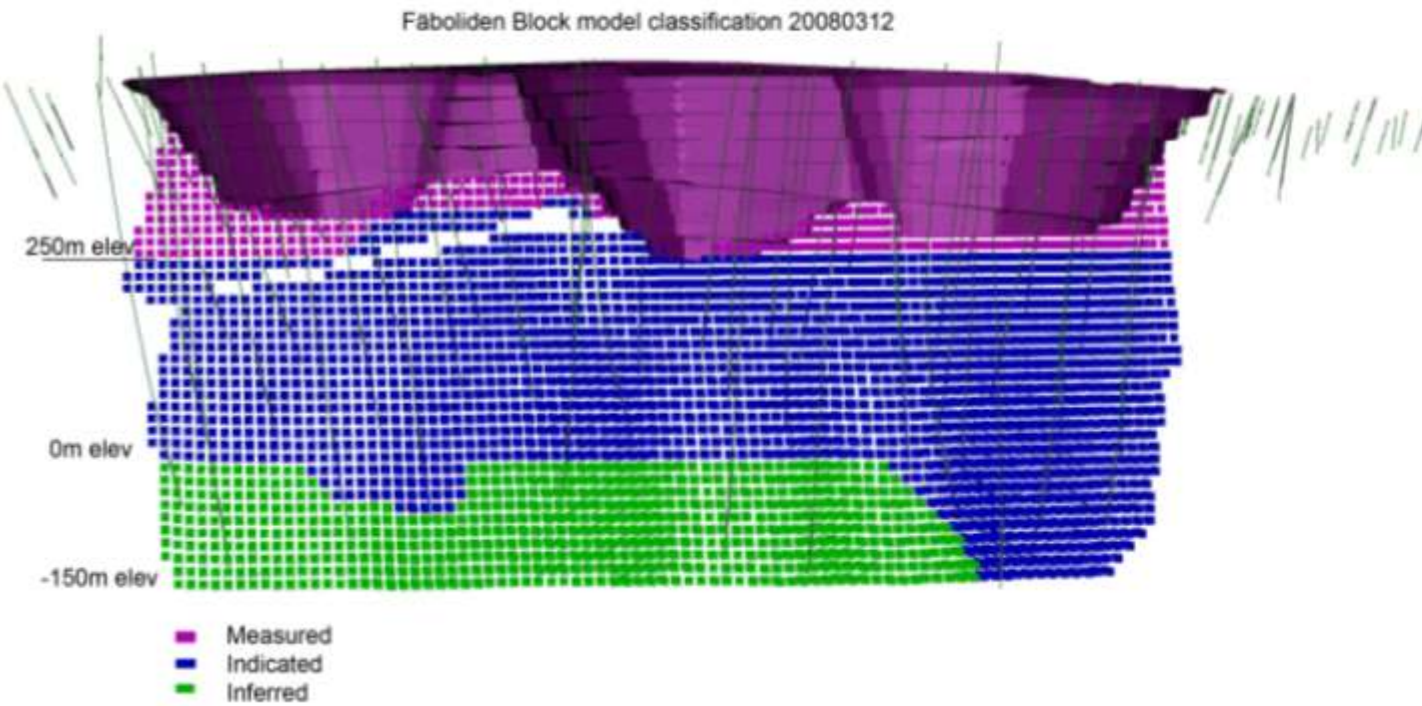

Figure 4.5 Block Model Classification

Mineral resource validation
Both global and local grade estimates were checked for appropriateness. Original gold grades were compared visually with block grades on a section by section basis. The interpolation methodology was found to appropriately reflect the general dip and strike of the interpreted gold trends, and to appropriately correspond to proximal borehole composite gold grades.

Mineral resource statement

The mineral resource estimated in March 2008, with a 0.4 g/t cut-off applied, is summarized in the table below.

		Tonnage (t)	Au g/t	Ag g/t	Cont. Au (g)	Cont. Ag (g)	Cont. Au kOz	Cont. Ag kOz
+250 m	Measured	24 961 000	1.26	3.27	31 409 000	81 688 000	1 010	2 626
	Indicated	1 435 000	0.97	1.95	1 398 000	2 798 000	45	90
250 m to -5 m	Indicated	30 500 000	1.23	3.69	37 376 000	112 604 000	1 202	3 620
-5 m to -150 m	Indicated	4 120 000	0.88	3.06	3 609 000	12 606 000	116	405
	Inferred	7 057 000	1.42	4.19	9 991 000	29 576 000	321	951
Footwall/Hanging wall	Inferred	2 470 000	0.61	1.49	1 524 000	3 670 000	49	118
Total		70 542 000			85 306 000	242 943 000	2 743	7 811

Table 4.2 Mineral resource statement made by Thomas Lindholm GeoVista AB

Compared to the statement in October 2007 this means that:

- The measured and indicated mineral resource (inferred resources not included) is increased from approx. 54 Mt to approx. 60 Mt.
- The average Au grade for the mineable reserve (i.e. the open pit with approx. 23 Mt ore) has been slightly decreased from Au 1.3 g/t to 1.2 g/t (see chapter 5.4). This is a result of a recent updating of the block model considering additional results from core drillings.
- The previous resource statement built mainly on a mathematically calculated classification, principally using the search radii for assignment of blocks to each class. The present statement is based on a more conservative approach that, in addition to the search radii also takes into account the demonstrated continuity of the mineralized zone. It is felt that the present classification better accounts for the ability to predict grades and thicknesses in each class respectively, than the previous one.

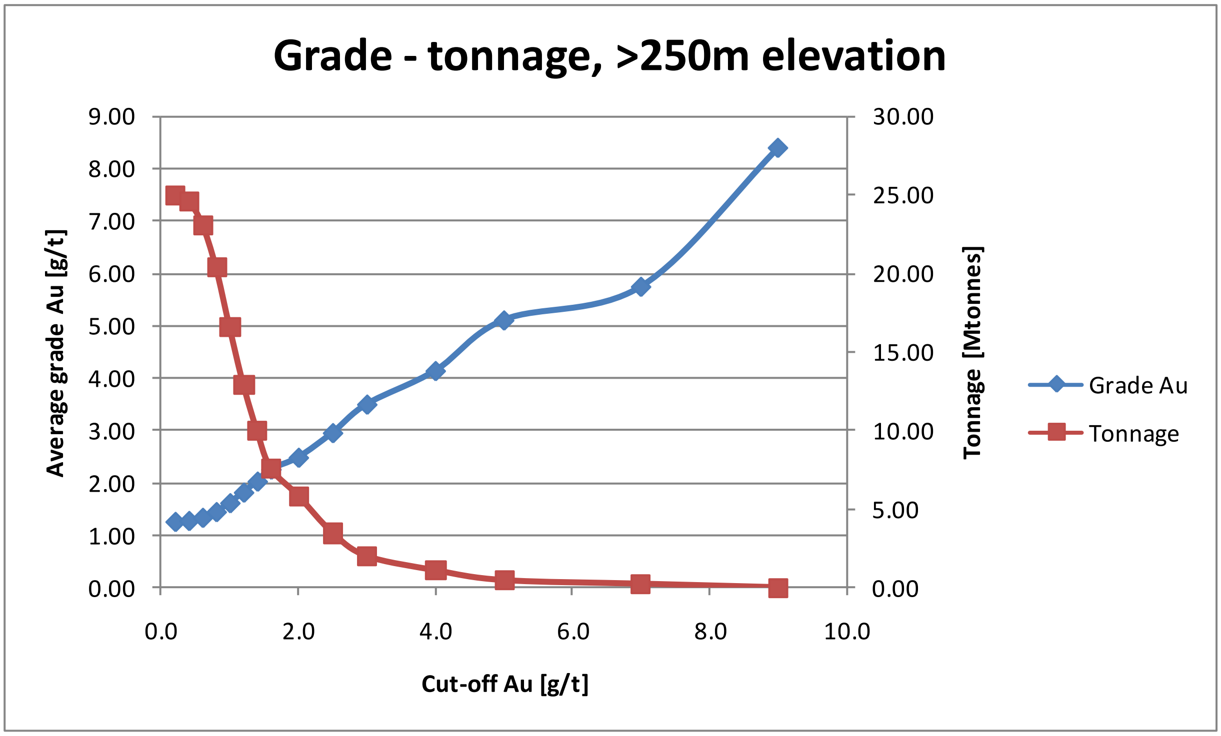

Figure 4.6 Fäboliden global grade and tonnage curves to approx. 200 m depth

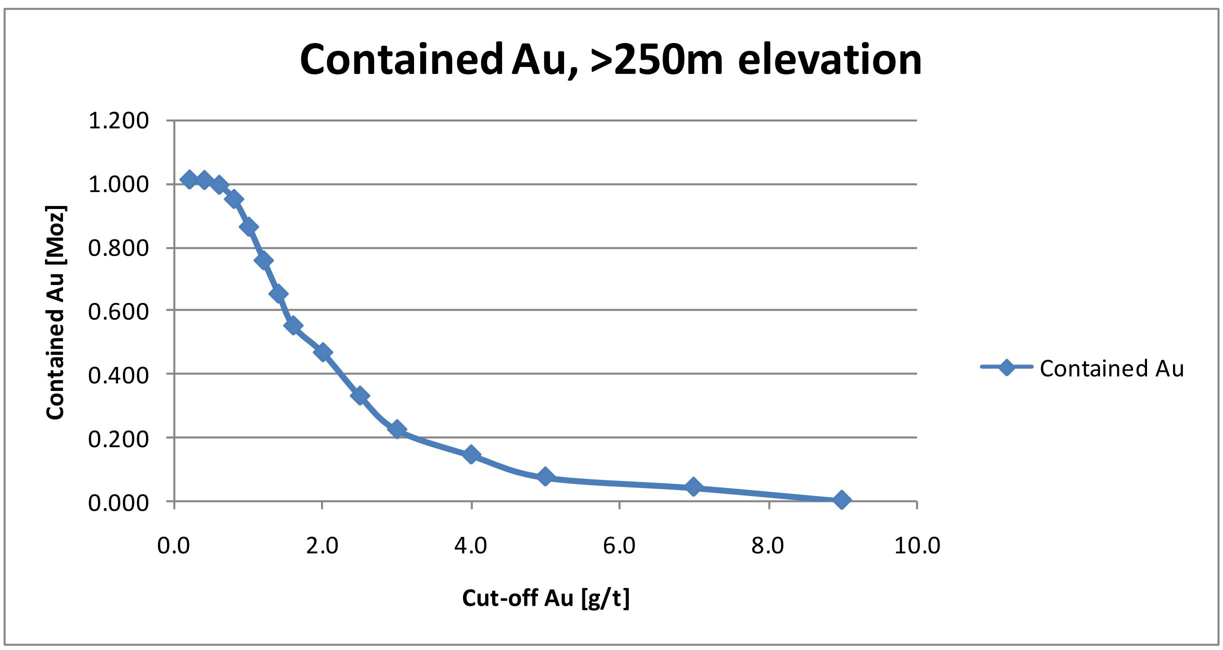

Figure 4.7 Contained MOz Au versus cut-off in Fäboliden to approx. 200 m depth

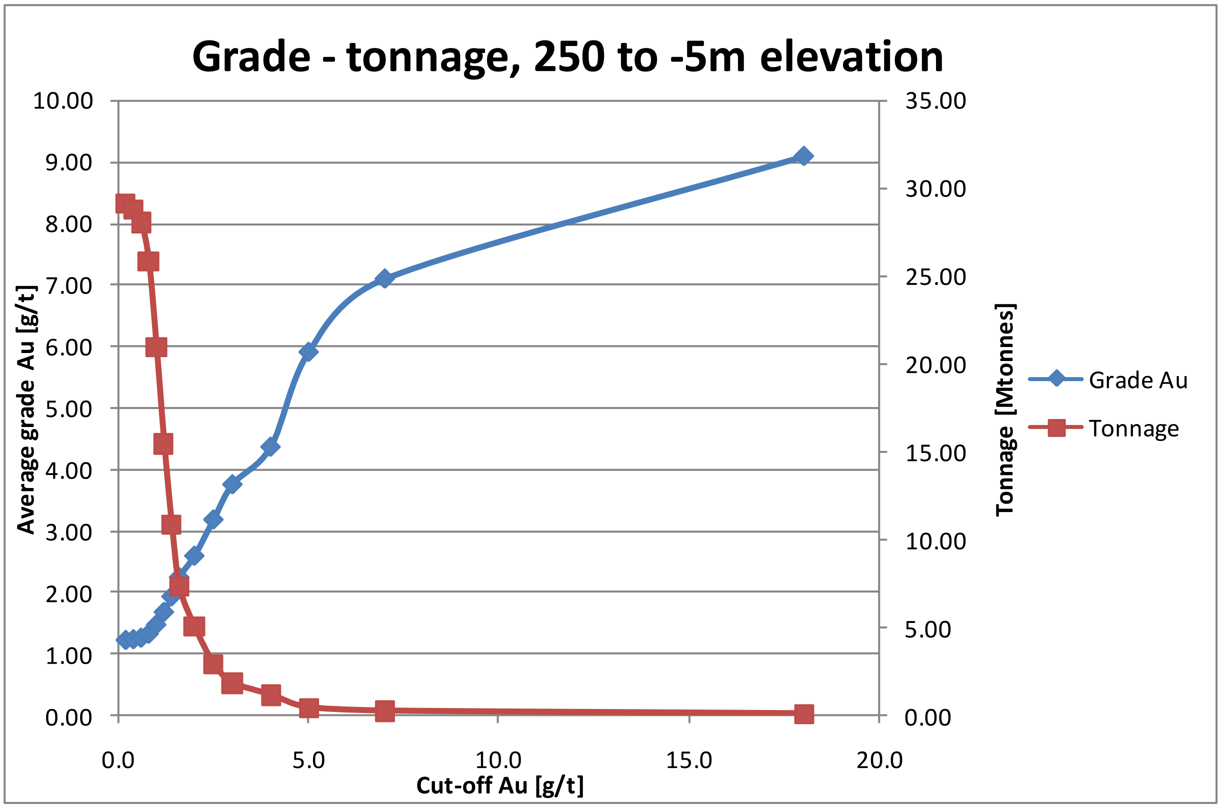

Figure 4.8 Fäboliden global grade and tonnage curves, 200-450 m depth

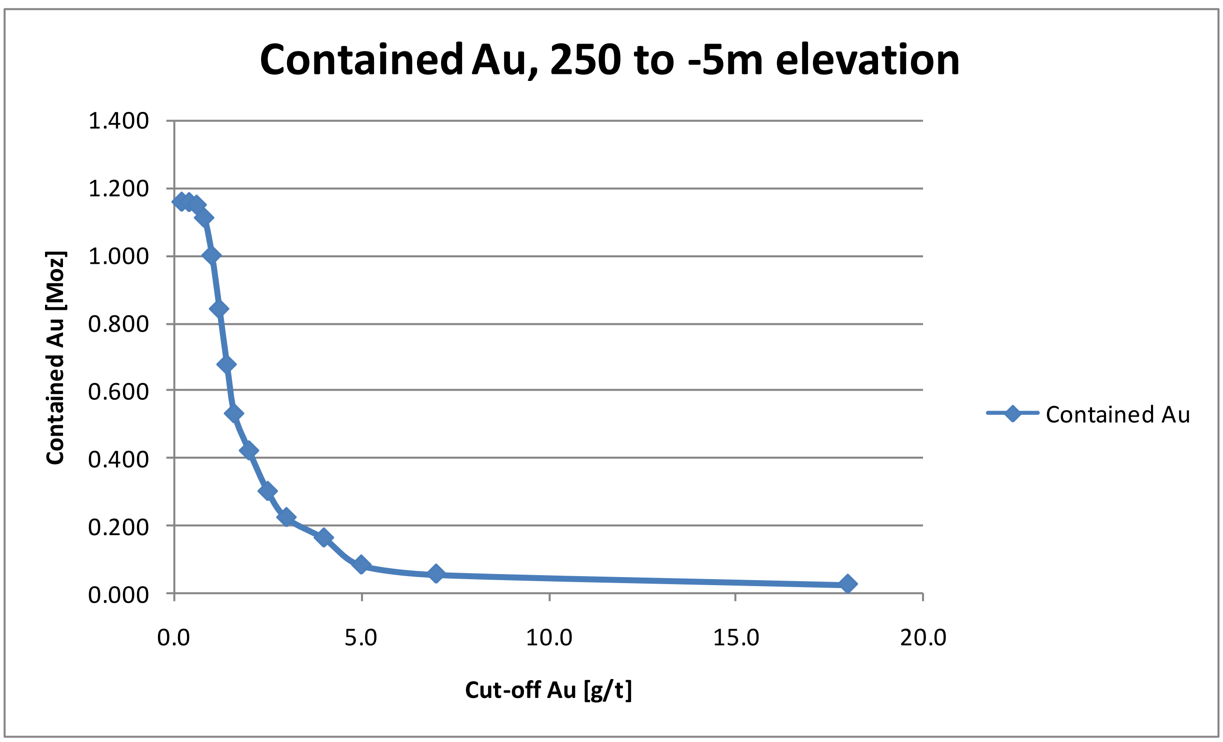

Figure 4.9 Contained MOz Au versus cut-off in Fäboliden, 200-450 m depth

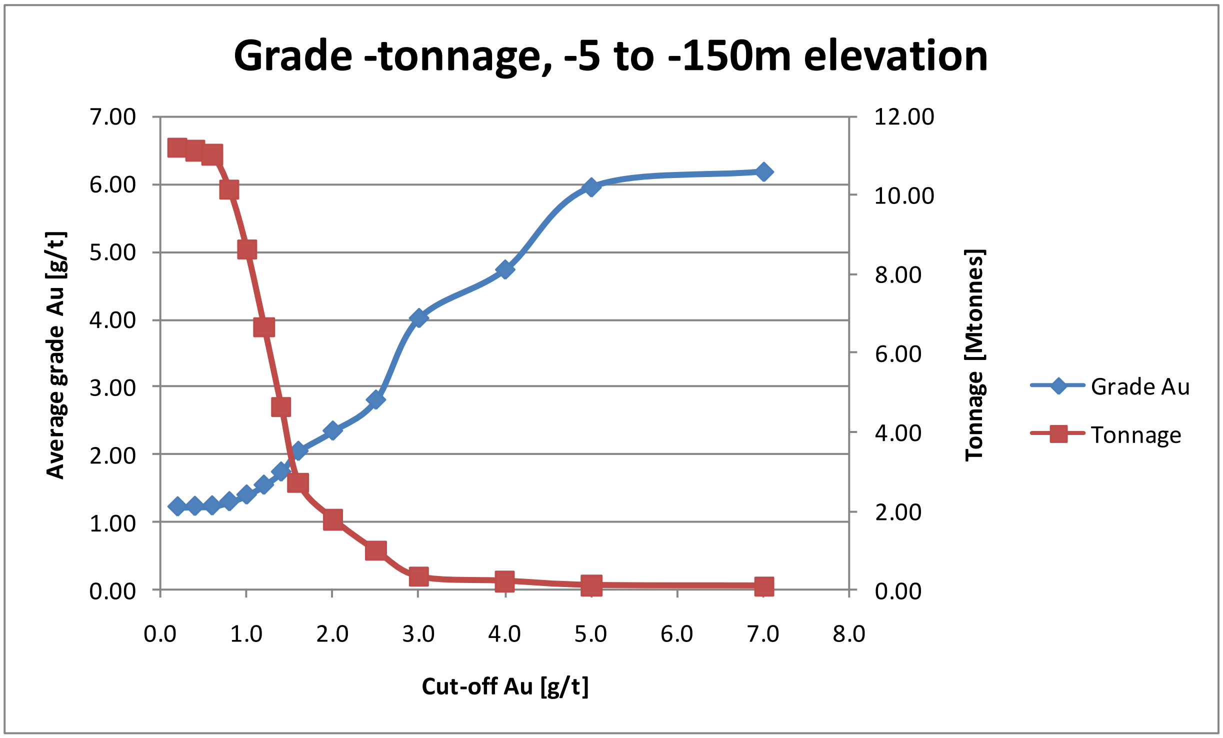

Figure 4.10 Fäboliden global grade and tonnage curves, 450-600 m depth

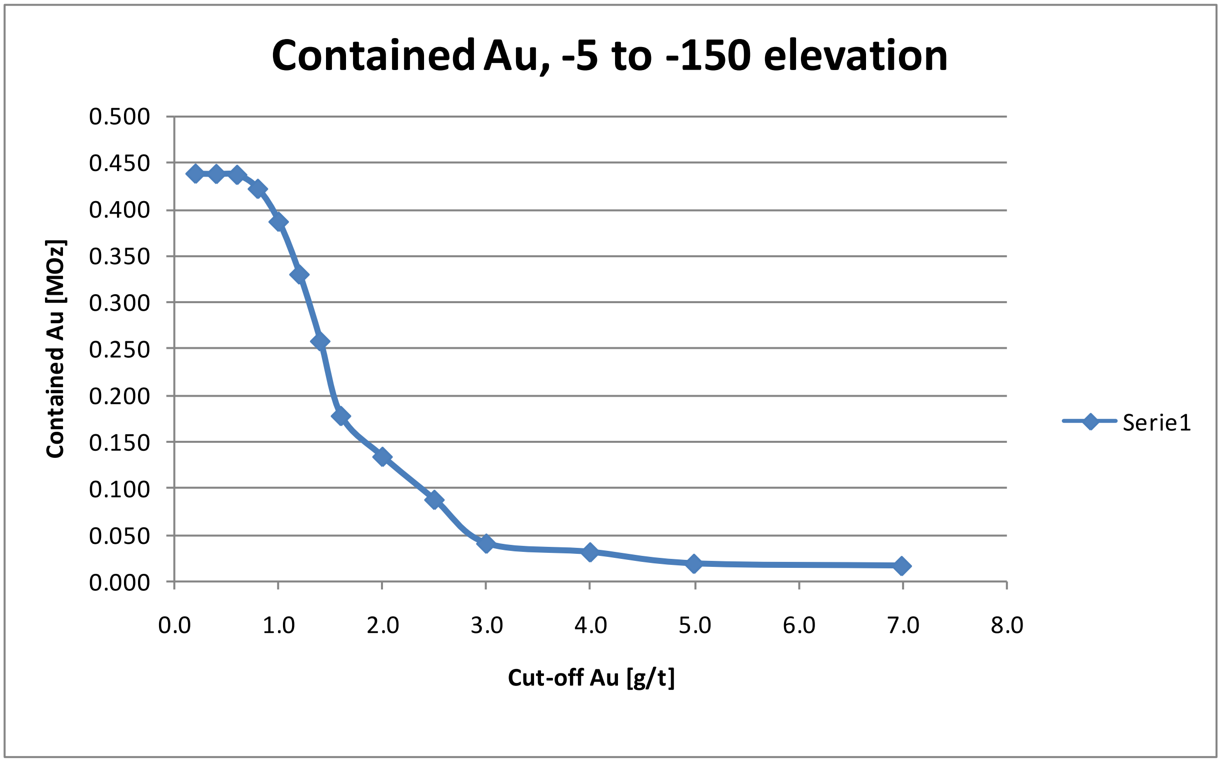

Figure 4.11 Contained MOz Au versus cut-off in Fäboliden, 450-600 m depth

Mineral reserve statement

"A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined."

The above text is an excerpt from the CIM Standards On Mineral Resources And Reserves (Canadian Institute of Mining, Metallurgy and Petroleum). It defines how to convert a Mineral Resource to a Mineral Reserve. The Mineral Reserve is reported in chapter 5.4.

5. Mining-Open Pit

5.1 General

The mineralization is initially planned to be mined in an open pit to a level of approx. 250 m above the sea level.

The geometry of the ore body as well as its location on top of a hill, makes it highly amenable to open pit mining. Assessment of ground conditions and core suggests that surface will enjoy good mining conditions in relatively strong, competent rock. The observed conditions suggest that an optimistic approach may be taken with pit wall angles. A tightly controlled bulk mining method in conjunction with selective techniques will be applied, however it will be imperative that good grade control is exercised to minimize ore losses and waste dilution.

The major geotechnical consideration for mining at Fäboliden is the strong foliation set running parallel to the strike of the orebody. The foliation may cause delamination of walls along the strike of the orebody which may affect pit wall angles, while it could be a propensity for blasting to break to a laminar contact, reducing dilution

Water is associated mainly with fracturing and crush zones, and apart from some initial flows in these areas, mining conditions are expected to be reasonably dry. This has positive implications for rock strength and mine design parameters. Also, due to the orebodies' location on the top of a ridge, the area is well drained and any precipitation will flow away from mining activities, reducing problems associated with water pressure and water ingress into the pit.

5.2 Pit optimization

Slope design criteria for initial studies of the open pit were based on experiences from Boliden's Aitik open pit mine.

Later, a rock mechanical slope stability study was prepared by Dr Jonny Sjöberg (PhD), Vattenfall Power Consultant (Rock Mechanical Assessment for the Feasibility Study of the Fäboliden Open Pit, December 2006, enclosed in Appendix 1(5)). This study confirmed that the initially used slope design criteria were justified.

The purpose of pit optimization is to find the most valuable pit from an economic point of view. The optimizations in Fäboliden have been done by SRK Consulting (UK) using Whittle version 4. Whittle is a product of Gemcom Software International. Data input to Whittle is a block model together with economic and technical parameters. Most of the used parameter values can be found in the table below. Note that initial investments and rehabilitation costs are not included in the optimization.

Densities	Overburden: 1.80 ton/m^3 Mineralized rock: 2.80 ton/m^3 Waste rock: 2.80 ton/m^3 Dolerite: 2.80 ton/m^3
Maximum mining rate	20Mton/year base case
Optimization cut-off	Marginal cut-off
Mining dilution	0%
Mining recovery	100%
Average height of waste dump	10 m
Top of pit location	450 m above sea level
Overall slope angles	Overburden: 20 degrees, Solid rock: 54 degrees
Selective mining unit	15*15*15 m
Maximum processing capacity	4.6 Mton/year
Process recovery - Gold	75% (0.50-0.60g/t) 78% (0.60-0.75g/t) 80% (0.75-1.25g/t) 85% (1.25-1.45g/t) 87% (1.45-1.75g/t) 90% (1.75-1.95g/t) 91% (1.95-2.95g/t) 93% (>2.95g/t)
Process recovery - Silver	50%
Exchange rate	6.5 SEK=1.00 USD
Overhead costs	Company: 3.0 MSEK/year, Mine site: 5.0 MSEK/year
Vertical hauling cost	0.20 SEK/ton/10 m
Processing cost	42 SEK/ton
Refinery charges	USD 5 all accounted from Au sales
Au price	135.033 SEK/g (600 USD/oz) base case
Ag price	2.476 SEK/g (11 USD/oz) base case
Discount rate	7%

Table 5.1 Costs and other parameters used in open pit optimization.

The parameters tabulated above have been determined by Lappland Goldminers AB. The slope angles are based on the rock mechanics study by Vattenfall Power Consultant. Process costs and expected recoveries are based on testwork performed by Msc. in Chemical Engineering Jan-Erik Sundqvist, PhD. in Mineral Processing Nils-Johan Bolin and MINPRO. The costs of mining ore and waste respectively are based on in-house studies as well as on comparison with those of Aitik open pit mine.

During the optimization a series of nested pits are created by varying a “revenue factor”. These pits can afterwards, in the analysis phase, be used to simulate different mining situations.

Pit optimization is an iterative process. For example, until the first optimization is done it may not be clear where the haul roads will be. If this is not known one does not know where to lay back the slopes to allow for the haul roads.

Whittle contains two extreme mining sequences (schedules) named ”worst case mining” and ”best case mining”. In worst case mining, or flat mining, each bench is completed before starting the next. In best case mining, or incremental mining, each shell, in the sequence of nested pits, is completed before starting the next. Worst case mining gives a lower limit for the net present

value of a pit and best case mining gives an upper limit. It is also possible for the operator to specify his own mining sequence using the nested pits.
From the sequence of pits, a few pits are chosen as a platform for the mine design. How to select suitable pits, from the sequence of pits, is primarily a question for the management.

The mineralised zone in Fäboliden has an approximately north-south extension. The block model of the mineralised zone was created by Thomas Lindholm GeoVista AB, Sweden. The size of the blocks was set to 15*15*15m to reflect the drilling density as well as the selective mining unit.

5.3 Pit Design

The resulting "optimized pit" was then subjected to design with consideration to equipment sizes etc. The design was made by Carl Boman, retired Chief Surveyor of Mines with a background in Boliden AB.

The pit was designed with 15 m high benches and with catch berms every second bench.
The berms has a design width of 14 m and it is estimated that this will give an effective catch berm width of >11 m for 90% of all cases.

Haul roads were made with a design width of 33 m and an inclination of 10%. The width usable for traffic will be 23 m and this will allow 2-way traffic for trucks with load capacities up to 180 tonnes. Over the lowest three benches the ramps have been designed for one-way traffic only, 20 m wide. In order to obtain the practical widths stated above for catch berms and haul roads it will be necessary to minimize backbreak at the crests by some form of cautious blasting.

Bench height	15 m
Slope design (both hanging wall and footwall)	Interramp angle 56°
	Berm design width 14 m (90% >11 m)
	Double benches (30 m between berms)
Haul roads	Width 33 m
	Width lowest 3 benches 20 m
	Inclination 10%
Minimum width pit bottom	30 m

Table 5.2 Design Criteria

The orebody is markedly elongated in the North-South direction, with some better grades in the upper/southern part and in the deeper/northern part and the lowest grades in the central and upper/northern part. The pit was designed in four phases, with an initial preproduction pit to provide construction material, principally for the tailings management facility. The preproduction pit reaches down approximately 90 m to the 385 m level. The haul road starts at the center of the pit and follows the footwall side towards the north and switchbacks on the hanging wall to 320 m level in the south end. The major part of the road is here designed 33 m wide in order to give room for two way traffic.

The final bench in all three cones is designed with no ramp access, assuming that a backhoe excavator will be loading the material from the last bench. Three benches have 20 m ramp and then the single ramp on 320 goes from 20 m to 33 m. The inter ramp is designed at 56 degrees.

Bench height is 15 m and not double benched to 30 m. The total volume will more or less be similar in both cases anyway. The berm width is 7.5 m.

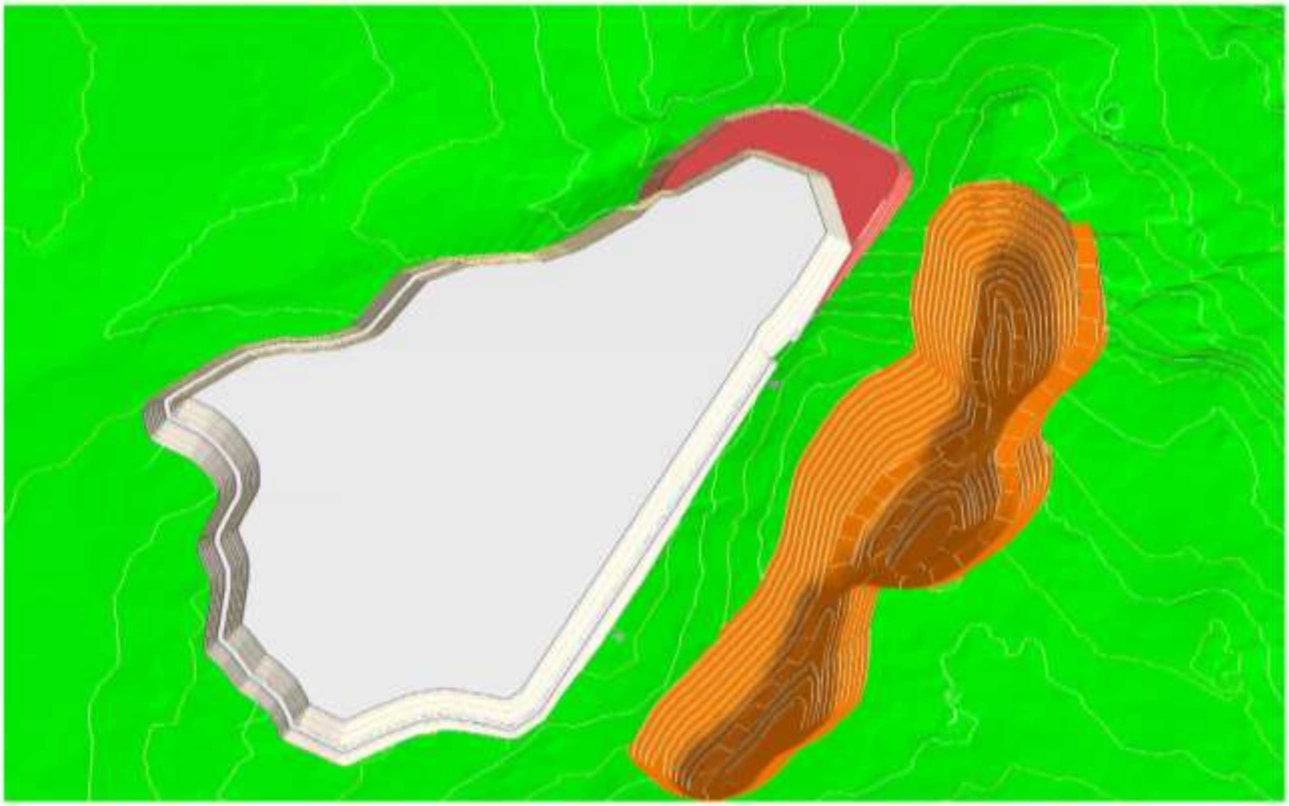

Figure 5.1 Open Pit Lay-out to the right, waste rock deposit to the left

5.4 Mineable ore reserve

The mineable reserve was calculated by Thomas Lindholm GeoVista AB from the pit layout. A cut off of 0,4 g/t Au was used. The reserve is summarized in the table below:

	Ore Tx1000	Au g/t	Ag g/t	Waste Tx1000
Total	22 897	1.20	3.12	84 973

Table 5.3 Mineable ore reserve open pit

The waste dilution by over-breakage has been estimated by adding a 1.5 m thick skin of material around the mineralized zone in the open pit. This corresponds to an additional 8.5% of material being classified as ore and thus trucked to the mill. The grade of the diluent has been estimated to 0.25 g/t by manually checking the zone of over-breakage on all drill-sections, and calculating an average.

5.5 Production Plan

The production plan was made by GeoVista AB. Ore processing in the mill has been set to 4.6 Mt/year. Preproduction of 10 Mt waste stripping is assumed to be carried out during one year before start of processing. This preproduction period has been considered advantageous since the operational stripping ratio will be lower and less equipment will be needed. Total annual mine production, ore + waste will be approx. 20 Mt (strip ratio 1:4).

The production plan for the open pit is summarized in the table below.

	Mine production					Mill feed		
Year	**Ore Tx1000**	**Au ppm**	**Ag ppm**	**Waste Tx1000**	**Soil Tx1000**	**Ore Tx1000**	**Au ppm**	**Ag ppm**
0	13			9 570	3 449	15		
1	4 300	1.16	3.29	15 000	4 923	4 600	1.09	3.11
2	4 200	1.49	2.87	15 400	1 437	4 600	1.39	2.72
3	4 200	1.22	3.39	15 600	2 284	4 600	1.15	3.20
4	4 200	1.20	3.21	15 800		4 600	1.12	3.04
5	4 200	1.35	3.77	15 400		4 500	1.26	3.55
Total	21 100	1.28	3.30	86 800	12 100	22 900	1.20	3.12

Table 5.4 Production Plan

The Au-grade will be as highest the second year of production, approx. Au 1.5 g/t. The graph below shows details of the grade variation.

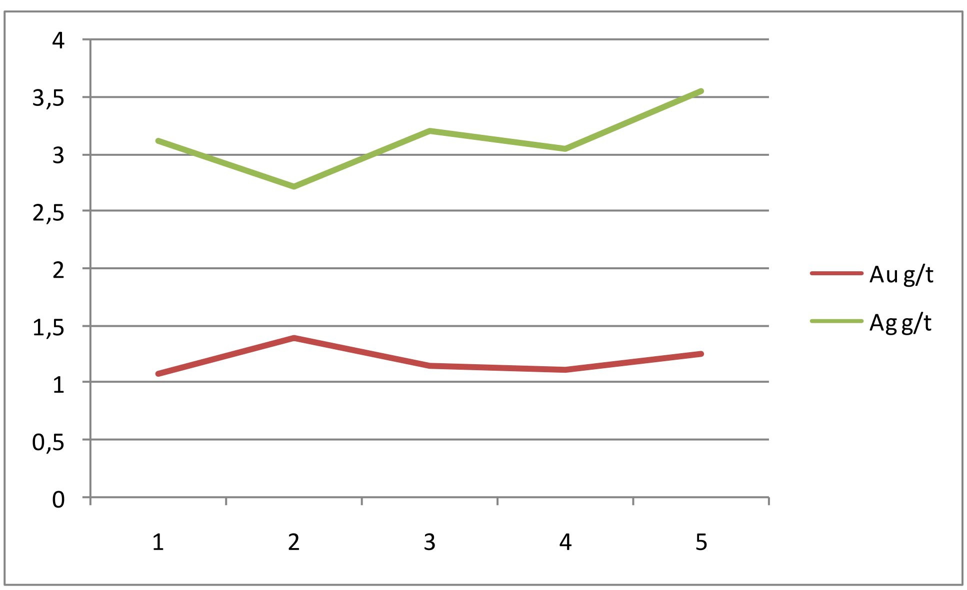

Figure 5.2 Grades Au and Ag in mill feed per year

5.6 Waste Dump Design

A waste rock deposit has been designed to contain <76 Mt. It will be built up by three benches with the material deposited in a natural angle of repose (slope 1:1).This will response in an average slope of 1:2.2 since each bench will be set back 20 m in relation to the prior.

In addition a tailings reservoir has been designed and will at its final stage contain about 50 M(m^3) tailings of which 1.5 M(m^3) consist of HR (high risk) tailings. Except for the tailings, HR waste rock will constitute <17 M(m^3).

5.7 Mine Operations

Production planning and grade control system

Efficient quality (grade) control procedures will be essential for the success of the operation. Four different rock qualities will have to be separated;

1. Ore to the mill
2. Diabase to stockpile, potential raw material for production of mineral wool
3. Waste rock, potentially acid generating to deposit within the tailings area (High Risk, HR)
4. Waste rock, non acid generating to waste dump west of the pit (Low Risk, LR)

The mine will have an integrated computerized system for production planning and quality control. Planning of blasts will be done directly on the computer and drill hole coordinates will then be fed into computers in the drill rigs. The drill rigs will use GPS for correct hole positioning.

Diabase can be identified visually when loading and a large part of the non acid generating waste can be identified from diamond drill holes. Classification of the other rock qualities will be based on sampling and assaying of cuttings from the production drill holes. The block model will be updated with the assay results and borders between different qualities established. After blasting, boundaries for loading of different qualities will be defined with regard to the redistribution of material that occurs during the blasting. Loading boundaries will thereafter either be marked directly on the rockpile or in the computerized production control system where boundaries together with the position of the loader's bucket, defined with GPS technique, are shown on a graphical display in the loader.

At least one sample per production drill hole will be assayed, which means a total of 30-40 samples per day. Also the trucks are equipped with graphical displays with which they will be directed to the correct destination with regard to the rock quality carried. Mine surveying are done with total stations and locations of bench crests, toelines etc. are fed directly into the planning system. The planning and control system also acts as a follow up system for production statistics and performance data.

Drilling

Production drilling will be done with crawler-mounted DTH (down the hole hammer) drill rigs. Waste rock will be drilled with 203 mm holes while in ore the drill hole size will be 165 mm in order to get a good control of rock size distribution in the mill feed.

It is estimated that 4 drill rigs of type Atlas Roc L8 or similar will be sufficient for the required drilling. Drilling will either be carried out on a dis-continous 3-shift basis or on a continuous 2-shift basis. A total of 13 persons will be required for operating the drilling.

Blasting
The total blasting operation is planned to be contracted to a supplier of explosives. The contractor will get a space within the industrial area for handling and storing of ingredients for the explosive. A slurry-type explosive and non-electric blasting caps will be used. The final mixing of the slurry is done in the charging truck when it is pumped down into the drill hole and it is first at this time that the chemical components form an explosive. Blasting will only be done daytime at scheduled hours. Size of the blasts will vary from 200 tonnes up to 200 000 tonnes of rock.

Figure 5.3 Drilling blast holes for test mining in Fäboliden

Loading and hauling
Loading and hauling will be done 3 shifts per day, 7 days per week.

The planned main loading units are a diesel-hydraulic shovel, type Komatsu PC 4000 or similar, and a large wheel-loader type Komatsu WA-1200 or similar. Both these units have a bucket capacity of approx. 20m^3 and an estimated loading capacity of around 10 Mt/year each.

For hauling the ore and waste rock to the crusher different deposit areas diesel-electric trucks, type Komatsu 730-E or similar will be used, performing a load capacity of 180 tonnes. It is estimated that 6 such trucks are needed.

Auxilliary equipment
On the waste dumps, one track dozer, type Komatsu D275AX-5 or similar, will be used to keep an even surface of the dumps. This will make it possible for the big trucks to keep a high speed and dump their loads without any delays, due to uneven surface or blockage of waste rock piles. The track dozers may also be used for road construction work from time to time.

One grader, type CAT 16H or similar and a small wheel loader will be used for road maintenance work. The wheel loader will also be used to clean around the shovel and to assist the track dozer with pushing and cleaning on the dumps.

A water truck will be required to control the dust from the haulage roads, deliver water to the drill rigs and spray water on the muck piles. Even if the drilling and blasting operation is performed in a proper way, there will be oversize material, which has to be treated in the mine.

Road maintenance
Road construction and maintenance have a great impact on the operation. The big trucks have a weight fully loaded of about 320 tonnes, which has to be considered when roads are constructed.

In the mine, the ramps shall be built with about 1.0 meter of blasted rock. If the road is constructed on poor ground, this layer may be accordingly thicker. On top of this, a layer of crushed rock, "base course", consisting of 50 - 150 mm material shall be spread to even out irregularities. This layer shall be about 30 cm thick. On the surface a 15 cm layer of finer crushed material, "surface course", consisting of 5 - 50 mm material shall cover the haul road. A slight crowning of 1 to 2 % may be adapted.

Road maintenance includes dust suppression with water trucks, adding of surface course material and grading. In the winter clearing the road from ice and snow has to be carried out as well as sanding the road. The road material can probably be produced from suitable waste rock stripped from the mine.

Lightning
In general, all machines will be equipped with sufficient light to be able to work without any problem around the clock. Especially crawler mounted equipment shall have extra lights around the machine to get good working conditions. In certain places, there will be stationary lights, such as the fuel station, permanent road crossings, etc. On the waste dump there will be mobile light towers, which can be moved as the dump progresses.

Maintenance services
Maintenance has to be planned and performed in a very efficient way in order to obtain the required high availability on the equipment. To ensure this, repair and maintenance contracts will be set up with the suppliers of the major production equipment. However, some service on the major equipment, for example tire work, work on loading buckets and on the dump bodies of the trucks as well as service on the auxiliary equipment will have to be done by own personnel. A well accomplished preventive maintenance schedule is necessary to guarantee the availability required. In the Mine Maintenance Section are included a mechanical and an electrical field crew, who will be working on continuous shift to carry out minor repairs at the mine site. The maintenance equipment includes a service truck and a fuel truck mainly for crawler mounted diesel driven equipment.

Dewatering
For the drainage of surface rainwater, drainage ditches will be constructed, encircling the pit. Submersible pumps are used for dewatering of the pit.

6. Mining-Underground

6.1 Background

The open pit production in Fäboliden will be followed by underground mining. A study has been carried out analysing the feasibilities using sub-level caving to mine the ore body below the open pit in Fäboliden. The ore body, significant in size, is assumed to be handled with big sized under ground methods like LKAB:s sub-level caving. The study shows that the technical and economical possibilities to continue mining underground are feasible after the open pit has reached its optimum depth.

The volume of gold ore is estimated to about 30 million tonnes with an average gold content of 1.23 g/tonne (1.07 g/tonne after dilution). The deposit continues down to an unknown depth. It's mineable over its whole length and gold contents vary, marginal quantities are on both hanging- and foot-wall side. An average of 0.2 g/tonne of gold in surrounding material has been estimated in all calculations of loadable ore.

The study for the underground production in Fäboliden is enclosed in Appendix 1(5) and summarized in this chapter.

6.2 Mining method

Sub-level caving is the only underground mining method applicable for mining Fäboliden under ground. The method has been used by LKAB for many decades both in Kiruna and Malmberget and the method has developed towards automation and distance control. The method means flexibility and possibilities to adjust for problems and reduce fluctuation in ore production. Recovery grade can be adjusted to mineral value, rich-high recovery and/or poor-low recovery.

Sub-level caving means ore extraction from top to bottom, 20-30 m at steps. The chosen geometry for Fäboliden is 20 m between production drifts and 25 m at height. The ore body in Fäboliden has several advantages when using sub-level caving:

- Dip 60 – 70 ° E to SE
- Width 20 – 40 m
- Large parts of hanging-and foot-wall are marginal ore.
- Good strength of ore- and foot-wall is good for high pace of development work and low costs.

Some draw-backs are:

- Impossible to see the difference between ore and waste.
- No difference in specific weight of ore and waste.

6.3 Grade control

When loading a blasted round ore, it is not possible to determine visually or by direct measurements if the grade is low or high and if extraction loading should be stopped. Normally that could be at 110 % but loading will be stopped at pre-determined extraction rates and levels, depending upon locally gained records and geology, gold price etc. In this estimation it is assumed that in-situ grades are 1.21 g/tonne in principal ore and 0.2 g/tonne in marginal ore.

In a mining method like sub-level caving, there will be ore losses and ore dilution from surrounding waste and from low grade ore. It is also normal to load more than the blasted volume of ore; extraction rate is often more than 100 %. Typical numbers for sub-level caving are: ore loss – 15 %, dilution – 25 % and extraction rate – 110 %. Average grade in run-of-mine ore in this study, is assumed to be 1.07 g/tonne.

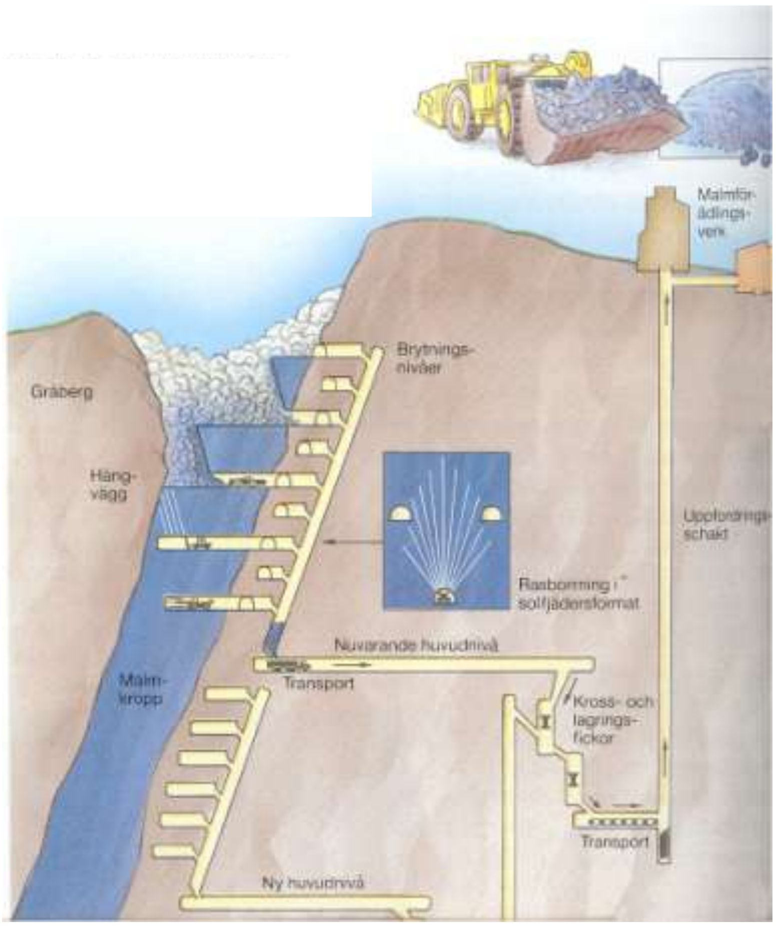

Figure 6.1 Sub-level caving as it is applied by LKAB in Kiruna

6.4 Infrastructure

The ore body, which is some 1300 meters long, is divided into three blocks. They are run like three adjacent and connected mines. Each block has a ramp that starts from the footwall ramp of the open pit. All ramps will have one-way traffic. The central ramp, which will be paved with asphalt, is for up-going traffic. The two ramps at each end are destined for down-going traffic.

At the first stage, there will be no installations such as a hoisting shaft or conveyors to take the ore to the surface. All ore will initially be transported by trucks to the surface. If sufficient quantities of ore are found at depth, there is always the possibility to make either a shaft or a conveyor at a later stage.

Each block is supplied with fresh air from a raise bored shaft, approx. 3 metres in diameter. Initially the used air is evacuated through the ramps and through the caved ore. The shafts are extended from level to level by long hole drilling and blasting. A concrete wall with fans is built at each shaft and on every level.

Systems for ventilation, water pumping, and electrical supply are extended as development drifting advances. There will be no network for compressed air.

6.5 Development drifting

A mining rate of 5 MTPA means that 35 vertical meters of the ore body are mined every year. The amount of development drifting is around 7.5 km per year. All of this will be done by contractor(s), including all rock reinforcement. Standard drill & blast technology is assumed, the choice of dominating method for rock reinforcement will be made when more knowledge of rock quality is available.

Production drilling
Drilling is done by standard hammer drills with 95 mm drill holes. Each machine drills typically 32, 27, 20, 18 and 11 metres. Three drill rigs are required.

Charging
Pumpable explosives are used, both in development drifting and in production. One vehicle for charging the up-holes is sufficient. Required capacity is 4-5 fans per day. Each fan contains around 3,500 tonnes of ore in-situ.

Loading
The ore is loaded onto trucks using (LHD) diesel-driven front end loaders. The haul distances from blasted fan to truck are short, in average around 60 meters. This gives a high capacity, provided of course that the waiting time for trucks is limited. The suggested size is a machine with 12-14 tonnes bucket capacity. Three machines must be in production at all times and one extra machine is needed.

Truck transportation
Transportation of ore to the surface should be done on a contract basis. There are several skilled contractors that can offer good prices and reliable performance. The trucks will be more or less standard highway trucks. Built with four axles they can handle payloads up to 40 tonnes. From the first sublevel, the cycle time will be 23 minutes, requiring 10 trucks for full production. From the second sublevel, one more truck will be needed. To keep this number of trucks running, three spare trucks will be needed. The cost for transportation will go up for every sublevel, but still after the 9 sublevels in this study, the cost is still reasonable.

However, at some point trucking to the surface will not be an option from an economical point of view. If continued exploration drilling at depth is successful and sufficient quantities of ore are localized, studies and investigations for a permanent transportation system should be initiated.

Time schedule

The "project" to develop and mine with 9 sublevels will take around 9 years. An approximate time table is:

- Years 1-2: Only development drifting and other preparatory works
- Year 3: Production starts during first quarter; full production is reached in Q4.
- Years 4-8: Full production, 5 MTPA.
- Year 8: Development drifting stops, declining production during second half.
- Year 9: Around 0.2 Mtonnes left to mine during the year.

There will be two blocks in full production at all times. In addition there will be one or two blocks where mining has either just started or is almost mined out. There must always be room for three LHDs in full production.

6.6 Persons involved in the Pre-Study

The underground study for Fäboliden is elaborated by Ingemar Marklund, Mining Engineer, now retired from LKAB. Ingemar was employed in LKAB for 42 years. He attended LKAB:s mining school before and during his study time. Worked at the Developing Department in Malmberget for 4 years. He was Manager for Mining services in Malmberget for 2 years and Manager for a plant in Malmberget for 2 years. He worked 15 years at Mine Planning Department in Malmberget of which 10 years as manager. He was also Manager of Mining R&D in LKAB for 11 years and CEO in Wassara AB(daughter to LKAB) for 3 years. He was awarded "Nitro Nobels Bergsprängningsstipendium" Blaster of the year 1995 and Atlas Copcos "Bergteknikpris" in the year of 2003.

An earlier scoping study for underground production in Fäboliden was made by M.Sc. in mining Karl Erik Rånman, at that time working as an independent consultant, with 28 years of relevant experience including 17 years previous employment in LKAB, three of those years as manager of Mine Planning and six years as Mine manager in the Malmberget mine. Karl Erik has also been consulted during this study.

M.Sc. in mining Torbjörn Naarttijärvi, with 38 years of relevant experience, currently manager of Mining R & D in LKAB, and M.Sc in mining Jan-Olov Nilsson, Mining R & D in LKAB have reviewed the study, verified or adjusted relevant capacities and cost assumptions.

Other persons that have supported with different kind of information are:

Per Vedin NCC	Develop. Engineer u.g.	Development works
Åke Kruukka LKAB	Manager Develop. LKAB Kiruna	Development works
Stefan Sundqvist	Area Manager BDX (Transportation)	Loading and hauling
Torbjörn Nilsson	Manager Prod.Drilling Malmberget	Drilling
Kjell Harnesk	Manager Charging Malmberget	Charging
Ulf Olsson	Manager Media LKAB	Radio, water mm
S-O Nilsson	Ventilation Planning Malmberget	Ventilation
Roland Vinsa	Electrical Planning u.g Malmberget	Electrical installations
Britt-Inger Bylund	Store Manager LKAB Malmberget	Material prices
Håkan Darehed	Area Salesman Volvo	Transportation

7. Plant and Process

The process plant and the gold leaching process are described below. The description is structured as follows:

- Design parameters
- Process
- Flow sheet
- Plant description
- Control philosophy
- Waste water treatment plant

7.1 Design parameters

The purpose of this chapter is to explain and motivate the selected design operating parameters for the proposed gold recovery circuits, such as leaching retention time, reagent consumption etc.

The used metallurgical data for the design are mainly from test work performed by Msc. in Chemical Engineering Jan-Erik Sundqvist and PhD. in Mineral Processing Nils-Johan Bolin on samples from pilot tests on mining material, which was selected and submitted by Lappland Goldminers. In case of lack of test work data, typical design data based on practical operational experience from several operations has been assumed.

Overall process design considerations

The intention is to concentrate a major part of the sulphur and arsenic content of the ore into a small product for separate disposal. By this method a net acid generation in the main tailings may be prevented on long-term basis, which is believed to lower the overall cost for tailings management significantly. Test work has indicated that achievable sulphur recovery by flotation is significant lower from a cyanide-destructed pulp compared to recoveries obtained from a fresh mill product. The latter has been shown in the pilot campaigns on test mining materials and on drill core samples by laboratory scale tests. The pulp chemistry with respect to the calcium ions content may play a role. In the cyanidation test work prior to the sulphur flotation, lime was introduced both during leaching and cyanide destruction for pH control. It is well known that calcium ions depress minerals such as pyrite, arsenopyrite and pyrrhotite in a flotation process, see TM_REP2005/081 and TM_REP2006/016. The preferred concept is therefore to produce a gold-arsenic-sulphur flotation concentrate, which will be leached in a separate CIL-circuit. This concept will also allow for an overall optimal grinding energy input for liberation of gold particles associated will sulphide minerals since it has been demonstrated that a major part of the gold will float to the sulphidic concentrate. Design parameters for the flotation circuit are reported in TM_REP2006/049.

Leaching Plant

The gold recovery plant comprises of the following main sections:

- Carbon-In-Leach (CIL) section for concentrate
- Carbon-In-Leach (CIL) section for flotation tailings
- Carbon elution column

- Electrowinning circuit dedicated for recovery of gold from concentrate leaching
- Electrowinning circuit dedicated for recovery of gold from flotation tailings leaching
- Cyanide destruction by means of the INCO process
- Ferric arsenate precipitation
- Reagent mixing, storage and distribution

The plant will be designed to treat 4,600,000 tonnes/year of ore grading 1,20 g Au/tonne, with peaks of 1.80 g Au/tonne. Mill availability has been assumed to be at 94.7 % corresponding to 8,300 running hours/year or 554 tonnes of ore/h.

Before flotation, the cyclone overflow will be fed to a magnetic separator in order to remove magnetic pyrrhotite content in order to decrease the cyanide consumption associated with formation of thiocyanate and ferrous cyanide complexes. It is also anticipated that the removal of pyrrhotite will significantly lower the oxygen demand in the concentrate leaching circuit. The pilot tests done by Stråssa have shown that only minor losses of gold to the magnetic concentrate will occur at a weight recovery of about 3 %. The overall weight recovery to flotation concentrate and tailings to be leached are estimated to 5.6 % and 91.4 % respectively at the design grade of 1% As and 2.5 % S.

Only minor variations in arsenic and sulphur content are expected in spite of variations in gold grade.

Leaching-Carbon Adsorption Circuits

Since the submitted samples for test work did not represent the design ore, with respect to gold head grades and weight recoveries to the flotation concentrate, an effort has been made to project the plant performance.

Based on the limited test work data reported in TM_REP2006/016, ultimate recoveries profiles have been constructed based on the assumption that the anticipated recoveries follow the threshold model. The model assumes that there is a lowest achievable residual gold content and that the recovery of the gold content above that value is constant and will correspond to the maximum achievable recovery based on the head grade. In figure 7.1-7.3 the constructed profiles are shown. The parameters (threshold and maximum recovery) have been determined by successive approximation in order to obtain a model for the cyclone overflow product that reasonable well corresponds to the weighted average recovery based on the concentrate and tailings models. For the construction of the models it has been assumed a weight recovery to the flotation of about 30 %, which corresponds to the actual mass balance of received test work material.

The used recovery profile for the flotation concentrate is based on the gold leaching profile obtained from the non-magnetic rougher concentrate reported in TM_REP2006/016. However, as stated in the report, the calculated head grade and the leaching characteristics indicated that the sample actually was a magnetic rougher concentrate. The actual reported analysis of the final pregnant solution showed a significant drop in the gold value, which most probably was due to an error in the gold assay or due to instability of the submitted final leach solution for assaying.

Therefore, it has been assumed that the actual gold tenor of the solution after 48 hours leaching was equal to the measured after 22 hours.

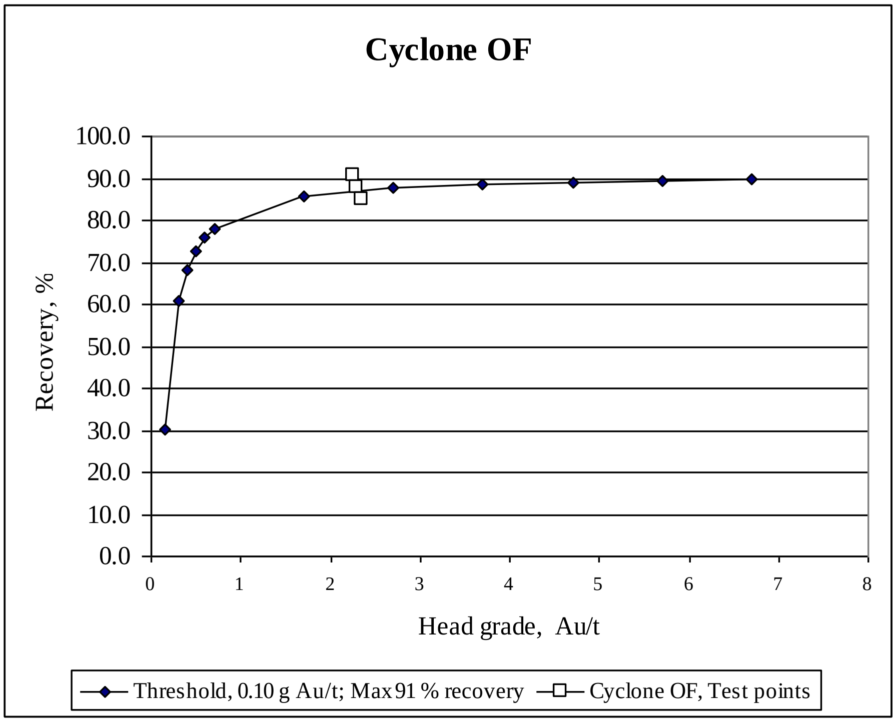

Figure 7.1. Cyclone overflow threshold model
(Recovery/100 = 0.91(Head grade-0,10)/Head grade)*

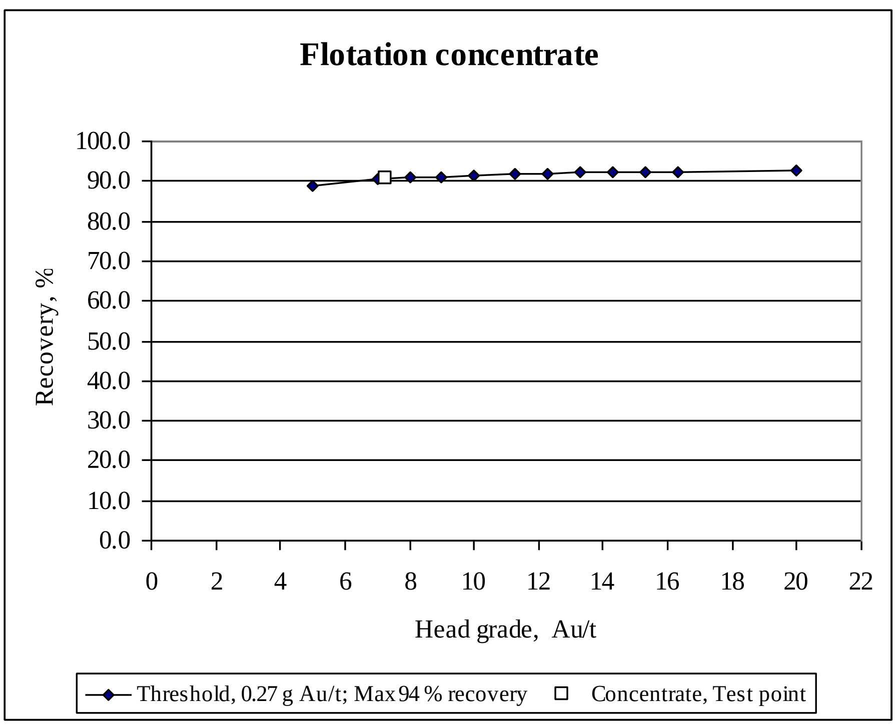

Figure 7.2. Cyclone overflow threshold model
Recovery/100 = 0.94(Head grade-0,27)/Head grade*

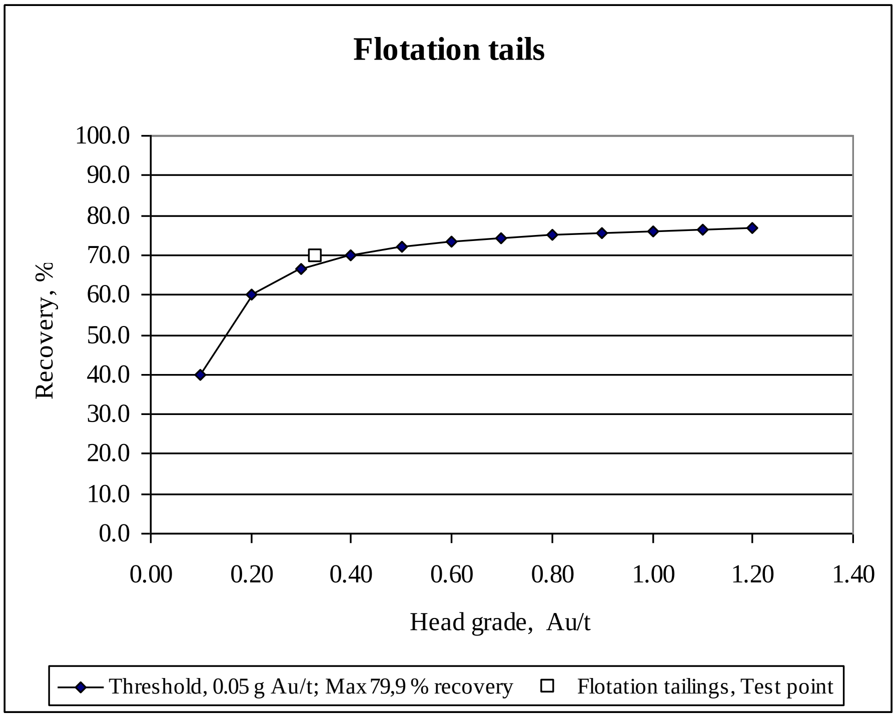

Figure 7.3. Cyclone overflow threshold model
(Recovery/100 = 0.799(Head grade-0,05)/Head grade)*

For modelling the carbon adsorption profile for the proposed CIL circuit, the Fleming model has been used. The model includes two parameters, the k- and n-value. Carbon adsorption testwork has been carried out only on cyclone overflow material. For reference and testwork results, see TM_REP2006/016. From the modelled test work data the k and n values for gold adsorption were in the range of 240-450 h-1 and 0.56-0.83 respectively, which indicate a rather typical and fast adsorption rate. However, since both fresh carbon and fresh process water were used in the test work, a slower adsorption kinetic may be expected in the full-scale operation. Even with regeneration of the carbon on regular basis, a loss in activity is normally observed. Reduced kinetic constants compared to the test work data have therefore been applied for respectively CIL- circuit.

Concentrate

Leaching
A projected design gold leaching profile for the proposed design grade of 19.79 g/t has been constructed. The constructed profile is based on the gold leaching profile obtained from the non-magnetic rougher concentrate reported in TM_REP2006/016, after correction as described above. Since the grade for the proposed design concentrate is higher than actual for the test work concentrate, it is reasonable to assume that the ultimate recovery also will be higher. For design purpose, it has therefore been assumed that ultimate recovery follows the threshold model in accordance to figure 7.2. Furthermore it has been assumed that projected leaching profile is proportional to the ultimate recovery. The projected leaching profile is shown together with the actual obtained from the rougher concentrate in figure 7.4.
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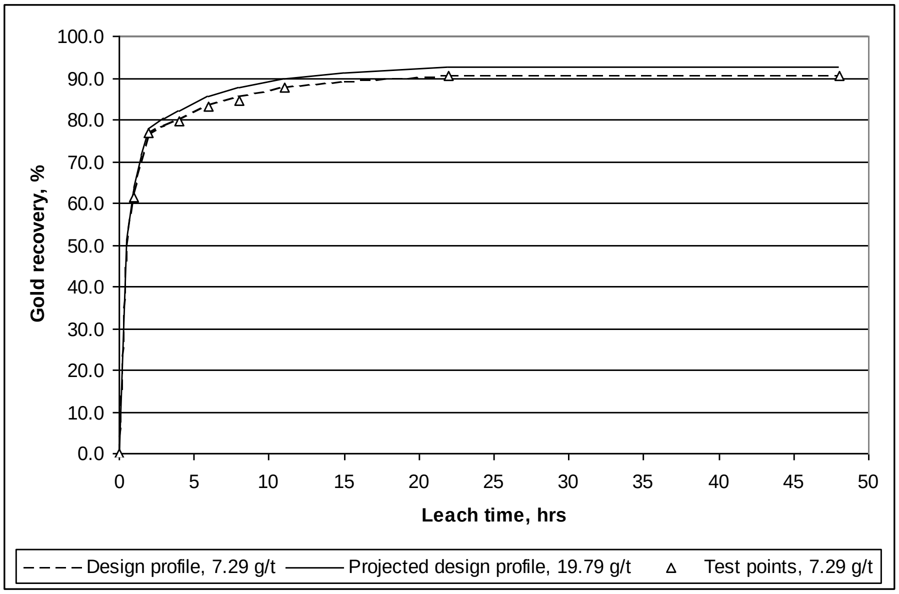


Figure 7.4. Design gold leaching profiles

The achievable leach recovery of a multi-stage CIL circuit will depend on number of stages beside the total retention time. For a CIL operation it has been found that an 8 stages circuit is convenient since it allows for flexibility in the selection of number of adsorption stages and will minimize recovery losses in case of by-passing a tank for maintenance etc.

In table 7.1 the predicted leaching profiles for an 8-stages circuit based on the projected design leaching are given. The prediction of the continuous profiles is based on assuming an ideal mixing system. For design a 24 hours circuit has been selected which gives a retention time of 28 hours at nominal flow rates. As can be seen from the table below a rather limit increase in recovery by a longer retention times can be expected. The gold leaching losses due to by-passing one stage is expected to be in the range of 0,5-0,6 % based on the feed to the circuit. In addition, soluble losses and losses by fine carbon have to be considered for overall recovery estimate.

		Estimated recovery, %								
	Leach time	Total	Tank 1	Tank 2	Tank 3	Tank 4	Tank 5	Tank 6	Tank 7	Tank 8
	20	91.21	65.67	13.46	4.06	3.08	1.96	1.39	0.89	0.70
	21	91.43	66.38	13.24	4.01	3.03	1.92	1.35	0.82	0.68
	22	91.60	67.05	13.00	3.96	2.97	1.87	1.30	0.82	0.63
	23	91.73	67.68	12.76	3.92	2.91	1.82	1.24	0.82	0.58
Design	**24**	**91.88**	**68.27**	**12.53**	**4.01**	**2.78**	**1.74**	**1.17**	**0.81**	**0.57**
	25	92.02	68.83	12.31	4.08	2.64	1.72	1.08	0.79	0.57
	26	92.11	69.36	12.08	4.14	2.53	1.69	0.98	0.77	0.56
	27	92.27	69.87	11.86	4.20	2.50	1.64	0.97	0.70	0.53
Nominal	**28**	**92.38**	**70.35**	**11.64**	**4.23**	**2.47**	**1.58**	**0.96**	**0.65**	**0.50**

Table 7.1. Predicted leaching recovery profiles in continuous 8 stages circuits.

Carbon adsorption
The design of the carbon adsorption circuit (CIL) is based on a 1-stage pre-leach followed by 7 tanks containing activated carbon. The k and n-values of the Fleming have been assumed to be 70 h-1 and 0.70 respectively. The required carbon inventory has been calculated to be about 32 tonnes and is based on a gold loading of 4000 g Au/tonne of carbon and 30 g Au/tonne of returned carbon. The barren gold tenor of the solution was assumed to be 0.04 mg/l.

Flotation tailings

Leaching
A projected design gold leaching profile for the proposed design grade of 0.70 g/t has been constructed. The constructed profile is based on the gold leaching profile obtained from the flotation tailings reported in TM_REP2006/016, appendix 4a-1 after correction as described above. Since the grade for the proposed design tailings is higher than actual for the test work concentrate, it is reasonable to assume that the ultimate recovery also will be higher. For the design purpose, it has therefore been assumed that ultimate recovery follows the threshold model in accordance to figure 7.3. Furthermore it has been assumed that projected leaching profile is proportional to the ultimate recovery. The projected leaching profile is shown together with the actual obtained from the rougher concentrate in figure 7.5.

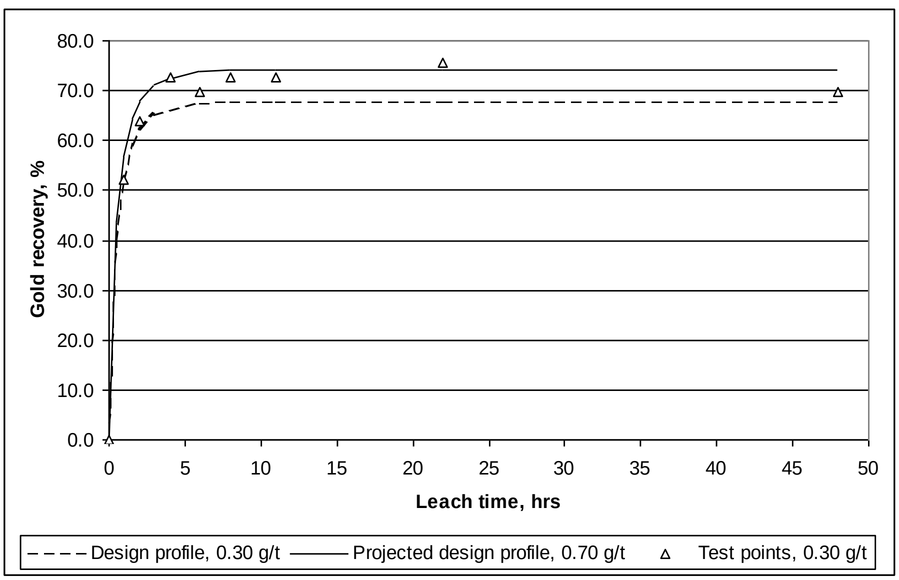


Figure 7.5. Design gold leaching profiles

In table 7.1 the predicted leaching profiles for an 8-stages circuit based on the projected design batch leaching profile are given. The prediction of the continuous profiles is based on assuming an ideal mixing system. For design, a 10 hours circuit has been selected which gives a retention time of 11.5 hours at nominal flow rates. As can be seen from table 7.2, a rather limit increase in recovery by a longer retention times can be expected. The gold leaching losses due to by-passing one stage is expected to be in the range of 0.1-0.3 % based on the feed to the circuit. In addition, soluble losses and losses by fine carbon have to be considered for overall recovery estimate.

		Estimated recovery, %								
	Leach time	Total	Tank 1	Tank 2	Tank 3	Tank 4	Tank 5	Tank 6	Tank 7	Tank 8
	5.0	72.14	35.76	17.63	8.55	4.66	2.38	1.54	0.99	0.63
	6.0	72.66	38.98	16.86	8.00	3.91	2.06	1.44	0.76	0.65
	7.0	73.24	41.67	16.11	7.61	3.45	1.94	1.07	0.76	0.63
	8.0	73.67	43.95	15.44	7.31	3.08	1.72	0.91	0.72	0.54
	9.0	73.90	45.92	15.34	6.58	2.60	1.49	0.90	0.68	0.39
Design	**10.0**	**73.98**	**47.63**	**15.16**	**5.85**	**2.46**	**1.15**	**0.86**	**0.61**	**0.26**
	11.0	74.02	49.13	14.85	5.22	2.34	1.13	0.80	0.44	0.11
Nominal	**11.5**	**74.06**	**50.40**	**14.49**	**4.86**	**2.14**	**1.07**	**0.71**	**0.31**	**0.10**

Table 7.2. Predicted leaching recovery profiles in continuous 8 stages circuits.

Carbon adsorption

The design of the carbon adsorption circuit (CIL) for the tailing leaching is also based on a 1-stage pre-leach followed by 7 tanks containing activated carbon. The k and n-values of the Fleming have been assumed to be 100 h-1 and 0.70 respectively. The adsorption kinetic has been assumed to be somewhat faster than assumed for the concentrate circuit since a more “clean” pregnant solution can be anticipated. However, the achievable loading will be lower by the lower gold tenor of pregnant solution. The required carbon inventory has been calculated to be about 101 tonnes and is based on a gold loading of 1500 g Au/tonne of carbon and 30 g Au/tonne of returned carbon. The barren gold tenor of the solution was assumed to be 0.02 mg/l.

Carbon elution and regeneration and Electrowinning

A standard AARL elution process is suggested for desorbing gold from the loaded carbon. The proposed capacity of the elution column is 6 tonnes of carbon. Loaded carbon from the two leaching circuits will be treated separately in the column. At the design capacity totally 431 strips will be conducted annually.

Two separate electrowinning circuits are required to manage to plate out the expected gold contained by the produced electrolytes. The following assumptions have been made:

- Total electrolyte volume: 98 m^3
- Gold recovery: 98 % for a 25 hours solution re-cycle
- Flowrate: 30 m^3/h
- Cell efficiency: 50 %

For carbon regeneration, a standard horizontal LPG-fired kiln is suggested. It is proposed that all stripped carbon is regenerated before recycled back to the CIL circuits in order to restore and maintain its activity.

Cyanide destruction

The widely accepted INCO process is suggested for destruction of the residual cyanide content in the pulp from the CIL circuits. However, the removal of thiocyanate is expected to be limited. Analyses of solutions from the carbon adsorption testwork conducted on magnetic cyclone overflow show that leach solution contained 250-330 mg SCN/l. By the removal of magnetic pyrrhotite it is assumed that generation of thiocyanate will be brought down to an acceptable level from an environmental point of view.

The required SO_2 and aeration dosage rates have been assumed in accordance to Boliden in-house experience since no test work has been carried out. The Vendor, who will license the process, normally does detailed design of the destruction circuit based on their own test work.

Ferric arsenate precipitation
It has been shown that the leach solution will contain ppm-levels of arsenic and antimony. A standard method to remove any residual arsenic in a pulp discharging from the INCO process is to add ferric in order to precipitate ferric arsenate. It is normally found that the required molar ratio of ferric to arsenate should exceed 3:1. For design of the concentrate Fe-As precipitation tank, a molar ratio of 6 to 1 has been assumed at 8 mg As/l, which is about 13 times higher than measured in one of solution from the carbon adsorption tests. The required total retention time has been assumed to be 1.5 hours.

Reagents

Sodium cyanide
The testwork reported in TM_REP2006/016 strongly indicated that both of the submitted rougher concentrates were magnetic. The obtained design cyanide consumption profile reported in appendix 2a-3 has therefore assumed to be generated from a magnetic rougher concentrate. For the construction of a design cyanide consumption profile it was assumed that 100 % of the sulphur was contained by the rougher concentrate and that the cyanide consumption is directly proportional to the sulphur content. It was further assumed that 32,4 % of the sulphur content will removed by magnetic separation. In figure 7.6 the constructed cyanide consumption profiles for the magnetic concentrate is shown. The prediction of the continuous profile is based on assuming an ideal mixed 8 – stages leaching system

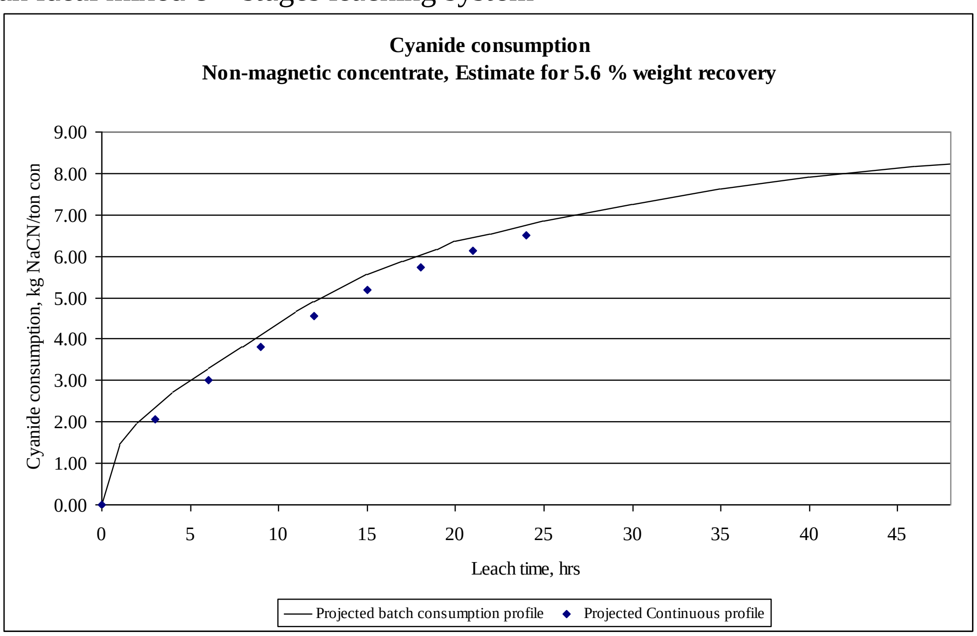


Figure 7.6. Projected cyanide consumption profile for non-magnetic concentrate.

For design, additional cyanide has assumed to be consumed by operating the last CIL tank at a free concentration of cyanide corresponding to 0.025 % NaCN. The overall cyanide consumption for the designed concentrate CIL circuit is estimated to 6.88 kg/ton of concentrate. For design, it is recommended to use a consumption rate of 10 kg/ton of concentrate to cope with higher consumption rates.

The design of the tailings leaching circuit with respect to cyanide consumption is based on the obtained profile reported in appendix 4a-3. In figure 7.7 the constructed cyanide consumption profiles for the flotation tailings are shown. Since the sulphur content in the testwork tailings was very low by the high weight recovery to the rougher flotation concentrate, the testwork profile may give an underestimation of the cyanide demand in the circuit. The design has therefore been based on a projected batch consumption profile that will give 25 % higher consumption rate compared to the test points.

For design, additional cyanide has assumed to be consumed by operating the last CIL tank at a free concentration of cyanide corresponding to 0.015 % NaCN. The overall cyanide consumption for the designed tailings CIL circuit is estimated to 0.43 kg/ton of tailings. For design, a consumption rate of 0.50 kg/ton of concentrate should be applied to cope with higher consumption rates.

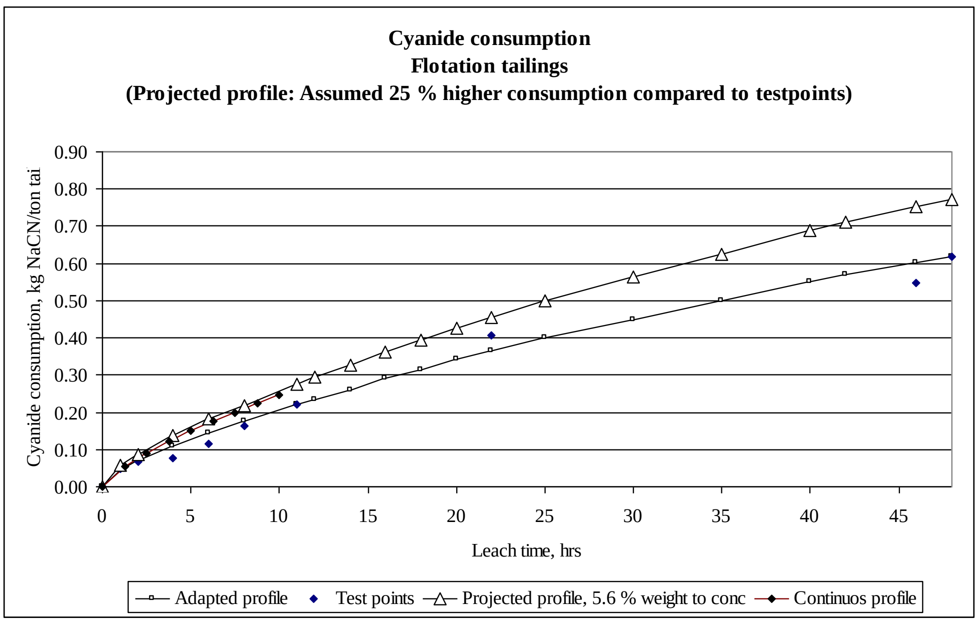


Figure 7.7. Projected cyanide consumption profile for flotation tailings.

The initial testwork carried out by AMMTEC LTD on drill cores indicates a strong effect from a finer grind on the cyanide consumption. The estimated cyanide consumption for the non-magnetic may therefore be underestimated.

Lead nitrate

It is known that the addition of lead nitrate sometimes improve the gold leaching rate by catalysing the oxidation of detrimental reduced sulphur species appearing in a leach solution, especially when leaching gold associated with pyrrhotite. It is also reported that lead nitrate may lower the cyanide consumption.

A typical reported addition rate of 200 g lead nitrate/ton concentrate has therefore been proposed in order to minimize the expected effect from a finer grind compared to what the testwork material represented.

7.2 Process

Background
A process for the extraction of gold has been designed. Basic data has been collected from mineralogical studies, laboratory tests covering all steps from grinding to cyanidation. Grinding and flotation tests have also been performed in a pilot plant.

As the ore is of relatively low grade and the ore body is very big, the keyword for the process has been cost efficiency. The most important to consider is the gold recovery and the consumption of energy and chemicals, mainly cyanide. It has also been an important issue to consider the disposal of the waste in an early stage of the work.

In this document the results from the tests are summarized and conclusions are made. Performed tests, enclosed in appendix, are:

- AMMTEC, 2001, metallurgical testwork, drill cores, laboratory scale
- SGAB Analytica, 2001, mineralogical examination of 5 samples
- Jan-Erik Sundqvist et al, 2001, laboratory grinding tests, drill cores
- Minpro, 2005-2006, pilot plant tests, ore from test mining
- Sandvik, 2005, abrasion- and impact twork index tests, ore from test mining
- Knelson, 2005, gravity tests, laboratory scale, drill cores
- Jan-Erik Sundqvist et al, 2005, leaching tests, bench scale
- Knelson, 2005, gravity tests, pilot plant, ore from test mining
- Jan-Erik Sundqvist et al, 2006, leaching tests, bench scale
- Minpro, 2007, complementary pilot plant tests, test mining ore

Mineralogy
A qualitative study has been done by SGAB Analytica. It shows that gold appears at the sizes of 3- 40 microns in grain boundaries of arsenopyrite. However, also small particales, 3-5 microns can be seen locked in the arsenopyrite.

An assumption made out of the testworks also tells that there is gold independent of the arsenopyrite, that gold is mainly cyanidable, but some seem to be locked in the gangue. A rough estimation to quantify the gold distribution gives:

- 60-65 % related to the arsenopyrite and cyanidable with “normal” grinding
- 20-25 % related to the gangue and easy cyanidable
- 10 % locked in the arsenopyrite and not cyanidable without fine grinding
- 5 % locked in the gangue
- 1 % related to the pyrrhotite

It is also noted that aresenopyrite and pyrrhotite are the major sulfides and that there only small amounts of calchopyrite, galena and other metal sulfides. The amount of the major sulfides can be calculated to roughly 3 respectively 4 % and the total contents is less than 8 %.

Tellurides, silver as well as gold, are also noted. Some graphite has also been observed. Fortuneatly the gangue contains some acid buffering minerals i.e. calcite which is of importance for disposal of the waste.

Samples
For the laboratory tests at AMMTEC Australia, Knelson Canada and by Jan-Erik Sundqvist et al., ½ HQ drill core was taken out. Amounts of 50 kg were tested

For the pilot tests at Minpro, Stråssa Sweden, 80 tons were taken out of 1,000 tons of blasted ore. The gold assays were 2.1 g/t (AMMTEC) and 3.3 g/t (from test mining) respectively. The target was to take out representative samples and the deviation from the expected gold contents 1.3 -1.5 g/t, is remarkable.

From other points of view the samples seems to be representative.

Figure 7.8 Test Mining at Fäboliden in 2005

General considerations
The most important factor for the economic result in the process is the recovery of gold. Anyhow, that is also a question of optimizing. The major costs are those for energy, chemicals and waste disposal. Of course manning and maintenance costs are of importance but not critical.

From the mineralogical studies an assumption can be that it is quiet easy to achieve a recovery of 80 -85%, but that it is much more difficult to get higher without fine grinding with rapid increasing costs for energy and chemicals. A basic idea has been to do a preconcentration of the gold and sulphur contents by gravity methods and/or flotation. In that way the leaching circuit can be dramatically reduced and most of the tailings non acidic and disposed in a cheap and simple way. Further it should be possible to grind only a part of the feed finer for higher extraction.

A very important item has been to confirm the possibility for autogenous grinding, as other alternatives are economically out of question.

Test results

Crushing
Samples were taken out from the pilot plant testing rocks and sent to Sandvik for estimating Impact Work Index and Abrasion Index. These were found to be 23.7 respectively 0.4042. Both are relatively high and show clearly what was observed in the pilot plant crusher, where crushing was difficult to handle.

Grinding
Already the laboratory results on Bond index etc indicated that the ore would be difficult to grind. Therefore it was very important to perform pilot tests to confirm that autogenuos grinding is possible and find out about energy consumption. Another purpose was to get the design criterias for a grinding circuit in a production plant.

The tests show that it is necessary to crush 80 % of the feed to less than 20 mm to get a stable circuit. The size of grinding stones has been 80-250 mm.

The tests have been performed in a closed circuit with a screen, sizes 200-500 mm, or with a hydrocyclone. The feed has been 300-500 kgs/h. With k80=80 microns the energy consumption has been 25 kwh/t and for 53 microns 39 kwh/t.

Gravity separation
As there are big differences in density between the gangue, the sufides and the gold, tests were done to study the possibility to perform a preconcentration of the gold and/or the sulphide contents. Laboratory tests were performed at Knelsons´ Laboratories as well as at Minpro with equipments working at more than 200 G (gravity). The results show very low recoveries as well as low grades of the gold in a concentrate. More than one ton of ore was also tested in Knelsons´ pilot plant in Harare, Zimbabve with the same result.

At Minpro, also tailings from flotation tests were investigated with disappointing result. The gold contents were 0.9 g/t and sulphur less than 0.1 %. It is known that most of the gold is cyanidable and therefore exposed. Probably the gold is separated physically only to a small extent.

The gravity tests seem to confirm the mineralogical study of SGAB Analytica.

Flotation tests
Tests at laboratory scale were performed at AMMTEC, Australia and at Minpro Sweden. They show very high recoveries for sulphur, 99 %, but less for gold, 72-84 %. Weight recoveries are 11-13 %.

Flotation was also tested during the grinding tests in the pilot plant with a little bit better results, 98-99 % for sulphur and 79-86 % for gold. As mentioned above most of the gold in the tailings is cyanidable, but obviously not possible to separate by flotation or gravity.

Figure 7.9 Process development in pilot scale at Minpro, Stråssa in 2005

Magnetic separation of pyrrhotite
Pyrrhotite has been shown to have an influence on the gold extraction as a consumer of oxygen as well as cyanide. Therefore it might have influence on the size of the leaching circuit, recovery and consumptions.

Tests have been performed at Metso Sweden and Minpro Sweden with interesting results. By magnetic separation, pyrrhotite is easily recovered and with a loss of gold less than 2 %.

Leaching
Leaching tests have been performed at AMMTEC, Minpro and by M.Sc. in Chemical Engineering Jan-Erik Sundqvist and PhD. in Mineral Processing Nils-Johan Bolin. The ore, flotation concentrates and tailings have been tested. High recoveries of 93% have been achieved at AMMTEC but with high consumption of chemicals and energy for fine grinding.

The Minpro tests show recovery levels of 85 -90 % with more realistically efforts. To reach this it has been necessary to leach the flotation concentrate as well as the tailings. The extraction of gold from the tailings is relatively rapid and slow for the sulphide concentrate. It is also noted that with the hydrocyclone in the grinding cirquit, finer sulfides are recovered as expected, which gives a higher gold recovery.

The tests by Jan-Erik Sundqvist et al. were performed in a relatively large bench scale, 10 kgs samples. In these tests also rates and consumption of sand adsorption on carbon were studied.

Most samples tested so far has had higher gold contents than what is expected for average ore. A theoretical extrapolation to 1.5 g/t gives a realistically recovery of 83-85 %. This will be confirmed in final leaching tests where drill cores with a grade range, 1.0-1.6 g/t has been taken out. When using the method with finer grinding of the concentrate the recovery will increase with 2-3 % to 85-88 %.

Finally it should be noted that in some tests on drill cores, lower recoveries were achieved. That is believed to be caused by high contents of graphite, which normally is not found in the ore. In the foot-wall of the ore-body however it is frequent.

Tests of tailing impoundment systems
Pilot plant tests, 12 t, was started in October 2006 to study the disposal of the waste. The idea was to dispose the sulfides in a center part of the impoundment and with the non acidic around it as a “beach”. This study will be continued as the results per August 2007 are very promising.

Summary
Laboratory as well as pilot plant tests shows that the gold extraction will be between 83 and 93 %. This depends of the ore grade and the optimisation between recovery versus the variable operating costs. These depend mainly of the cyanide costs and the electric energy costs for grinding. Consumption will be 0.8 kg/t respectively 25 kwh/t. The last figure is corresponding to k80=90 microns, which roughly is thought to be the optimal size before the separation of sulfides and further grinding of the arsenopyrite concentrate to k80 minus 20 microns. If the grade is 1.2 g/t this will give a recovery of 83 %.

7.3 Flow Sheet

The proposed process is outlined in the figures below.

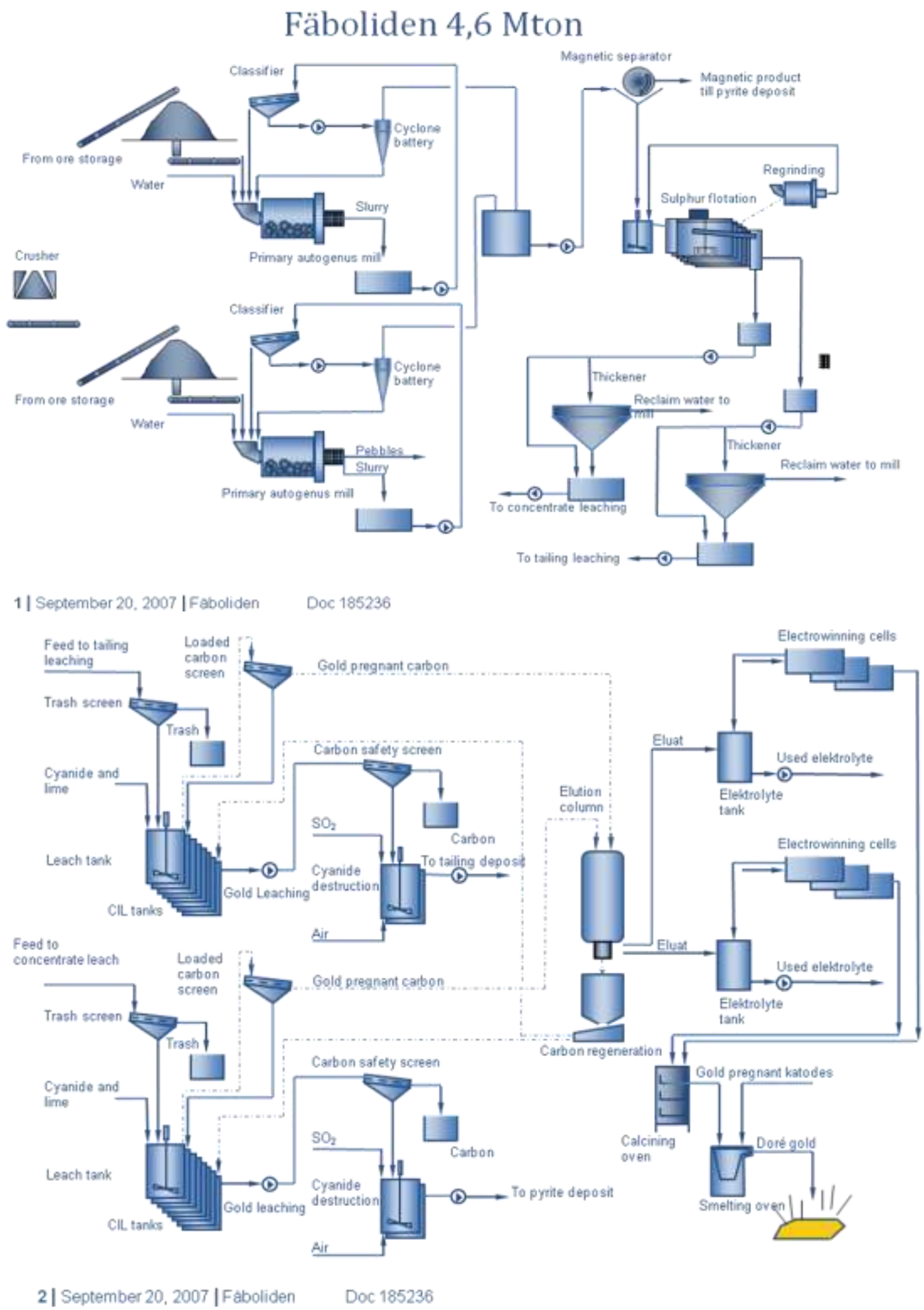


Figure 7.10 Flow sheet

7.4 Plant Description

General
The process plant is designed to treat 4.6 Mtonnes per year. The process plant consists of:

- Crushing plant
- Ore storage
- Two singel lines AG-mills
- Flotation department
- Gold leaching of concentrate and tailing separatly
- Common gold recovery department
- Reagents preparation and fuel arrangments
- Tailing pumping separatly from concentrate leach line and tailing leach line
- System for water supply
- Waste water treatment plant

Infrastructure buildings connected to the concentrator are placed nearby to the concentrator. The following buildings are included:

- Workshop (Mechanical and electrical)
- Office for operating and maintenance personel
- Canteen and Locker room

The process control system is composed of standard modules and will provide the plant with a high level of automatic control. Development of the recovery process can easily be accomplished. The control room will be the center of operations for the entire plant, and the equipment will be conducted remotely. The design of the process plant is based on design data from laboratory and pilot tests. Lappland Goldminers has provided the design data for the flotation department and the gold leaching department.

A detailed description of the concentrator/process can be found in Appendix 1(5) and Appendix 2(5).

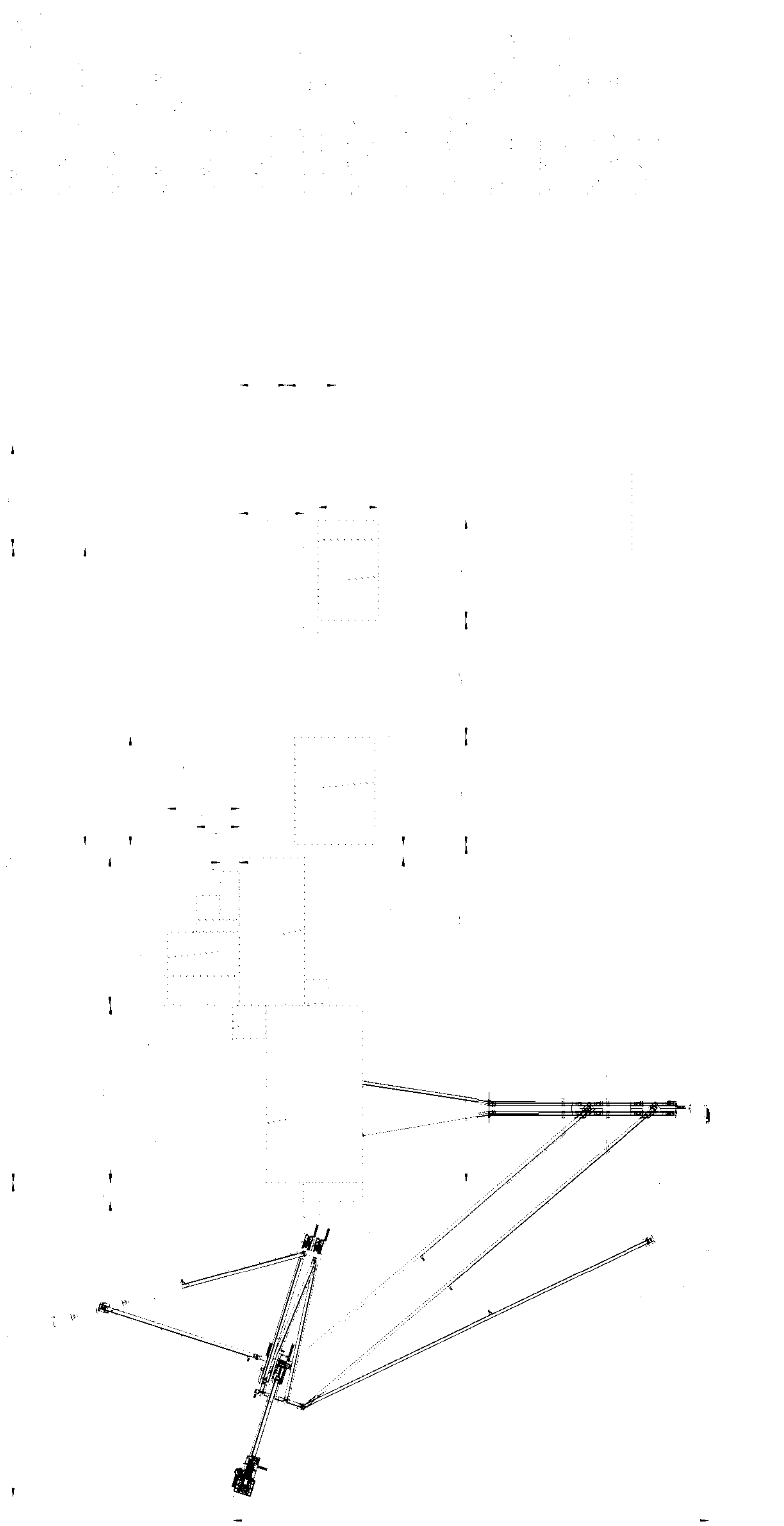


Figure 7.11 Plant Layout

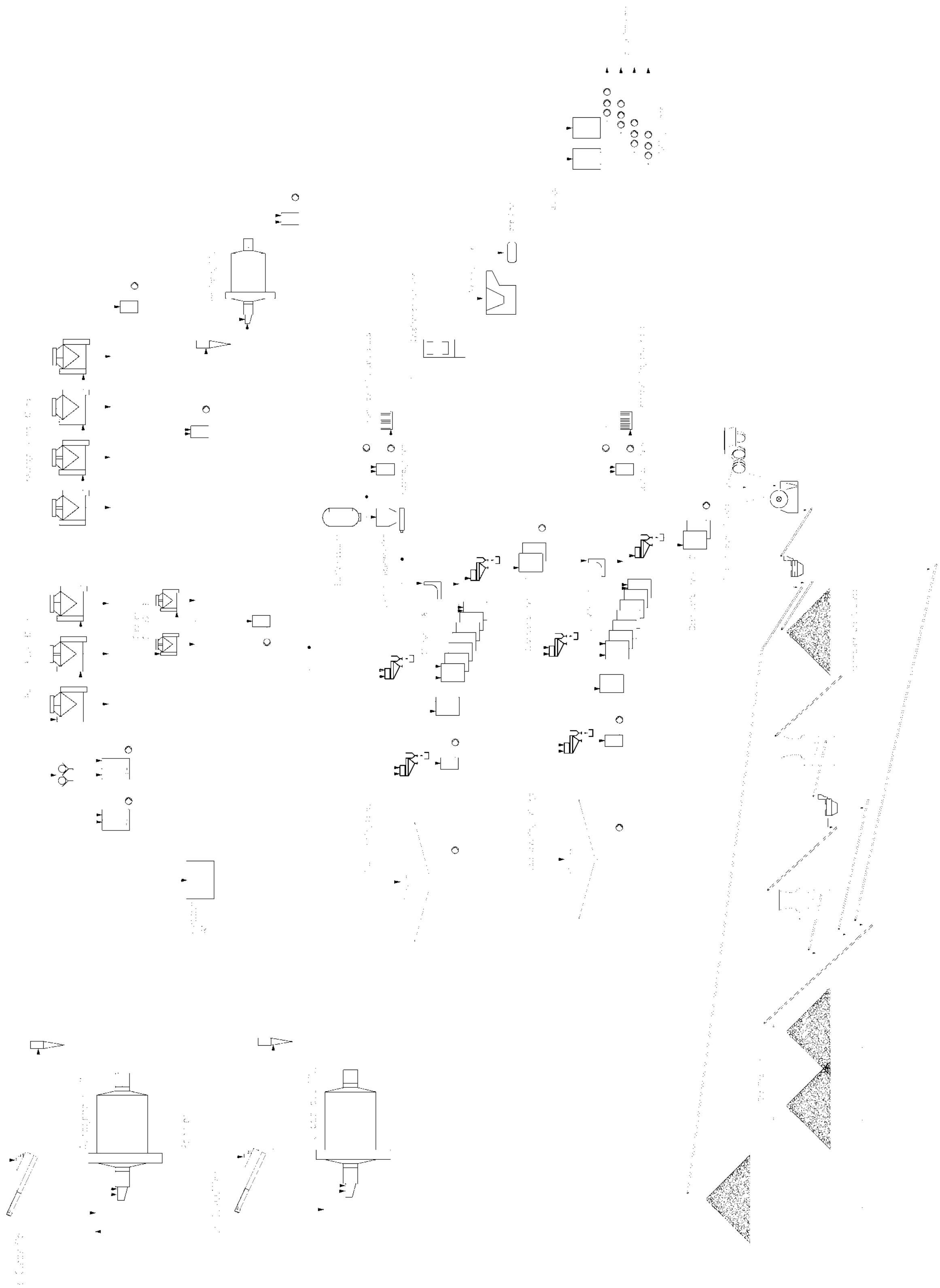


Figure 7.12 Process flow sheet

Crushing, conveying and storing
In between the mine and the concentrator a crushing plant is planned, consisting of a complete system of crushers, screens, feeders and conveyors together with steel supports and other required auxiliaries. The ore is dumped in a truck dumping pocket sized for trucks with a loading capacity of 200 tonnes. The crushing is performed in two stages. After screening, a middle product is stored in stockpile 1 and after secondary crushing the fine products is stored in stockpile 2. The living capacity of stockpile 2 is 7.400 tonnes which give a nominell storage time of 12 hours.

The discharge from the stockpile into the two grinding circuits is conducted by the use of two separate conveyors and their feeders. The two different crushed products will be mixed on the conveyors under guidance of the control system in order to feed the mills with the required relationship (which varies) between the fractions.

Grinding
In the grinding area there are two parallel circuits designed for 2.3 Mtonnes each. Each circuit consists of one primary fully AG-mill, spiral classifier and cyclone cluster. The mills are of great discharge types. The mills are approximately 8.3 metres in diameter and 11 metres long and have a 4000 kW motor. The bearing for the trunnion is of hydrostatic type and the mill in one of the circuits has a discharge trunnion screen where also pebbles can be discharged. The mills are equipped with steel capped rubber lining, which is replaced with assistance by a relining machine.

Flotation and leaching
The flotation area is common for the two milling lines and includes magnetic separation, rougher flotation circuit, scavenger flotation circuit, a cleaner flotation circuit and regrinding mill for cleaner tail and scavenger concentrate. Sulphur rougher flotation circuit consists of three 130 m^3 separate flotation cells and the four scavenger floatation cells are also 130 m^3. The cleaner circuit consists of two 80 m^3 flotation cells. The concentrate from the cleaner circuit feed the concentrate leaching circuit in the gold leaching department. The tailing from the scavenger circuit is lead to the tailing gold leach circuit.

Gold leaching
The leaching of gold by cyanide is divided in two lines, concentrate leaching and tailing leaching. The dissolved gold adsorbs on active carbon in the so-called CIL- process, Carbon in Leach. The tailing leach circuit consists of one leach tank, seven CIL tanks, two cyanide destruction tanks and two arsenic precipitation tanks. In total 12 tanks, with the volume of 1,000 m^3 .

The concentrate leach circuit consists of one Pre-preparation tank, one leach tank, seven CIL-tanks, two cyanide destruction tanks and one arsenic precipitation tank. In total 12 tanks, with the volume of 155 m^3.

To feed the leaching lines with slurry of correct density, thickener for both lines are installed. The gold recovery process equipment is common for both the concentrate leach and the tailing leach. The gold that has been adsorbed on the carbon dissolves in the elution process and the extraction is made by electrowinning of the solution. The catodes are calcinated, smelted and moulded to doré-bullions. The tailing from the two lines will be deposit separatly in the tailing dam.

7.5 Control Philosophy

Operating Strategy
The following will be considered when developing the process control philosophy:

- The plant is in operation 24 hours a day and 365 days a year, with only short stops for scheduled and preventive maintenance
- Manning level philosophy/requirement
- Identification of critical measurements/actions from safety or process view points
- Camera System Layout

Process Control philosophy
Overall objectives are:

- *Flexibility*: The nature of the Fäboliden ore body requires the plant process control system to be as flexible as possible. Changes to the process methodology and to the controlled parameters will be needed on a regular basis as the recovery processes are developed and fine-tuned. One of the main objectives will therefore be to ensure that such changes can be easily accomplished without major reconfiguration of the control system.
- *Standardization*: No special or customized equipment is to be used, and only the normal software configuration should be necessary. The control system shall be composed of standard modules, readily available from a major manufacturer.
- *Automation*: A further important consideration is the degree of automation to be implemented in the process control. With so many different processes in use, and such a range of process variables to control, a high level of automatic control is necessary. This will help to reduce the staffing levels, as well as improve quality and reduce the possibility of errors.

Conventional control panels versus operator stations
For the reasons given above, a control system consisting of dedicated control panels and discreet single-loop controllers would not be suitable. As much functionality as possible should be implemented in software. Field instruments should be flexible and multi-functional to the maximum extent. A modern electronic control system is required, that can be easily altered and expanded as the process demands.

Special Control Requirements
Process instrumentation and control requirements have been reviewed at their present status to determine if there are any special requirements that need to be taken into account. In the design as it stands, all the process control loops associated with this plant are of a conventional nature and do not warrant any special treatment. Should any unusual or special needs be identified, these will be highlighted for attention.

System Architecture
It is proposed to make use of a suitable PLC- and PC-based control system, with ready-made software modules for the standard control functions, and with a high degree of flexibility. The plant is large and has to be controlled from several central located process stations, so the control system functional units will be widely spread at different location. There will also be a need of a

redundant data network between process control units and control rooms.

A typical control system is the ABB 800xA series of programmable logic controllers and associated SCADA (supervisory control and data acquisition) hardware and software. A control system based on this architecture, or equal, is proposed for the Fäboliden plant.

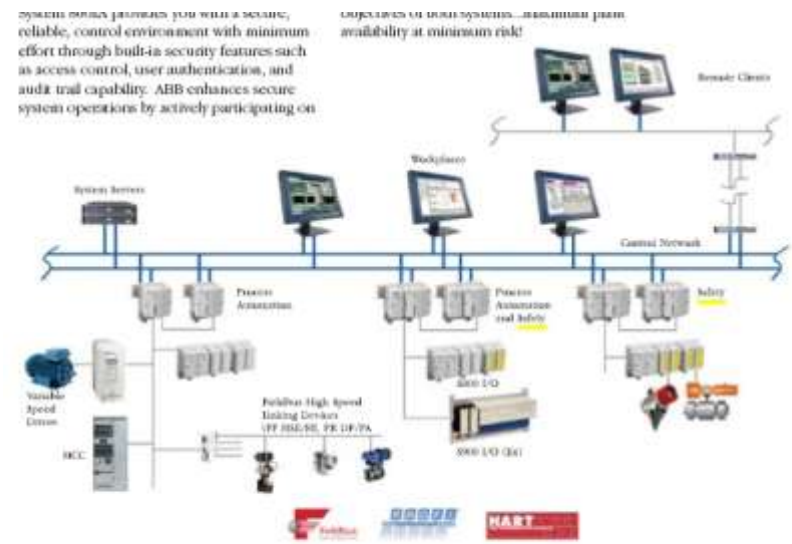


Figure 7.13 Typical System Architecture

Communication between the processor of the control system and the PLC modules and field instrumentation can be implemented by means of conventional cabling to various concentrators located in the field. It is also possible to consider the new field bus technology, which would make it easier to add functionality in the future, as well as distributing some of the system intelligence amongst the field instruments.

Reliability of the process control system is critical to certain point, as it is a full commercial production plant. The consequences of a short-term loss of production would be costly as to justify full dual redundancy of the control system. However the communication links has to be redundant. It is therefore accepted that a failure of the control system or its ancillaries will cause some disruption to the process, until the fault is rectified. No dual redundancy of processors but redundancy for data links is therefore proposed.

The Process Station structure will be designed so a failure of one of the Process Stations will not affect the whole plant. For example grinding Line1 and 2 will therefore be controlled from different Process Stations.

Control circuit philosophy

A standard control circuit will be applied, in which each item of electrically driven equipment has a local control box with a four-position switch.

Sequence control circuitry shall be designed for fail-safe operation so that on loss of control supply or on circuit device failure, the circuitry will either stop motor drives, or assume a state that ensures maximum personnel and plant safety. Techniques such as backup or second level protection, permissive interlocks and separate safety and control circuits should be used.

Safety interlocks, such as emergency stop pushbuttons, pull cords, etc. shall be provided for operator safety. Inputs from the corresponding field devices must be hard-wired directly into drive control circuits, and not applied via PLC software. These inputs will be monitored by the control system for diagnostic purposes.

Control interlocks, as required for the protection of equipment against overload or damage and for the prevention of maloperation, shall normally be wired to the control system only and shall generally be active in the remote mode only.

A control interlock for an electric drive that is required to be active in the test mode shall be treated as a safety interlock, i.e. an input from the corresponding field device shall be hard-wired into the drive starter circuit and shall be monitored by the control system for diagnostic purposes.

Circuit design should be subjected to a safety assessment or risk analysis to identify possible plant malfunctions or failures, the probabilities and consequences of such failures, and the effectiveness and reliability of safeguards.

Where intelligent motor protection relays are used, they should be interfaced to the PLC control system by using the in-built communication capabilities and “daisy chaining” of the devices wherever possible.

Control Room

The control room will be the centre of operations for the entire plant, and it is envisaged that the operators in the control room will have responsibility for all the processes taking place. Operations on equipment will be conducted remotely, even automatically in many cases, and it is not necessary for the control room to be in visual contact with the operations taking place. It could therefore be sited in the most convenient location, with a view to ease of access, freedom from dust and dirt from process operations, and convenient to administrative offices.

The control room will be large enough to accommodate all operators needed for operation with all the necessary document storage, printers and other accessories, plus a further engineering position for development work. The engineering station may be located in a separate room adjacent to the control room. For operator well being, maximum use will be made of natural lighting in the control room.

Suitable heating, ventilation and air conditioning equipment will be provided for both the control room and the adjacent technical rooms.

Visual system

The visual system will allow all video cameras to be connected to TCP/IP network. The benefit will be that operator and process engineers can access any camera in the network within the computer network. It will also be possible to monitor video cameras trough the Operator Station.

Data Collection and Management
A process historian will be integrated with the control system, which will make available process information and historical data for trending and management purposes. Full capability for in-house development of this facility will be provided. The application will preferably run under Windows and use open standards for data exchange. Provided the necessary safeguards are put in place, the control system can be networked to an administrative computer network, allowing access to the process data from the normal office-based computers.

7.6 Waste Water Treatment Plant

Background
The mineral process for gold extraction needs a number of chemical agents, with minor or major impact on the water environment, when the excess process water is discharged. For the actual situation at Fäboliden the extensive use of sodium cyanide (NaCN) is of special sensitivity for the environment, even if the cyanide is transformed into cyanat and ammonia.

However even a discharge of ammonia nitrogen is a delicate matter with respect to the water environment. In this case the annual use of sodium cyanide will cause – after the transformation into cyanat and ammonia nitrogen – a discharge of nitrogen of about 840 tonnes that would be equal to a discharge from two of the main cities in the region, about 160 000 person equivalents. It has therefore been decided to build an external treatment facility for the removal of the nitrogen from the wasted process water. After thorough considerations and analysing different treatment models, it was decided to operate a pilot test on biological treatment of the water. A pilot plant based on the activated sludge process aimed for nitrogen removal was established at the Svartliden gold mine, some 30 km from Fäboliden.

The tests were running on a very similar waste water from this mine, during the fall of 2006. In December 2006 substantial results were achieved that encouraged to continue the planning with an activated sludge system for the external treatment facility. The plant configuration will be based on the Sequencing Batch Reactor (SBR), one of the most acknowledged biological nitrogen removing processes.

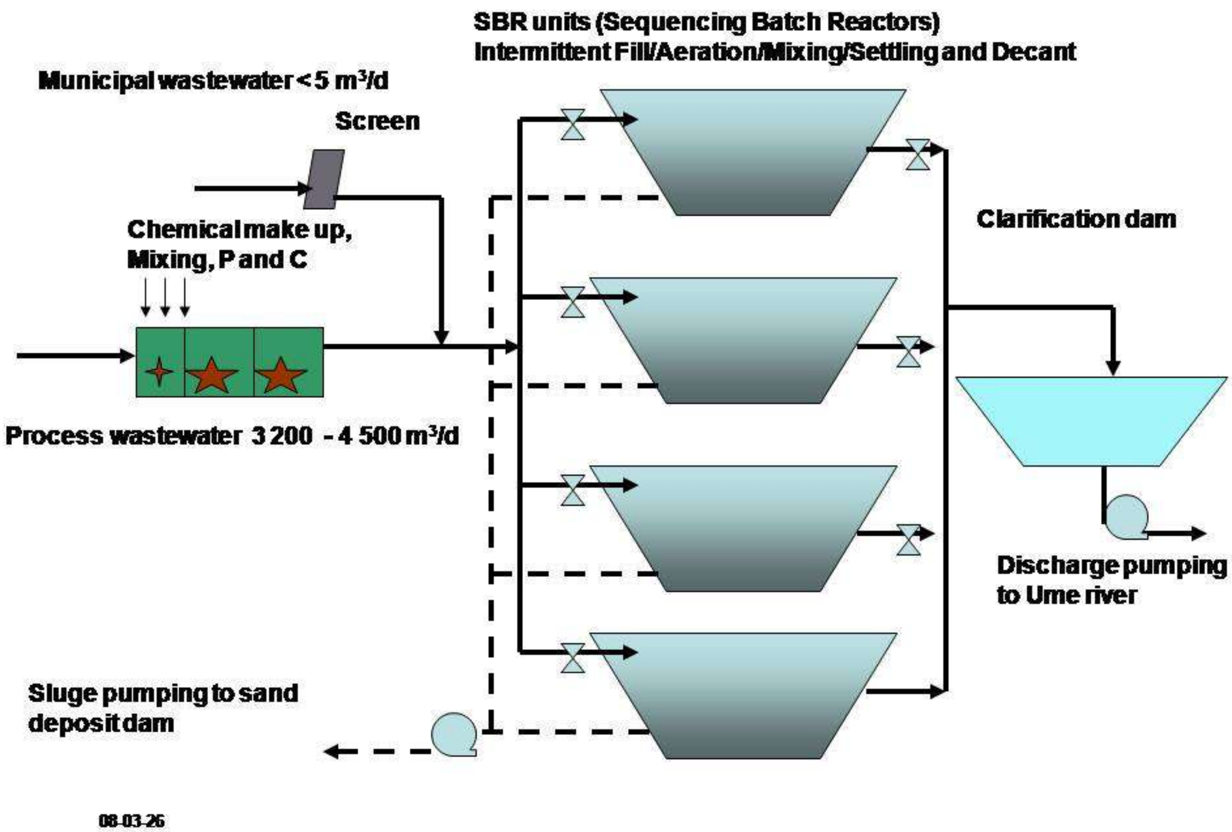


Figure 7.14 Simplified process scheme for the SBR plant in Fäboliden

Simplified process description

Process water rich in ammonia will be directed to the treatment plant from two points in the gold enrichment process:

- Water from the gravity thickeners, especially during winter conditions when the outside temperature is low. On the other hand is the water from the thickener always at about 15 ^{o}C.
- Water from an external “process water” pond, during summer conditions, when the ambient water temperature is 12 – 15 ^{o}C.

The treatment plant will contain the following steps:

- A flash mixing chamber where phosphoric acid is added along with pH-adjustment agent – either acid or alkali.
- The chemicals are then efficiently mixed into the wastewater in three chambers in series.
- A separate line will receive domestic wastewater from the staff inside the facility. This stream is treated by a fine grade screen, prior to the mixing with the main stream in the mixing stage.
- The water is then distributed to a split box, where the water is directed by a process automation system to the main treatment.

- The main treatment contains four SBR-units which are operated on preset time and level directed systems. Oxygen is supplied for the oxidation of ammonia nitrogen to nitrate, and then into nitrogen gas by adding an agent containing easily degradable organic carbon, for instance ethanol.
- After three phases within the SBR-reactor – mixing, aeration and mixing with the organic carbon – this starts the settling phase in the reactor. This is done under "ideal" conditions, with no disturbing factors, such as incoming or leaving water or recirculation streams. This phase takes normally about 40 minutes.
- The final phase in the reactor is the decanting phase when the purified water is discharged to the effluent pond. It will also be possible to recirculated the purified water into the process plant

The process will produce excess sludge by convention. This sludge will be pumped to the sand storage pond.

Key data for the plant
The following data serves as a basis for the design of the plant:
- Design water flow 370 m^3/h.
- Normal daily flow 9,000 m^3/d.
- Design amounts of NH_4-N 1,665 kg/d.

The expected quality of the treated water from the plant will be as follows:
- Discharge concentration of nitrogen, as total N < 20 mg/l
- Discharge concentration of phosphorus, as total P < 0.30 mg/l

Main plant data:
- Four SBR-units each one of 4,000 m^3
- Aeration capacity 4 * 390 kg O_2/h
- Decanter capacity 4 *1,100 m^3/h

8. Infrastructure

8.1 Introduction

Fäboliden is situated in an area with excellent infrastructure such as roads, telecommunication and broadband. A new built 130 kV power line will provide the site with the required electricity. In addition, Lappland Goldminers is involved in a Wind Mill Project located in Fäboliden close to the process plant, which in a number of years might turn out to be an alternative solution supplying the process plant with environmental friendly energy to a long term steady price. Water pipes to Öre älv and Umeälven will be built. Office, maintenance buildings and storages will be situated inside the industrial area as well as dams for process and clear water.

8.2 Dams

Process water reservoir

The Process water reservoir serves as a supply of process water to the processing plant. The demand for stored water is about 1.5M(m^3). The volume is determined in order to secure the operation during winter freezing of the natural runoff to the system. The volume includes an extra capacity to store floods (return period 10 years) when there is a need to discharge water from the tailings reservoir. The water contains some chemical compounds, thus discharge from the process water reservoir will not take place under normal conditions.

The reservoir is located about 500 m south of road no. 360 and west of the road to Fäboliden village. The process water dam is a conventional rock fill dam with a vertical impervious core of till. The downstream embankment can be composed of waste rock or a combination of waste rock and till. Between waste rock and till and between waste rock and foundation fine and coarse filters are needed. The choice of embankment material is first and foremost dependent on material available at the beginning of the construction process. To minimise the dam height, the upper edge of the core will be isolated against frost damage.

Clear water pond

The clear water pond is a water reservoir that collects purified water. The purified water is pumped to the recipient Umeälven. If there should be a demand due to process water shortage, the water can be pumped back into the process. The localisation of the clear water pond was governed by the benefit of sharing dam walls with the process water reservoir.

8.3 Civil Work, Roads and Pipelines

Roads

The roads which belong to the plant have a superstructure which is mainly made of crushed rocks from the site. A grading plan sifter is used to produce the correct fraction to the different superstructure on the roads and plans on the plant. Most important are the entrance road, interchange and road crossing with the Fäboliden road. The roads and plans have a surface course of gravel except on the interchange where its asphalt and on the helicopter platform which is made of concrete. Dewatering of the roads and plans are constructed with ditches and storm water pipe systems with street inlets.

The entrance road and the interchange are constructed with rules of Swedish Road Department "ATB-väg", the frost heave designs the super structure. The truck roads are designed after traffic weight.

The Fäboliden road crossing has a traffic warning light which will be the safety arrangement for the road crossing.

A proposal to lower the speed limit to 50 km/h will be made for the interchange on road 360 and the Fäboliden road crossing.

Pipelines
The pipelines are as follows:

- Pipes in the plant area
- Pipes connecting to the dams.
- Pipes connecting to Umeälven
- Pipes connecting to Örån

Reinforced plastic pipes are used making it possible to reuse excavated material as refilling. It's only the bottom layer that has to consist of smaller fractions. The depth for the pipeline varies depending on the pipeline, ground surface, geology and temperature. Most of the pipelines are pressure pipes, which are regulated with pumps. The pumps are dimensioned by the different flows in the pipes. The main pump station is placed below the process water reservoir.

The Umeälven pipeline has a length of 17,800 pipeline trench meters. The pipelines´ intake and outlet in Umeälven is especially formed to minimize the impact at the river and its environment. The pipeline to Örån is shorter, approx. 2 km.

8.4 Buildings

Civil work -buildings
The buildings in general are designed with substructure and ground floor of concrete and a frame work of precast concrete columns and beams, structural steel roof trusses and overhead crane beams. The cladding will be insulated precast concrete sandwich elements and roofing internal and external trapeze corrugated metal sheet with mineral wool insulation.

The office building will be as a module type temporary office, to be used as site office during construction period and converted to a permanent office upon completion of contract.

Fresh water supply
Two deep drilled holes provide freshwater. Each well has its own pump acting as a backup unit. Pipes distributing water to the Office building, Concentrator, Maintenance and Waste Water Treatment Plant Buildings (WWTP) will be installed at a frost free depth.

Hot water will be produced by using a heat exchanger for the Office building. A circulation system for hot water is to be installed to reduce water consumption with reducing waiting time. The other buildings will have electric heaters for hot water.

Industry water
Recovered water from a cistern in the Concentrator Building is distributed to the Maintenance building and the Waste Water Treatment Plant for use in fire hydrants and in Truck Washing unit. Industry water is also used in the concentrator process.

Sewage
Sewage from the buildings is lead to the Waste Water Treatment Plant. A separator of grease from the kitchen will be installed. Sewage from the Maintenance Building and Truck Washing unit will be lead to an oil separator. Buildings will have pipes for roof drainage installed inside the buildings.

Building heating
Hot water for heating the building will be produced in the Concentrator Building. A small district heating system connecting the building distributes the heat. An exchanger for heat and hot water production is installed in the office building. The Maintenance building is also heated with use of this system.

Waste Water Treatment Plant is heated with electrical radiators. During normal operation warm air from blowers and compressors will heat up the building.

Radiators are installed in offices and the restaurant. Air heaters are used in Maintenance building. To keep the warm air inside powerful air curtains are used at the gates.

Cooling
No comfort cooling will be installed. Kitchen uses stand alone refrigerators and freezer.

Compressed air
Mechanical workshop and Sewage Treatment Work are equipped with a compressor each. Pipes distributing compressed air will be installed around the walls.

Ventilation
All buildings will be ventilated with mechanical air treatment units. Heat recovery for these units will be installed, that means, warm exhaust air will preheat used cold outside air.

Work areas for welding are equipped with smoke extractors served by a separate fan.

Power supply in general
Power supply by 400V, TN-S system, provided by cable systems are mostly in the ground and indoors by trays.

System Design, Electrical Outdoor areas
Outdoor area contains the following electrical installations:

- Power supply by cables between switch gear in Process building and electrical
- board in lockers
- Power supply by cables between switch gear in Process building and pump
- stations (3 pcs)
- Power supply by cables between 10/0,4kV transformer and switch gear in Waste Water Treatment Plant Building (WWTP)
- Power supply by cables between switch gear in WWTP and electrical board in the Maintenance building
- Power supply by cables between electrical board in Lockers and staff parking

- Power supply by cables between electrical board in Maintenance building and truck parking
- Outlets for staff parking and truck parking
- Lighting of internal roads, connection of gate, lighting of working areas by search light mounted on top of a mast
- Light and outlets (1*32A and 4*10A) for helicopter area
- Communication cables by optic fibre and manoeuvre cables (FKAR-PG 48*2*0,5) parallel with power supply cables
- Electrical Bonding (EB) connected to main connection-point in room for switch gear and parallel with power supply cables
- Fence alarm including software
- CCTV including software

Lockers, Dining and Office building
Lockers, Dining and Office building contain the following electrical installations:

- EB by Cu-wire except concrete foundation and reinforcement
- Electrical board

Maintenance building including Cold storage and Recycling
The maintenance building contains the following electrical installations:

- EB by Cu-wire except concrete foundation and reinforcement
- Cable trays for all electrical installations except office area
- Electrical board, one for each part, for outlets, lighting, cranes, tools, HVAC cabinet, main doors
- Outlets in normal extent for general purpose and for tool equipments such as welding, drilling, compressed air units
- Lighting fittings with SON-T lamp in maintenance, with fluorescent lamp in offices, cold storage and recycling room
- Wall mounted lighting fitting located to doors and entrance
- Outlets for communication (computers and telephones) in the office area

Waste Water Treatment Plant Building
Waste Water Treatment Plant Building contains the following electrical installations:

- EB by Cu-wire except concrete foundation and reinforcement
- Cable trays for all electrical installations
- Switch gear for power distribution to maintenance building and waste water process plant
- Electrical board for outlets, lighting, tools, HVAC cabinet
- Outlets in normal extent for general purposes
- Lighting fittings with fluorescent lamp
- SON-T lamp and outlets for maintenance purpose for outdoors basins
- Wall mounted lights fittings located to doors and entrances

Building Management System (BMS) installations, System overview
The BMS system controls the HVAC systems and partly alarms/control of the electrical systems. The system is built up by PLC´s communicating via the office computer local area network,

Ethernet TCP/IP. Alarm handling, historical data, dynamic flow charts etc are reached by an ordinary web browser in any PC connected to the network. Each building will be fitted with a PLC which serves all the HVAC systems in that particular building.

Communication
Each PLC will contain a communication module for Ethernet TCP/IP. Communication between all PLC´s and to other process systems will be trough the common fiber optic network connecting all buildings on the site.

PLC
The PLC´s will all have a built-in web server which provides all necessary information to the user. Dynamic pictures, historical data, user data, alarm statistics, event statistics etc is stored in the PLC memory. Apart from system of I/O-, power- and communication modules PLC´s will also have the ability to add distributed I/O-modules and other intelligent components via a field bus system. Choice of protocol should be Profibus, Modbus or any other established protocol on the market.

Control cabinets
Each mechanical room will be fitted with a cabinet containing:

- PLC
- I/O-modules
- Communication module
- Operation panel fitted in cabinet front
- Fuses and other electrical components
- Motor starters
- Frequency converters and soft starters will be placed as near as possible to served electrical motor.

System connections
The BMS system will be connected to the production automation system via Profibus DP or Modbus. The BMS system will be connected to the waste water automation system via Profibus DP or Modbus. Waste water process will be visualized in BMS system.

Alarm
Alarms, generated in the BMS system, will be indicated on each operation panel/PLC. The alarms will (can) also be rerouted to the following destinations:

- E-mail
- Production automation system
- SMS to mobile phone

9. Environment and Tailing Storage Facilities

9.1 Introduction

It is a well known fact that mining activities has an impact on the environment. In the immediate surroundings of the site in Fäboliden there are areas that are protected due to their nature and historical values. This has resulted in different kinds of mitigation measures to insure that the negative impacts caused by the mining and processing are minimized. The company has in several respects aimed at “green” and environmental sustainable solutions for the planned activities.

This chapter of the feasibility study considers a concise description of all the permits that have been required. Further on the purpose and function of water management and remediation is described. These are the two most cost demanding preventive measures for the environment. More detailed descriptions - about the surroundings, how the mining area might impact on the environment and what measures will be taken by the company - is assembled in the Environmental Impact Assessment (EIA) that together with the permit application has been submitted to the environmental court.

There are specific demands and legislation that regulates management of hazardous substances that are described in this chapter. Environmental monitoring which is an operation cost is also described below.

9.2 Permits and Regulation

The process of developing a mining project such as Fäboliden is regulated through permits and reports in several steps. Application for permits is handled by different authorities. In the table below the permit process for the mining area in Fäboliden has been summarized chronologically.

Date	Permit /regulation	Legislation	Authority	Comments
Several occasions since 1999	Exploration permit	Minerals´ Act (1991:45)	Mining Inspectorate	The company has 9 exploration permits in the Fäboliden area
	Exploration work, with significant impact on the natural environment	Environmental Code (1998:808)	Supervisory authority (The County Administrative Board)	
	Working plan	Minerals´ Act (1991:45)	Landowners and other parties affected, if disagreement occurs Mining Inspector handle the case	
2002	The gold mineralization in Fäboliden is declared to be a mineral deposit of national interest	Environmental Code (1998:808)	SGU (Geological Survey of Sweden), Swedish Environmental Protection Agency and The County Administrative Board	
June 2004	Exploitation concession	Minerals´ Act (1991:45) and Environmental Code (1998:808)	Mining Inspectorate together with The County Administrative Board	

January 2005	Mining test permission	Environmental Code (1998:808)	The County Administrative Board	100 ton ore
October 2007	Application: Mining establishment permission; mine, process plant, water treatment plant, waste rock deposit and tailing reservoir	Environmental Code (1998:808)	Environmental court	Application submitted to the Environmental court in April 2006
17th of December 2007	Master Plan		Lycksele Municipality	
21st of December 2007	Environmental permit		Environmental Court	
May 2008	Appeal of Environmental Permit		Supreme Environmental Court	
2008	Designation of land		Landowner and Mining Inspector	
2009	Building permit	Planning and Building Act (1987:10)	Local Authority, Lycksele Municipal	

Table 9.1 Summary of the permits and regulations process for the development of Fäboliden

9.3 Fees

The fee for environmental supervision is regulated by legislation (SFS 1998:940). The fee for the activity in the mine and process plant is 187,000 SEK, for the deposit activity 46,750 SEK and for the water treatment 28,250 SEK which gives a yearly environmental supervision fee of 262,000 SEK. This is included in the overall operation cost for the planned mining area.

9.4 Water Management

Water is used as a transport medium for grinded ore from the mills through the process. Slurry with tailings and water is deposited in the tailing reservoir. Water from the reservoir is recirculated via the process water reservoir back to the process plant. Water is also recirculated in the process plant via gravity thickeners.

Water balance

Water balance calculations have been conducted for Fäboliden mining area. The main purpose is to verify the water availability for the process and the need of discharging water from the system. The natural run off has been calculated with a hydrological rainfall-runoff model where daily precipitation, potential evaporation and temperature have been used as input data. The results have been compiled to monthly values. The main conclusion is that there are adequate water resources in the area to supply the processing plant with production water and to retain open water on the HR tailings after termination of the mining operation.

Four water courses will be affected by the mining area (Umeälven not included in the list since the effect is neglectable):

- Lördagsbäcken will be most affected since the catchment area will be reduced with about 75%.

- Fäbodbäcken´s catchment area will be reduced with about 30% when the tailings reservoir has reached maximum size.
- Storbäcken´s catchment area will be reduced with about 7%
- Örån´s total water volume will be reduced by 3.6 % as a result of the water transfer to Umeälven.

The discharge of water is estimated to 3.7 M(m^3)/year under normal conditions. For dry and wet years (return period 100 years) the corresponding values are 2.4 and 5.0 M(m^3)/year. The discharge of water will continuously be pumped from the clear water pond to Umeälven. The water transfer pipe has capacity to transfer the inflow to the reservoir system during a wet year with a return period of 100 years. For a wet year with a return period of 1000 years, discharge to Örån is necessary.

Water treatment
The discharge water is polluted from the enrichment process and needs treatment before it reaches a natural recipient. An external biological treatment facility will be built for the removal of the nitrogen from the wasted process water. The plant configuration will be based on the Sequencing Batch Reactor method, one of the most acknowledged biological nitrogen removing processes.

9.5 Remediation

The remediation is done to adapt the mining area to a natural life state. The target perspective in Sweden regarding this is the next glacial period. The international practice regarding remediation of mining areas is that it should be sustainable for 1000 years or more.

The remediation of the mining area will be implemented with best available technique (BAT) at the time of termination of the mine. Therefore, it is difficult to specify in detail how the area will look after remediation. The remediation of open pit, industrial areas, reservoirs and waste rock deposit will be conducted with minimum affect on the surrounding environment and water courses.

Area/object	General method
Open pit	Water filling and leveling of slopes
Industrial area	Surface clean-up and till cover
Tailings reservoir	Water cover of the HR part and creation of sand heath with landscape adaptation and vegetation
Process water reservoir	Dam cut open, surface clean-up and till cover
Clear water pond	Dam cut open, surface clean-up and till cover
Waste rock storage	Possibly till cover
Diabase storage	Possibly till cover
Temporary storage of HR waste rock	Surface clean-up and till cover
Till storage	Vegetation

Table 9.2. Summary of general remediation techniques to be applied at different parts of the mining area.

Open pit

With the planned quarrying methods the open pit cannot be refilled with waste rock. The remediation of the open pit will consist of water filling by natural runoff after the pumping has closed down. The rock surface around the pit is cleared in order to receive a plateau. The distance from this plateau to the water will differ since the open pit follows the ground and the water surface is horizontal. The till slopes around the pit will be leveled out. If necessary, material from the till storage can be used to establish vegetation on the till hills. At steep rock slopes, protection walls of rock will be constructed. To make the open pit to act as a natural lake, an outlet will be designed to allow controlled discharge from the pit.

Industrial area

A thin layer (appr. 0.5 m) of the ground surface will be excavated and deposited in the open pit. The remaining ground will be covered by a thin till-layer (appr. 0.3 m on 60 000 m^2) and vegetated. Since the industrial area is located on a till area, vegetation may be established without refilling of new till. The cost for removal of equipment and removal of buildings will be covered by selling vital parts of the mining equipment.

Tailings reservoir

The selection of remediation method for the tailings reservoir included an evaluation of several techniques. All methods are considered to fulfill the demands on minimized sulphide weathering. The evaluation covered the following methods:

- Permanent water cover.
- Dry cover (0.3 m compacted clayey till combined with 1.5 m protection cover of unscreened till).
- Raised groundwater table.
- Deposit in open pit below the ground water surface.

Conventional dry cover of 5 km^2 is considered too expensive in relation to the environmental benefit.

Deposit in open pit is not a realistic solution in Fäboliden as the ore deposit continues at deeper levels. Open pit deposition would put a halt to further excavation of the pit.

A groundwater model has been set up in order to assess the conditions for a raised groundwater table. The results show that the conditions at Fäboliden are suitable for retaining saturated conditions in the tailings reservoir. Specifically, in the area of HR tailings, there is an upward gradient of the groundwater flow. The hydraulic gradient in the deposited material is higher than in the underlying till, which causes a major horizontal groundwater movement through the LR tailings out through/under the dams.

The water cover alternative is regarded as a better alternative than the raised groundwater table, mainly because of a reduced need of natural resources (till). Establishing a raised groundwater table would create a need for an additional till volume of 3 M(m^3) that would be deposited on the tailings reservoir. Furthermore, the water cover gives better opportunities to create a landscape similar to that of today. Today there is a peat bog with small open water surface, which can be "re-created" by using the water cover method as the remediation alternative.

If the production is completed after 15 years (5 Mton/year), the tailings reservoir is expected to cover 5 km^2. The tailings reservoir will at this stage be delimited by two natural hills and two 40 m high dams. The reservoir will contain about 50 $M(m^3)$ tailings of which 1.5 $M(m^3)$ consist of HR tailings. Except for the tailings, HR waste rock will constitute <17 $M(m^3)$. The total volume stored in the tailings reservoir is 65 $M(m^3)$.

The proposed remediation program has taken the load on environment, risks and costs into consideration. A volume of 48.5 $M(m^3)$, about 75% of the reservoir volume consist of LR tailings which has buffering properties. This material will enclose the HR material which helps minimize the presence of leakage water with high contents of acids and/or metals as the LR tailings acts like a filter. The “filter” will be at least 100 m wide. To minimize the oxygen supply to the deposited HR material, this part of the reservoir will be water covered. The reservoir will also be covered by 2 m of LR tailings by rerouting the distribution of the final part of the LR tailings.

The major part of the reservoir, about 70%, will be remediated to a sand heath. A profound study has been conducted in order to define the potential of mobilizing metals in the tailings reservoir. The results show a substantial chemical weathering of arsenopyrite from the LR tailings during the first 10-20 years. The weathering of the material means that measures has to be undertaken to minimize the leakage of hazardous discharge from the tailings reservoir to the recipient. For example Iron Sulfate will be added to the tailing in order to bind free arsenic ions.

Described, from the northern dam towards the eastern dam, the following measures will be taken during the remediation phase:

- The downstream slop of the northern dam will be covered with till and sowed with some kind of suitable grass. Upstream, between open water and the dam construction a forest curtain will be planted, which will demand approximately 0.5 m thick till coverage over the tailings surface. Parts of the tailings surface will be remediated through water coverage which will give rise to a new lake, “Fäbodsjön”, surrounded by plants commonly seen in the vicinity of open water and wet lands. A couple of wooded islets will be constructed in the lake. Remaining areas south of the water coverage will be covered by a thin layer of till (5-10 cm) enabling establishing of vegetation. As protection against surface erosion a relatively large amount of plant curtains and groves will be planted. These will also offer shelter to wildlife. The required till thickness for establishing of groves and curtains is approx. 0.5 m.

Except for the area covered by trees and bushes, a large part of the dam area is thought to be covered by grass and lichen. To guarantee that the choice of plant types is suitable some test cultivation are required. These are proposed to be made during the time of operation. The final method of remediation will be shaped during time and will be adapted to latest technology and new knowledge acquired by amongst others these tests. Hence, in this stage the measures proposed to be carried out in the remediation phase of the tailings reservoir are:

- Adjusted deposition of LR tailings during the last six months to cover the deposited HR tailings.
- Placing of till and maybe waste rock as wind protection and landscape improvement.
- Establishing of vegetation on the reservoir area (trees, bushes, grass and lichen).

- Preparation of collector ditches for runoff and leakage, with reactive barriers which purpose is to accomplish adhesion of solved metals. Initially outgoing water from the tailings dam will be processed in the water treatment plant.
- Removal of surrounding drainage ditches.
- Securing of a long time stable system for controlled diversion of run off, including suitable spillway. Approximately one meter is proposed as a suitable water depth in the reservoir.
- A till coverage of dam slopes. This can be done continuously during the operation time of the facility.
- Removal and refilling of leakage control wells. This can be done when satisfying results of the remediation of the tailings dam has been verified. Establishing of vegetation on dam slopes can be started and carried out gradually during the operational phase of the facility.

Process water reservoir
The process water reservoir will, after the mining facility has been closed, be emptied. The water will be led to the water treatment plant, treated and released to the recipient in a controlled way. The dam construction surrounding the emptied reservoir will be removed in the lowest point to prevent any accumulation of water. The remaining dam construction will be remediated to resemble a natural ridge. This will be achieved by a 0.3 m thick till layer on the dam wall.
The reservoir area (approx. 0.21 km^2) will be covered with till and vegetation will be established. The uppermost 0.5 m will if necessary be excavated and the material deposited in the open pit. The excavated area will then, if needed, be covered with a 0.3 m thick layer of till before the establishing of vegetation.

Clear water pond
The clear water pond will be abunded after the closure of the process water reservoir. The pond will be emptied through controlled release of water to the recipient. When the pond is emptied the dam construction will be removed in the lowest point to prevent further accumulation of water. The remaining dam construction will be remediated to resemble a natural ridge. This will be achieved by a 0.3 m thick till layer on the dam wall. The enclosed pond area (approx. 0.025 km^2) will be covered if needed, i.e. if the pond bottom is polluted by sediments, with a 0.5 m thick layer of till and then be vegetated.

Waste rock deposit, LR waste rock
The waste rock deposit is designed to contain <76 Mt. It will be built up by three benches with the material deposited in a natural angle of repose (slope 1:1).This will response in an average slope of 1:2.2 since each bench will be set back 20 m in relation to the prior. The remediation of the LR waste rock deposit result in either no remediation measures or establishment of a till cover (0.3 m). The till cover will be vegetated and a system for diversion of surface run off will be constructed. All infrastructures will be removed.

Depending on how much waste rock that is classified as HR material and the proportion of marginal ore that is processed, the volume of waste rock may be smaller than 76 Mt after 15 years of operation.

Diabase deposit

The diabase that is cut out in the open pit (approx. 5 Mt) will be deposited in a separate deposit. The aim for the deposit is to sell it as construction material. Three scenarios at the time of closure of the mine are:

- All material is sold. Only a surface clean-up is needed.
- There is some diabase left after closing the mine. The liability for the deposit can be passed over to local establishment for selling.
- The entire diabase deposit is left. In this case, the same remediation measures as for the LR waste rock will be applicable.

Temporary deposit of HR waste rock

The temporary deposit of HR waste rock will be cleared off by removal of the surface layer to the open pit. The ground will be covered by a layer of till (0.3 m) and vegetated.

Till deposits

Within the mining area two till deposits will be established. One with till from stripping off the open pit, which will act as a noise protection during the operational phase. Another is located at the western part of the tailings reservoir consisting of material stripped off from the dam foundations. These deposits will contain 4 and 1 M(m^3) respectively. During the remediation these deposits will be used as surface cover at other deposits in the area. The total need for till cover is estimated to 2 M(m^3), of which 1 M(m^3) is assigned for the tailings reservoir. The distribution of the till in the remediation plan is presented in Table 3.

Remaining till deposits will be landscaped to natural forms and vegetated. Some measures to control surface run off and temporary sediment traps may be necessary during the initial phase after closure of the mine.

Object	**Principal method**	**Till (m^3)**	
		min	**max**
Open pit	Water filling and leveling	0	30 000
Industrial area	Surface clean-up, unqualified cover	0	20 000
Tailings reservoir	Sand heath, landscaping, vegetation	625 000	1 125 000
Process water reservoir	Dam cut off, surface clean-up, unqualified cover	18 000	130 00
Clear water pond	Dam cut off, surface clean-up, unqualified cover	3 000	23 000
Waste rock deposit	Unqualified cover	0	300 000
Diabase deposit	Unqualified cover	0	50 000
Temporary (HR) waste rock	Surface clean-up, unqualified cover	0	25 000
Total demand:		**646 000**	**1 703 000**

Table 9.3. Compilation of till demand for remediation of the mining area.

Control and supervision

As the remediation is carried out, appropriate control and supervision is needed for some time. The time is different for different parts of the area, but at least 30 years after closure is required. Significant deviations from the remediation plan will be adjusted during this time.

Costs
The cost for the remediation plan is estimated to 56 MSEK, as stated in the environmental permit Lappland Goldminers received December 21st 2007. Of this 5 MSEK has to be paid when the permit is taken into use and 20 MSEK has to be paid before construction works start.

Hazardous substances
Uses of hazardous substances – e.g. the amount of cyanide, sulfur oxide and explosive - in the planned mining area total the amount that makes the establishment comprised by the so called SEVESO legislation (SFS 1999:381 and SFS 1999:382). That result in increasing demands on risk assessments and safety measures during the planning phase as well as during the operating phase, including a safety report.

Safety issues have to be part of the planning process for the mining area. A preliminary safety report has been conducted together with the environmental permit application. A final version will be submitted for approval to the Swedish Work Environmental Authority six month before operation start according to current regulations.

The safety report contains an internal rescue plan which will finally be established after several consultations with the local rescue force of Lycksele Municipality. Cost for the work with safety report and risk assessments are included in the overall operation cost.

Monitoring
The company will continue to monitor the surrounding environment. The monitoring will be performed according to the environmental monitoring program that will be established for the mining area. Previous monitoring of water chemistry, bottom fauna and fish in nearby streams in Örån/Öreälvens catchment area will continue. The monitoring of river Umeälven has just began and will continue with water chemistry control points up and down streams the discharge of water from Fäboliden.

The yearly cost for environmental monitoring is estimated to 200.000 SEK.

10. Capital Costs

The capital costs are summarized in the table below and in the enclosed specifications. Some basic assumptions for the calculation are:

- The industrial area has been designed with high standards, implying long technical life. The design is described in chapters 7-9.
- The designed capacity for the plant is 4.6 MTonnes per year.
- The prices for different kinds of equipment in the process plant have been required through tenders from single suppliers, which mean that the tangible cost increases on mechanical equipment during the recent years (20-50%) have been covered for.
- Contingencies have been assessed to 10% of the capital cost.
- Engineering, procurement and construction management (EPCM) has been assessed to 8% of the capital cost.

Evaluating the assessments made by Outotec AB, Lappland Goldminers has come to the conclusion that the proposed design is slightly overestimating the technical standards and technical life required for the mine and process plant in Fäboliden. Due to this the Company has decided to slightly lower the technical demands for infrastructure and buildings and to supply the plant with some used equipment (an example of this is the recent procurement of a re-grinding mill which have meant a 50% reduction of the calculated cost). Through such a strategy, not affecting the capacity or environmental commitments, it has been assessed that the calculated costs can be reduced by 10%.

As can be read from the compilation of capital costs on the next pages:

- The total capital cost is assessed to approx. 1 532 000 000 SEK.
- Crushing, conveying and grinding correspond to some 40% of the total cost.
- Infrastructure, such as buildings, dams and roads, corresponds to some 25% of the total cost.

Compared to the Pre-Feasibility Study (December 2005) the total cost (capital cost, mining equipment for the open pit and closure cost) has increased from 1248 Mkr to 1826 Mkr. The increase of almost 600 MSEK is explained by:

- Increased environmental demands identified during the environmental permit process, approx. 200 MSEK
- Price increases in general on mechanical equipment, approx 150 MSEK
- Increased EPCM and Contingencies due to higher investment cost, approx. 75 MSEK
- Other: Cost for technical equipment not covered for in the Pre-Feasibility Study, approx. 175 MSEK.

Table 10.1 Capital Cost Summary

Crushing and conveying			
	Mech	KSEK	115 238
	Power	"	10 333
	Process Control	"	3 538
	Civil	"	97 006
	Subtotal Crushing and Conveying		**226 115**
Grinding			
	Mech	KSEK	207 879
	Power	"	16 321
	Process Control	"	7 563
	Civil	"	83 591
	Subtotal Grinding		**315 354**
Flotation			
	Mech	KSEK	66 197
	Power	"	12 667
	Process Control	"	4 150
	Civil	"	46 932
	Subtotal Flotation		**129 946**
Leaching + Tailings			
	Mech	KSEK	153 625
	Power	"	29 850
	Process Control	"	10 892
	Civil	"	61 251
	Subtotal Leaching + Tailings		**255 618**
General			
	Mech	KSEK	0
	Power	"	17 467
	Process Control	"	32 814
	Civil (Control room, not specified)	"	43 048
	Heating, ventilation & air condition (HVAC)	"	45 600
	Subtotal General		**138 929**
Infrastructure			
	Office building	KSEK	17 875
	HVAC	"	2 903
	BMS	"	1 260
	Power	"	1 870
	Maintenance buildings	KSEK	42 298
	HVAC	"	3 085
	Power	"	2 027

	Waste water treatment	KSEK	7 633
	Mech	"	31 577
	HVAC	"	536
	Power	"	600
	Water outside (dams, spillways etc.)	KSEK	67 350
	HVAC	"	1 105
	Outdoor lighting etc.	"	4 228
	Roads and Outdoor pipes	"	84 200
	Subtotal Infrastructure		**268 547**
Tailing area			
	Tailing dams	KSEK	**84 000**
	Tailing dams, water spray system	"	**1 975**
SUBTOTAL			**1 420 484**
Reduction due to overstandard (-10%)			**-142 048**
Contingency (10 %)			**127 844**
Equipment Spares			**24 169**
EPCM (8%)			**102 274**
TOTAL CAPITAL COST, KSEK			**1 532 724**

11. Operating Costs

11.1 Process Plant

Cost for staffing comprises a management group including a production manager, two engineers and one administrative resource. Further on the shift operations are assumed to be performed with six teams, i.e. holidays are included in the roster. In total there are five persons per shift, two in the crusher, one in the grinding circuit, one for flotation, magnetic separation and tailing pond, and finally one in the CIL-circuit, detox and water treatment plant. One of the operators will also have a position as shift leader. Daily operations as gold room, chemical handling, shift reserves etc. will be performed with seven operators working Monday to Sunday.

The major consumption of electric energy takes place in the grinding cirquit. Almost 100 tonnes of ore has been tested at Minpros pilot plant in Stråssa. Autogenous grinding has been performed and the finess of the product has varied between 51 and 115 micrones, k80. At the planned k80=90 micrones, the energy consumption will be maximum 24 kWh/ton. Regrinding of arsenopyrite concentrate or the scavenger product in the flotation circuit is calculated to 20 kWh/ton, corresponding to 1 kWh/ton ore.

According to Sandvik the crusher will consume 2.3 kWh/ton.

In the flotation- and the CIL systems agitation and compressed air take 6.0 kWh/ton according to machine lists.

The consumption of chemicals in the flotation is established in pilot plant tests at Minpro and as for leaching, by bench scale tests, 10 kgs, performed by Jan-Erik Sundqvist et al.

The operation costs in the water treatment plant are calculated by Sweco on the basis of pilot plant tests.

During the first five years of operation the major maintenance cost will be wear parts, mainly those in the crushing and the grinding systems.

The operating costs for the process plant are summarized on the next page.

Table 11.1 Operating costs process plant

Personal	Number of empl.	SEK/y	SEK/ton	
Management	4	2.900		
Shiftop.	24	10.800		
Dayop.	7	3.000		
Maintenance	6	2.900		
Sub Total	*47*	*19.600*	***4.30***	
Energy	kwh/ton			0.30SEK/kWh
Crushing	2.3	3.220	0.70	
Grinding	24.0	33.120	7.20	
Regrinding	1.0	1.380	0.30	
Flotation	3.0	4.140	0.90	
CIL	3.0	4.410	0.90	
Others	1.7	2.346	0.51	
Sub Total	*35*	*48.616*	***10.50***	
Chemicals	g/ton			price kr/kg
Cyanide	790	43.608	9.48	12.00
Lime	3060	12.650	2.75	0.90
Cu-sulfat	190	10.488	2.28	12.00
SO2	940	5.198	1.13	1.20
Xantat	120	5.520	1.20	10.00
MIBC	14	1.472	0.32	1.60
PbNO3	11	1.012	0.22	20.00
Soda	27	0.460	0.10	3.70
LPG	95	1.748	0.38	4.00
Carbon	26	1.564	0.34	13.00
Others	17	0.368	0.08	
Fe-sulfat	91	2.300	0.50	5.00
Sub Total		*86.388*	***18.78***	
Water Treatment plant		9.200	**2.00**	
Parts				
Crusher		9.500	2.06	
Mill		9.200	2.00	
Sub Total		*18.700*	***4.06***	
Others		6.000	**1.30**	
Total			**40.94**	

11.2 Mining

The staff for the mine will include the following positions:

- Manager
- Mine Superintendent
- Mine Geologist
- Surveyor
- Grade control
- Shift boss
- Driver excavator
- Driver truck
- Driver wheel loader
- Driver grader
- Driver dozer
- Driller
- Driller grade control
- Charging people
- Dewatering and service
- Electrician
- Service and repair

Operations are expected to be ongoing 24 hours a day and all around the year. Shift operations are assumed to be performed with 6 teams. The cost for staffing corresponds to approx. 30% of the total operating cost for the mine.

Costs for diesel and explosives constitute major part of costs, corresponding to some 20% of the total respectively.

Estimated operating costs for mining are summarized below.

Table 11.2 Operating costs mining open pit

Mining	SEK/tonne	Annual cost	Number of employees
Drill and blast	**2,68**	53 600 000	
Explosives	1,89	37 800 000	
Bits and Rods	0,25	5 000 000	
Service	0,19	3 800 000	
Personnel	0,35	7 000 000	13
Load and Haul	**5,80**	116 000 000	
Personnel	1,60	32 000 000	57
Diesel	2,20	44 000 000	
External services	1,70	34 000 000	
Miscellanous	0,30	6 000 000	
Maintenance	**1,15**	23 000 000	
Personnel	0,80	16 000 000	31
External services	0,08	1 600 000	
Road mtrl.	0,15	3 000 000	
Lubricants	0,13	2 600 000	
General	**0,89**	17 800 000	
Personnel	0,33	6 600 000	12
External services	0,40	8 000 000	
Miscellanous	0,16	3 200 000	
		-	
SUM	**10,52**	210 400 000	113

As for the underground production the required capital cost is estimated to 278 MSEK, covering parts of the pre production and development works and mobile machinery.

Operating cost for underground production is estimated to approx. 50 SEK/Tonne, covering parts of pre production and development works, production and services.

11.3 Sum of operational costs

Mining Open Pit
SEK/Tonne 10.5
Cost for mining equipment for open pit production is estimated to approx. 240 MSEK

Mining Underground
SEK/Tonne 50
Capital cost for development works and mining equipment for underground production is estimated to approx. 280 MSEK.

Process Plant
SEK/Tonne 41

12. Gold Price

12.1 Refining

Gold is a precious metal and one of the most chemically inert and valuable metals encountered in nature. Gold has received the chemical symbol ”Au” after the Latin name ”Aurum”, which means ”brilliant sunset”. The metal has unique combinations of physical and chemical characteristics, it is extremely tensile and malleable, a remarkable conductor of heat and electricity as well as being strongly reflective. Gold does not corrode, discolour or rust when it is exposed to air or water. This combination of characteristics contributes to a large extent to the value of gold. Gold is used primarily in jewellery production, but is also used within industry, for example, for circuits and microchips, and within dentistry.

Pure gold is called 24 carat or 1,000 parts pure gold, but the metal is then very soft. To increase the hardness, gold is alloyed with other metals such as silver or copper. The most common gold grade in jewellery is 18 carats.

For Fäboliden the final product will contain recovered gold, silver and the possible content of platinum group metals within doré metal bars. Usually doré metal also contains a small percentage of impurities. The doré metal needs to be refined in order to separate the precious metals and to produce each metal with the quality required before the metals can be sold on the spot market.

There are several refinery alternatives available within Europe. Examples of refineries specialized on treating doré and high-grade precious metals in metallic form are Argor-Heraeus, Metalor and Pamp, all with their plants located in Switzerland. Jonson Matthey with a plant at Toronto, Canada may also be an option to consider. Examples of precious metals refinery plants and operations with larger copper smelters are Boliden at Rönnskär in Sweden, Umicor at Hoboken in Belgium and Norddeutche (NA) at Hamburg in Germany. For refining of doré metal, the “specialized” refineries are usually more favourable compared to the terms offered by the “copper smelter refineries”.

Indicative quotations regarding payable metals and refining charges were received from refineries and smelters during February 2007. The cost for the freight plus the refining charges plus the cost for having a mining company representative at the refinery overseeing the weighing and assaying and the costs for assays amounts in the most competitive alternative to approx. 5 US$/oz Au contained in the doré.

When selling precious metals, the unit price used in Europe is the market price reported at the market place, London Bullion Market Association (LBMA). When the doré has been refined and when the payable metals content has finally been agreed upon and settled between the mining company and the refinery, the refinery then will credit the metals to a metal account owned by the mining company and the metals are then ready to be sold on the market. The mining company can then elect when to execute a sale. When sold, the sales revenue typically will be received three days after the date of a sale transaction on the mining company’s bank account. The sale of the refined precious metals does not need any special marketing of the metal quality or any metal price negotiation or contacts from the mining company with the end user.

The mining company also has the option to sell its produced doré to the refinery. Typically in this case is that a preliminary payment for 90% of the gold content is settled on the day the doré shipment arrives at the refinery and that the payment is received on the mining company's bank account three days later. The remaining final payment is then made three days after the refinery and the mining company finally has settled/agreed regarding the sampled and assayed metal content of the shipment.

12.2 Gold Price

Current gold and silver prices of 1st of February 2008 have been used for the financial evaluation of the project:

- Current price Au 933 USD/tr. oz
- Current price Ag 17 USD/tr. oz

13. Financial Evaluation

13.1 Basic assumptions

The financial evaluation is based on the following assumptions:

Basic Assumptions	
Capital cost Process Plant	SEK 1,533 millions
Capital cost mining open pit	SEK 240 millions
Closure cost	SEK 56 millions
Residual value after 11 years	15% of capital cost
Required working capital	SEK 100 millions
Annual process capacity	4.6 Mt
Environmental permit	5.0 Mt
Ore open pit	23 million tonnes
Number of production years open pit	5
Ore underground	30.5 million tonnes
Number of production years underground	6.5
Average Au grade open pit (dilution incl.)	1.2 g/t
Average Au grade underground (dil. incl.)	1.07 g/t
Average Au grade open pit + underground (dilution incl.)	1.13 g/t
Average Ag grade (dilution incl.)	3.2 g/t
Average recovery grade Au	84%
Average recovery grade Ag	50%
Gold price, Au (1st of February 2008)	USD 933 per tr. oz
Silver price, Ag (1st of February 2008)	USD 17 per tr. oz
USD (1st of February 2008)	6.44 SEK
Operating cost process	41 SEK/tonne ore
Operating cost mining open pit	10.5 SEK/tonne
Investment cost underground	SEK 278 millions
Operating cost mining underground	50 SEK/tonne
Interest rate (Net present value)	5%

Table 13.1 Basic assumptions

13.2 Results

Accumulated cash flow, including reclamation costs, but excluding interest and financing costs, is estimated to approx. SEK 3,500 million. Pay-off time is 4 years and the net present value amounts to approx. SEK 2,000 millions. The evaluation is summarized in the table on the next page.

		Accumulated	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mine production																
Overburden stripping	Tonnes	12 093 163	0	3 448 717	4 923 761	1 436 977	2 283 708	0	0	0	0	0	0	0	0	0
Waste rock	Tonnes	84 973 058	0	9 567 992	14 641 009	15 019 153	15 250 408	15 411 536	15 082 960	0	0	0	0	0	0	0
Ore	Tonnes	53 397 048	0	14 902	4 625 743	4 580 796	4 585 056	4 596 866	4 493 685	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	2 900 000
	W/O	1.59														
Processing																
Ore	ton	53 397 048	0	14 902	4 625 743	4 580 796	4 585 056	4 596 866	4 493 685	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	4 600 000	2 900 000
Au grade nominal				0.55	1.11	1.40	1.15	1.12	1.26	1.25	1.25	1.24	1.06	0.93	0.89	0.86
Gold (recovery considered)	kg	51 020	0	6	4 293	5 696	4 427	4 323	4 752	4 793	4 793	4 755	4 079	3 590	3 440	2 074
	Oz	1 640 321	0	199	138 007	183 145	142 327	138 987	152 776	154 094	154 094	152 886	131 131	115 420	110 586	66 669
Recovery Au	%	84.2%	0%	75%	84%	89%	84%	84%	84%	84%	84%	84%	84%	84%	84%	84%
Ag grade nominal			0	0.37	2.51	2.48	2.34	2.54	2.68	2.35	3.32	3.30	3.30	3.30	3.30	3.30
Silver (recovery considered)	kg	86 050	0	3	7 193	6 230	7 336	6 987	7 976	7 590	7 590	7 590	7 590	7 590	7 590	4 785
	Oz	2 766 579	0	89	231 261	200 295	235 861	224 645	256 444	244 024	244 024	244 024	244 024	244 024	244 024	153 841
Recovery Ag	%	50.0%	0%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%
Revenue																
Gold in Doré	Oz	1 640 321	0	199	138 007	183 145	142 327	138 987	152 776	154 094	154 094	152 886	131 131	115 420	110 586	66 669
Gold revenue	k kr	9 855 901	0	1 198	829 219	1 100 430	855 172	835 103	917 959	925 878	925 878	918 616	787 905	693 503	664 456	400 584
Silver in Doré	Oz	2 766 535	0	44	231 261	200 295	235 861	224 645	256 444	244 024	244 024	244 024	244 024	244 024	244 024	153 841
Silver Revenue	k kr	302 880	0	5	25 318	21 928	25 822	24 594	28 075	26 716	26 716	26 716	26 716	26 716	26 716	16 843
Total revenue	**k kr**	**10 158 782**	**0**	**1 203**	**854 537**	**1 122 358**	**880 994**	**859 698**	**946 035**	**952 594**	**952 594**	945 332	814 621	720 219	691 172	417 427
Total operating costs	**k kr**	**4 724 623**	0	102 370	386 392	373 801	378 184	369 796	362 363	415 881	415 881	415 830	414 915	414 255	414 051	260 904
	kr/t ore	88.48														
	USD/oz Gold	447.25														
Operational results	**k kr**	**5 434 158**	**0**	**-101 167**	**468 145**	**748 558**	**502 810**	**489 902**	**583 672**	**536 712**	**536 712**	**529 502**	**399 706**	**305 964**	**277 121**	**156 523**
Capital costs (total)	**k kr**	**2 026 189**	**803 331**	**945 370**	**0**	**0**	**0**	**45 000**	**106 860**	**30 000**	**28 000**	**25 000**	**24 000**	**18 000**	**628**	**0**
Open Pit	**k kr**	**1 748 701**	**803 331**	**945 370**	**0**	**0**	**0**									
Underground	**k kr**	**277 488**	**0**	**0**	**0**	**0**	**0**	**45 000**	**106 860**	**30 000**	**28 000**	**25 000**	**24 000**	**18 000**	**628**	**0**
Equipment Spares	k kr	*24 169*	0	24 169	0	0	0	0	0	0	0	0	0	0	0	0
Sustainable capital	k kr	*18 000*		0	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500	1 500
Closure cost	k kr	*31 000*	0	0	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583	2 583
Reclamation bond	k kr	*25 000*	5 000	20 000	0	0	0	0	0	0	0	0	0	0	0	0
Overhead cost	k kr	*52 000*	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	4 000	
Residual value	k kr	*-300 000*	0	0	0	0	0	0	0	0	0	0	0	0	0	-300 000
Working Capital	k kr	0	40 000	40 000	20 000	0	0	0	0	0	0	0	0	0	0	-100 000
Cash flow	**k kr**	**3 553 801**	-852 331	-1 134 705	440 062	740 474	494 727	436 818	468 728	498 629	500 629	496 418	367 622	279 881	268 409	548 440
Accumulated	**k kr**		**-852 331**	**-1 987 037**	**-1 546 975**	**-806 501**	**-311 774**	**125 044**	**593 772**	**1 092 402**	**1 593 031**	**2 089 449**	**2 457 071**	**2 736 952**	**3 005 361**	**3 553 801**
Present Value of Expected Cash Flow		**4 179 722**														
Net Present Value (NPV)		**1 950 176**														
Internal Rate of Return (IRR)		**20.5%**														

Table 13.2 Financial evaluation

13.3 Sensitivity Analysis and Discussion

The gold assays for cores from Fäboliden have been analyzed at ALS Chemex Laboratories in Vancouver and the results from these analyses have been used when defining the Au grade of the mineral reserves and resources. The same cores have systematically and for comparative reasons been checked at OMAC Laboratories during the last two years. In total more than 300 tests have been performed indicating a strong linear deviation between the results where *OMAC shows approx. 6 % higher values compared to ALS Chemex*. In addition, the results from the *test mining that was carried out in March 2005, clearly indicates Au grades that are significantly higher* compared to exploration core ore drilling results. The Au grades that are applied for the financial evaluation in this Feasibility Study are therefore assessed to be *under estimates*.

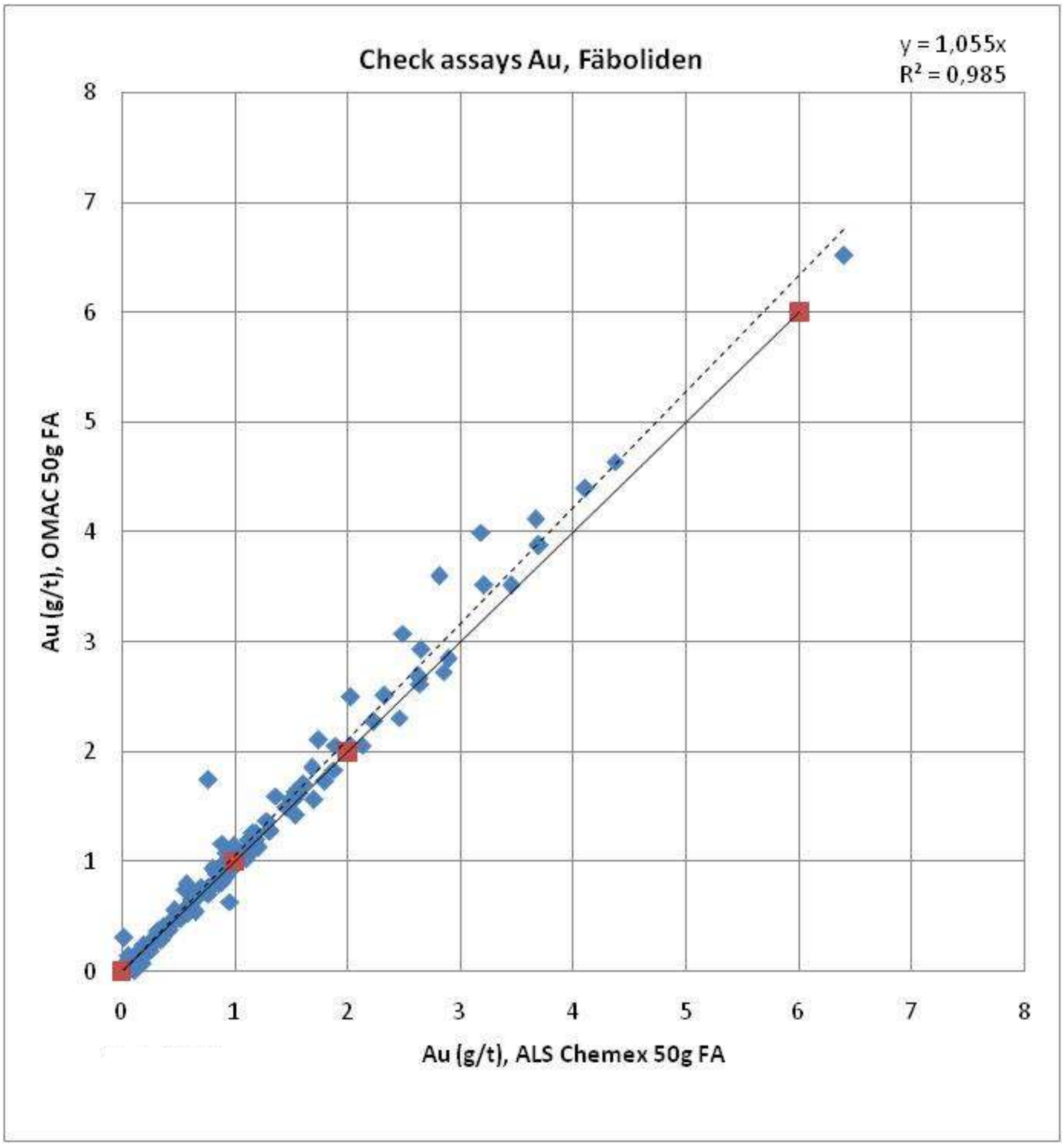


Figure 13.1. Comparison of test results (up to autumn 2007) from ALS Chemex and OMAC Laboratories (dotted line). There is a strong linear deviation between the results where OMAC shows approx. 6 % higher Au grades.

Results from sensivity analysis calculations are outlined below:

- Applying the *6% higher average Au grade* in accordance with the results from OMAC would mean a cash flow of SEK 4,249 million (NPV SEK 2,421 million) and a pay off time of 3.5 years.
- Applying the *30% higher average Au grade* would mean a cash flow of approx. SEK 7,052 million (NPV SEK 4,367 million) and a pay off time of 2.3 years.
- *Increasing the gold price with 10%* would mean cash flow of SEK 4,655 million (NPV SEK 2,705 million).
- *Decreasing the gold price with 10%* would mean cash flow of SEK 2,578 million (NPV SEK 1,281 million).
- *Increasing the total investment cost with 10%* would mean cash flow of SEK 3,405 million (NPV SEK 1,801 million).
- *Decreasing the total investment cost with 10%* would mean cash flow of SEK 3,702 million (NPV SEK 2,099 million).
- *Increasing the annual processing of ore* from 4.6 Mt to 5.0 Mt would mean a cash flow of SEK 3,791 million (NPV SEK 2,158 million) and 3.5 years pay-off time.
- Applying an *interest rate of 8%* would mean NPV cash flow of SEK 1,320 millions.

As shown in the figure below a gold price of approx. 700 USD/oz means a discounted cash flow of +/- 0 (assuming that the general conditions of today are unchanged). Assuming a gold price of 1,200 USD would imply a cash flow (NPV) of approx. SEK 4,000 million.

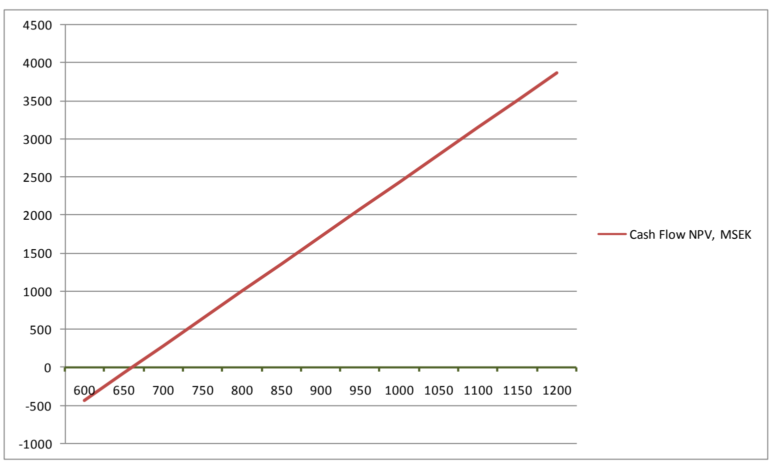


Figure 13.2 Cash Flow Net Present Value with different Au prices (USD/oz, horizontal line)

Changes in Au grades, compared to the average grade of 1.13 g/t, are analyzed in the figure below. It shows that only smaller increases in the Au grade imply tangible increases in cash flow. For example 15% higher average Au grade (i.e. 1.3 g/t instead of 1.13 g/t) would mean a NPV cash flow of approx. SEK 3,081 million while 50% higher average Au grade (i.e. 1.7 g/t instead of 1.13 g/t) would mean approx. SEK 5,873 million.

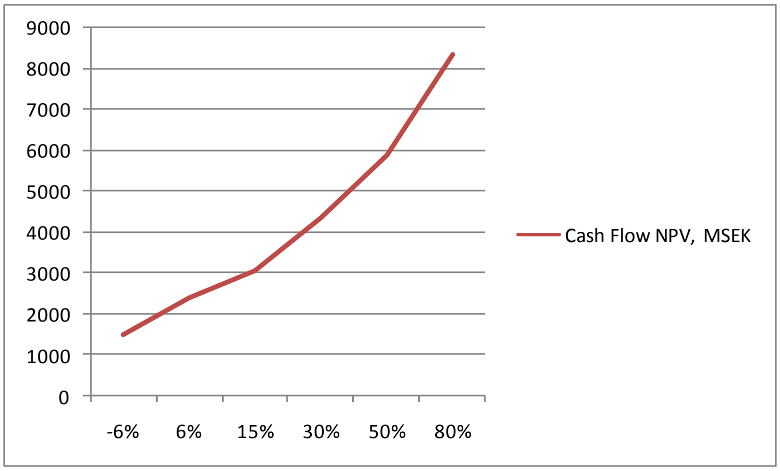


Figure 13.3 Cash Flow Net Present Value MSEK with different Au grades (changes in percent compared to the average grade of 1.13 g/t)

13.4 Risk Management

A number of risks/possibilities possibly affecting the financial results have to be considered and managed when carrying out the project:

- *Major future changes in the gold price*: The project means processing of large volumes of low grade ore which makes it sensitive for the development of the gold price. Hedging of parts of the future gold sales has to be considered.
- *Exchange rate changes/fluctuations*: The present exchange rate of approx. 6.4 SEK per USD means that SEK is relatively strong in relation to USD. This is favorable when buying equipment where the payment is in USD. As for the future when production starts the project would be favored by a stronger USD since the selling of gold will be in USD. On the other hand the price for tires, fuel and explosives and other commodities are often depending on the exchange rate and a stronger USD means higher prices. Dealing with financial risks, among other risks related to the exchange rate and gold prices, requires a financial strategy. Such a strategy will be developed by Lappland Goldminers with the aim to deal with financial uncertainties in a proactive and efficient way.
- *Variations in Au grades*: The average Au grade (dilution included) for the ore body is assessed to approx. 1.1 g/t. Recent checking of laboratory results and the test mining carried out back in 2005 clearly indicates that the actual grades will be higher. If this is the case the financial outcome of the project will be affected in a most positive way. Further testing will be carried out during the coming years and before production starts.
- *Investment cost increases*: Costs for mining and process equipment has increased in a tangible way during the last years and further increases can be expected. During 2008 detailed design will be carried out for the Fäboliden process plant. The design work will focus on developing and improving the preliminary design accounted for in this study, aiming at cost-efficient production, environmental concern and reductions of the investment cost. The procurement of the process plant will be carried out through competition between different possible suppliers.
- *Delays in the delivery of required equipment*: The immediate procurement of a mill to the process plant is of crucial importance for the time schedule of the project since the delivery time (including on-site installation) is estimated to approx. three years. Lappland Goldminers is therefore presently evaluating a number of mill tenders and is planning to sign a contract on this in the nearby future. As for other equipment delivery times are less than 18 months.

- *Major future increases in the energy price*: Energy prices have increased during the last years and further increases can be expected. Lappland Goldminers has the intention to establish a long term agreement for energy supply with one major supplier, avoiding future fluctuations in the energy price. In addition, Lappland Goldminers is involved in a Wind Mill Project located in Fäboliden close to the process plant, which in a number of years might turn out to be an alternative solution supplying the process plant with environmental friendly energy to a long term steady price. As for future mining activities the production will require large amounts of diesel fuel to the trucks and also to other mining equipment. However, the price for diesel fuel is usually not negotiable to any larger degree, which will mean that current market prices will have to be paid.
- *Appeals of the received Environmental Permit:* Lappland Goldminers received the required Environmental Permit from the Environmental Court in Umeå by the 21st of December 2007. The permit has been appealed by the Swedish Environmental Protection Agency and by some private persons and the Supreme Environmental Court will deal with the appeals in May 2008. However, since the Swedish Environmental Protection Agency is questioning technical and economical details, not the project itself, Lappland Goldminers is confident that the appeals can be sorted out during 2008 and that the Environmental Permit can gain legal force later on during this year. By this the time schedule for the implementation of the project won´t be affected on the condition that the project receive a permit to start preparatory construction works during summer 2008.
- *Changes in EU and Swedish legislation*: Legislation concerning environment, taxes, work safety etc is continuously changing/developing. The Environmental Permit Lappland Goldminers has received takes all foreseeable coming changes initiated by EU and Sweden into consideration within the legislative fields of health, environment and safety. This is a good thing since it reduces the uncertainty for future operations in the mine and in the process plant. Over and above this other legislative changes will take place not foreseeable today. These will mean uncertainties in different aspects which have to be dealt with when occurring.
- *Unexpected environmental impact*: Lappland Goldminers has within the frames for a comprehensive Environmental Impact Assessment worked with high ambitions to identify potential environmental risks and to implement risk eliminating measures and mitigation measures into the project. The environmental issues has to stay in focus, during the construction phase as well as when the mining and processing is getting operational, with the objective to avoid any future unexpected environmental impact.
- *Disruption to operations due to labor disputes or health and safety issues*: Different kind of personnel policies will be elaborated in good time before operations start. These will guide the management staff and help operations to run in a smooth, efficient and safe way. However, policies don’t mean all kind of problems can be avoided and there is no doubt that unexpected and unwanted events will occur. Such events have to be dealt with in a proactive way and with an ambition to avoid any future operational disruptions.

14. Time Schedule

A preliminary time schedule is enclosed. The project is estimated to be carried out during 3 years, production starting in the beginning of 2011. Some basic assumptions are:

- Procurement of a mill in March 2008 with a delivery time of approx. three years.
- Permit to start preparatory construction works during summer 2008.

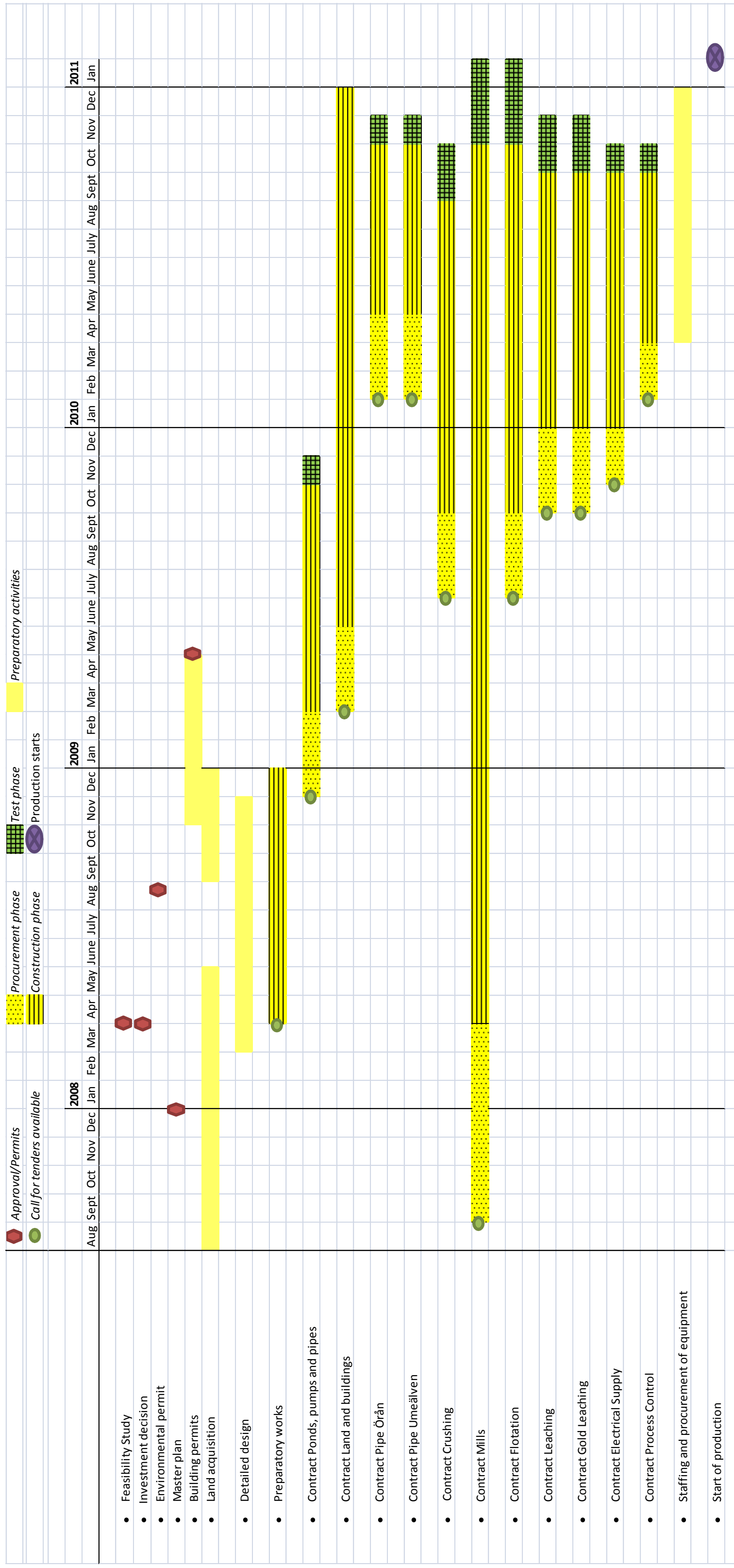


Figure 14.1 Time Schedule

15. Resources – Staffing and Operations

15.1 Introduction

This section addresses required human resources. Lappland Goldminers AB intends to operate Fäbolidens processing and mining operations with minimum number of staff possible and will therefore need to attract employees with a high level of skills and experience. The project is situated close to other major mines allowing skilled staff to be recruited locally.

15.2 Local Employment Conditions

Salaries and wages in Sweden (excluding the municipal tax payments by employers) are broadly equivalent to rates prevailing in other Nordic countries. The unions are industry wide and are responsible for the negotiation of labour contracts covering wages, salaries and employment conditions. Other matters such as holidays, parental leave and sick leave are mostly covered by legislation. Due to the fact that unions and employers traditionally seek consensus when resolving issues, the rate of industrial disputes is very low. It is expected that all staff will accommodate themselves in either privately owned or rental accommodation within a 50 kilometers radius from the site and will be responsible for their own transport to and from the work.

The process plant will operate on a shift basis. Each shift will be supervised by an experienced process plant manager who also will act as mill operator. Maintenance will operate on a single shift basis with a call out roster to cover breakdowns outside normal working hours. Mining operations are expected to work around the clock.

15.3 Project Structure

The projects corporate activities will be based in Lycksele under the supervision of a CEO reporting to the Board of Directors. The CEO will be responsible for the following activities:

- New business development
- Liaison with shareholders and investors
- Treasury activities and

Fäbolidens operations will be overseen by a Site Manager based at the mine who will have the following heads of departments reporting to him:

- Liaison with Swedish local and national government agencies.
- Administration and Health and the Environment
- Mining and Geology
- Process Plant

The Site Manager will be responsible for the following functions:

- Development of administrative and operations procedures
- Overall supervision of the Fäboliden operations
- Liaison with local authorities, and
- Ensuring agreed throughput and cost budgets are met.

Manning levels
The different manning levels are shown below.

Administration	5
Mining and Geology	147
Process and Plant Maintenance	47
Total	200
Contractors –nominal	>25

Table 15.1 Manning levels

The numbers stated in the table assumes that contracts will be let for the following activities:

- Maintenance and service
- Resource and grade control drilling
- Maintenance staff cover during weekends and public holidays
- Maintenance work carried out at major shut downs
- Bullion transport refining
- Transport, logistics and expediting
- Environmental sample analysis, and
- Information Technology support.

Consultants will also supplement Fäboliden staff on an as needed basis to cover exploration, mine planning and training roles.

Administration Department
The Administration Department will be responsible for the cost control, payroll, purchasing, health, safety and the environment implementation and monitoring. It will not be responsible for any corporate activities.

Though the Administration Department will be headed by the Site Manager, almost all of the administrative duties will be performed by the Mine Accountant. The numbers of staff employed in the Administration Department is summarized below.

Site Manager	1
Account	2
Health, Safety and Environment Officer	1
Secretary	1
Total	5

Table 15.2 Administrative Staff

The Site Manager will make use of contractors and consultants to cover administrative functions such as procedure development and purchasing. The Health Safety and Environment Officer will be responsible for the environmental monitoring of all discharge points from the mine as wells as ensuring that the plant and mine operates in accordance with Swedish environmental and safety regulations. The Administration Department will carry out the following functions:

- Accounting
- Payroll
- Purchasing
- Community relations
- Environmental monitoring
- Environmental reporting
- Safety monitoring, and
- Safety training and reporting.

Mining and Geology Departments

The Mining Department will be headed by a Mine Superintendent whose prime responsibility will be to oversee the mining. It is proposed that initially the Geology Department will also report directly to the Mine Superintendent, though as the mine develops and potential underground or/and satellite operations commence, the Geology Department may report directly to the Site Manager. Since the Chief Geologist will act as both the Regional Exploration Manager and resident Mine Geologist, the majority of the geological supervision of the mining will be undertaken by the Mine Geologist. Pit Technicians will be used as samplers, dig supervisors and survey assistants.

Mine Superintendent	1
Chief Geologist	1
Mine Geologist	1
Pit Technicians and Samplers	2
Mine Planning/Surveyor	1
Drilling and Blasting	13
Loading and Hauling	57
Maintenance	31
Others	40
Total	147

Table 15.3 Mining and Geology Department Staff

The Mining Department will carry out the following functions:

- Mine planning, extraction and supervision
- Geological estimation and reconciliation
- Grade control
- Mine production

Process Plant Department

The Process Plant Department will be headed by a Mill Superintendent who will have reporting to him all staff responsible in the process plant, maintenance and laboratory activities carried out on the site.

The manning level of the Process Plant Department is summarized below.

Management	4
Shift Operations	30
Day Operations	7
Maintenance	6
Total	47

Table 15.4 Process Plant Department Staff

The Process Plant Department will carry out following functions:

- Process plant operations
- Tailings dam operations
- Operation of the mine site laboratory
- Maintenance of the Process Plant and mine site infrastructure.
- Training

15.4 Organization

The future organization is shown below.

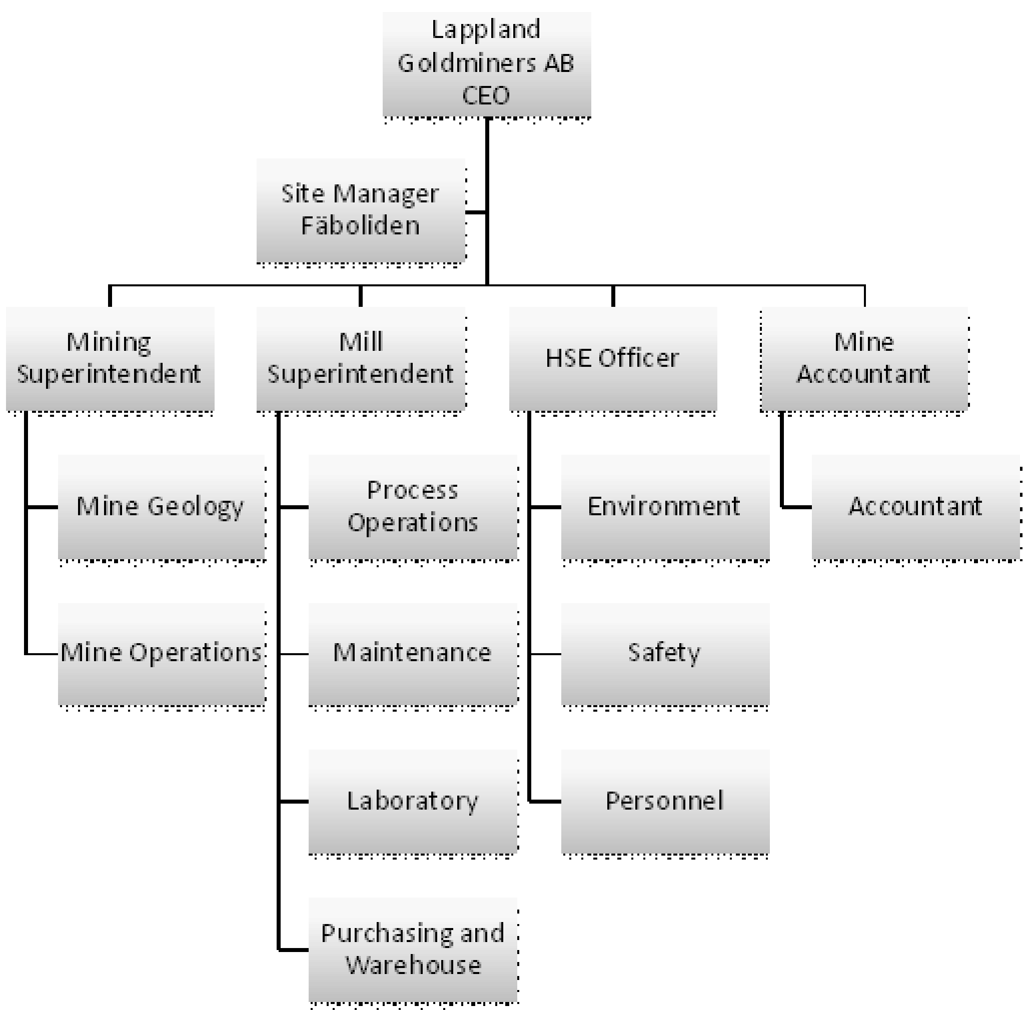


Figure 15.1 Organization of Fäboliden Gold project

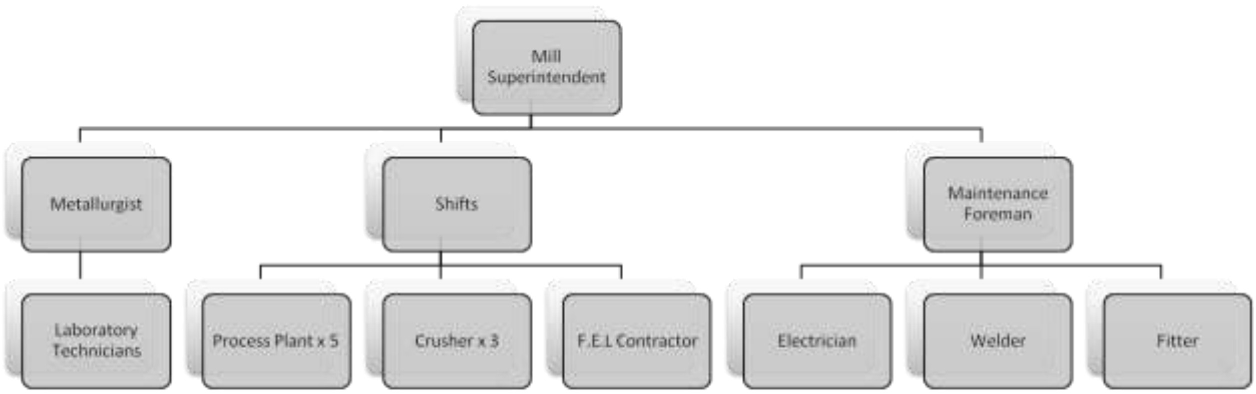


Figure 15.2 Process Plant Organization

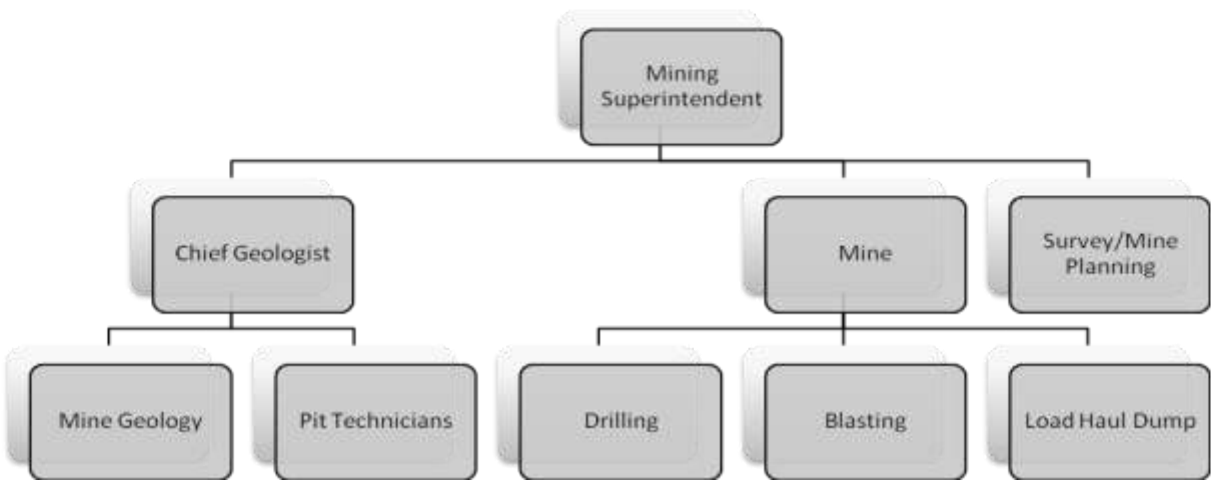
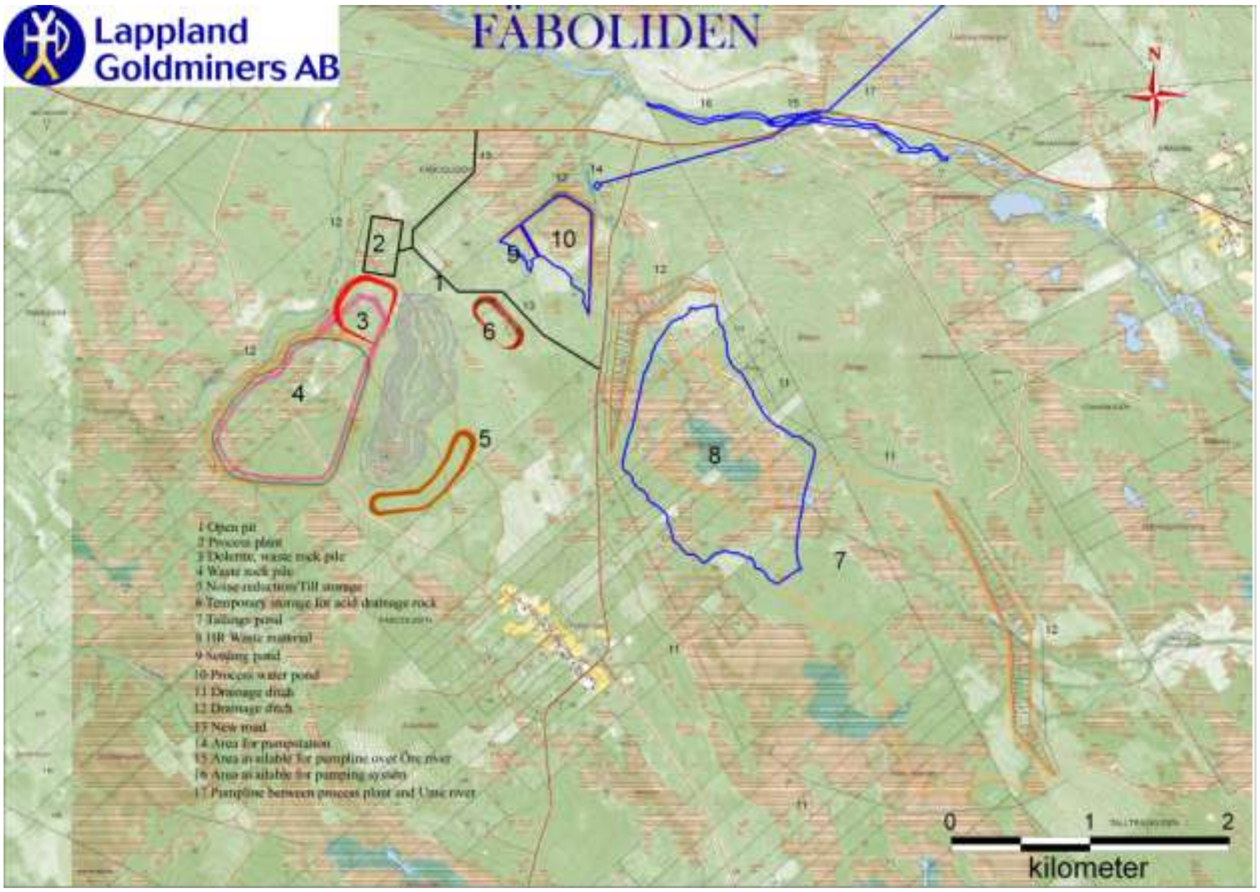


Figure 15.3 Mining Organization

15. Site Map

An overview map outlining the land use for the mine and the process plant can be found below.

In Appendix 1(5) the same overview map is enclosed in the scale of 1:10 000.



Tables

Figures

Date and Signature Page

The undersigned have had the general responsibility for the respective part of the Feasibility Study, titled *Fäboliden Gold Project*, dated April 2008.

- Jan Nilsson, Managing Director, Outotec (Sweden) AB: Outotec has coordinated the work and carried out the engineering design and capital cost estimates for the process facility.
- Thomas Lindholm, Mining Engineer, GeoVista AB: Responsible for defining mineral resources and reserves.
- Karl-Åke Johansson, Managing Director, Lappland Goldminers AB: The financial evaluation has been compiled by Lappland Goldminers based on the results brought forward by the different companies involved.

The content of the report is intended to conform to the regulations of SveMin (The Swedish Mining Association).

Signed and sealed

3rd of April 2008

Jan Nilsson	Thomas Lindholm	Karl-Åke Johansson
Managing Director	Mining Engineer	Managing Director
Outotec (Sweden) AB	GeoVista AB	Lappland Goldminers AB

***Lappland Goldminers AB** is a mining and exploration company. The company is listed on the market place "First North" and on the Norwegian OTC list.*

Lappland Goldminers has secured a number of gold deposits along the so-called Guldlinjen ("The Gold Line") in Västerbotten, Sweden. The company's strategy is to develop a profitable, producing mining company with centrally located processing plants, in Fäboliden in Sweden and in the Haveri area in Finland, which are supplied with ore from one or several mines through the company's own exploration, or alternatively through acquisitions.

Lappland Goldminers is a member of SveMin (the Association for Mining, Mineral and Metal Producers in Sweden - formerly called "Svenska Gruvföreningen").

For further information about the company, please visit; www.lapplandgoldminers.se

Lappland Goldminers AB