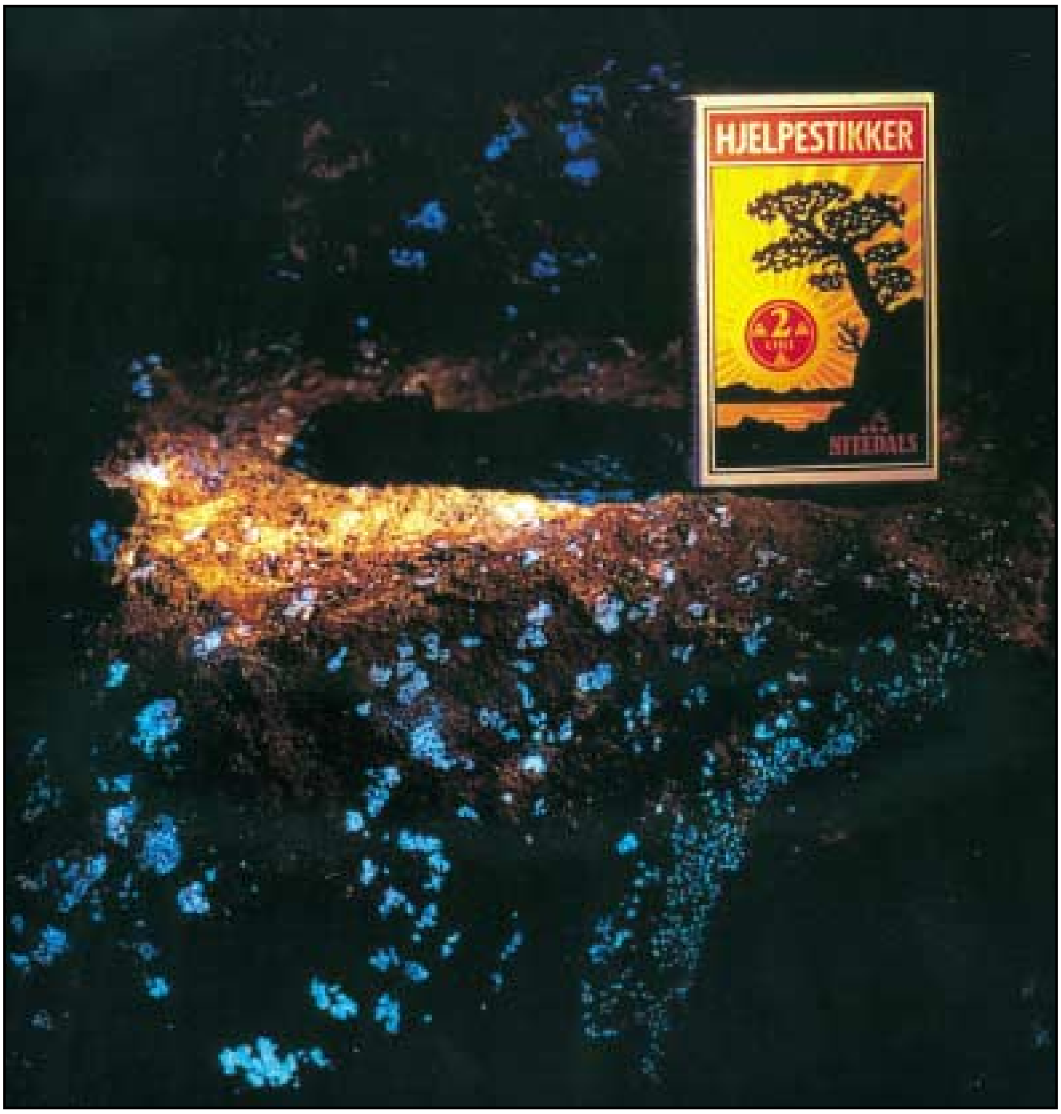
PROSPECTUS

by

Peter Skaarup and Kåre M. Lande

Invitation to participate in the development of the Målviken tungsten skarn occurrence, Brønnøy Kommune, Nordland Fylke, Norway



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Brønnøy kommune

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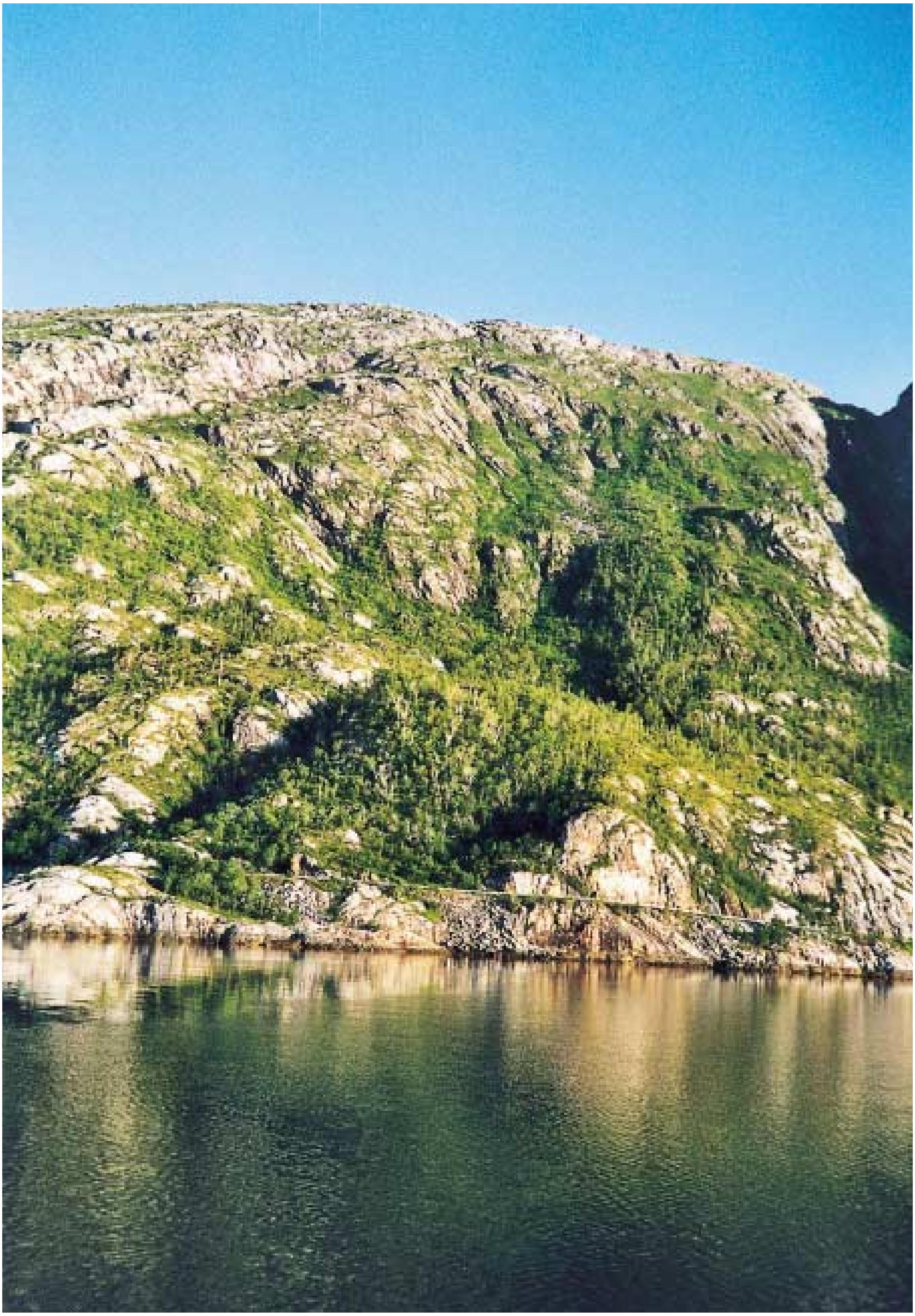
NORDLAND FYLKESKOMMUNE

February 2001

Front page:
Sample of the very rich scheelite mineralisation in skarn from the “Peter zone” of the Målviken occurrence.
Photo: Roar Berg Hansen, Brønnøysund

THE MÅLVIKEN TUNGSTEN SKARN OCCURRENCE
Brønnøy Kommune, Nordland, Norway



1. SUMMARY

The private prospectors Peter Skaarup and Kåre M. Lande are looking for partners/investors in the Målviken tungsten skarn project, Brønnøy Kommue, Nordland, Norway. The occurrence is held by claims owned by Kåre M. Lande.

The tungsten-bearing mineral scheelite ($CaWO_4$) is present in skarn bands within a 30-60 m wide and 900 m long zone which may be followed from sea-level and up to 500 m. At present 89 exposures of rich ore have been outlined into 13 zones representing the ore. The 13 zones normally occur in two and up to six sub-parallel levels within two-thirds part of the main zone. The remaining one-third part is unexposed as it is covered by a thick layer of soil or boulders. The last part represent a further potential for mineralisation, which it is not included in the inferred ore-reserve.

The mineralised skarn bands have a width from 1 to 4.5 m and make up a total length of 949 m when the 13 zones are taken together. Assuming the frequency of the mineralisation and the grade of the ore is constant downwards to 100 m below sea-level, the occurrence has an estimated resource of 2,509,000 t ore containing 0.7-0.9 % WO_3. Compared to the world's proven tungsten reserves grading more than 0.3% WO_3, the Målviken occurrence is placed as no. 1 in Europe and no. 21 in the world, if the grade is 0.9% WO_3

The occurrence has a unique location. It is crossed by a main road and power line. Water is to be taken close by. A landing quay for deep-seated vessels may be constructed at Tosenfjorden at the main entrance to a future mine. The nearest house is at a distance of 2.5 km. But otherwise the area is urbanised and a skilled labour force can be recruited locally. The airport is in Brønnøysund. The local main town with 3,600 citizens is 70 km away.

The tungsten market has been depressed since China's re-entry at the beginning of the eighties. The situation worsened with the break down of many central planned economic countries. At present tungsten is sold from stocks and China subsidised the production.

The forecast for tungsten production, however, is promising within a period of 5 years for the following six reasons: 1) The world's stocks of tungsten are depleting. When the stocks are emptied the present over-supply situation will change into an under-supply situation as the production capacity is lower than the consumption. 2) The worlds proven tungsten resources will be exhausted in 2007 by the actual consumption. 3) Chinese tungsten production is unprofitable at present prices. China is now gaining control over the production in order to make it profitable. This has raised the price since July 2000. 4) Reconstruction of economy in the former central planned economy countries especially in Russia should raise the consumption by 15%. 5) USA has increased budgets for armour production after falls since 1991. Present and future NATO partners have the same needs. 6) WHO and NATO investigations of health consequences and environmental impact of the use of depleted uranium during the Gulf and the Balkan War may lead to a return to the use of tungsten in armour penetrating ammunition and war-heads in missiles. This together with the new use of tungsten in golf-clubs and "green bullets" for hunting may increase the demand considerably.

Possibly the first sign of a new era for tungsten is the increase in the low – high tungsten concentrate price from US$33 – $44 in July 2000 to $55 - $73 in January 2001 per mtu contained WO_3.

A development plane for the Målviken occurrence has been set up. The first part is a continuation of the ongoing exploration phase. It comprises systematically mapping, sampling and analysing of the new-found exposures and searching for others; drilling to prove the existence of mineralisation at greater depth; tests of ore-recovery by relevant methods on a laboratory scale; and the final account of planning and environmental questions in order to receive a mining permission. The investment in these matters is estimated to be US$ 615,000 or 5.5 million Norwegian kroner. The Norwegian development funds may give financial support for 30%, and in certain cases 50% of the exploration costs.

The second part comprises a drilling program, construction of a mine and scheelite production. The investment during this phase has been calculated by Karl-Åke Johansson, Minexp AB. Karl-Åke Johansson on basis of the present geological knowledge has evaluated mining methods and mine construction and calculated the capital expenditures and operating cost. The cash flow is compared to a revenue from marketing scheelite concentrate at a market price of US$ 50 per mtu.

The total investment of the exploration and production phase is calculated to be US$ 15,151,000 or 134,850,000 NOK in a 300,000 t per year production of ore grading 0.8% WO_3 per t in situ. The break-even point after 10 year of production at a rate of 0% return lies at a market price of US$ 53 per mtu WO_3. The actual high price US$ 73 relevant for scheelite concentrate has an 18% rate of return. At a market price of US$ 80.5 the rate of return of the investment is 25% per year.

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Content:

2. BACKGROUND AND PREVIOUS WORK

At the end of the sixties the Norwegian Geological Survey went through their collections with an ultra violet lamp and discovered scheelite in samples from the gold-arsenopyrite occurrence in Kolsvik Ore-field 15 km S of Målviken. A/S Sydvaranger and Norsk Hydro started a systematic search for scheelite over wider areas of Nordland. Scheelite was found at different localities but only in small amount.

One of the present prospectors, Peter Skaarup, being a stipendiat in 1972 at the Geological Institute of the University of Copenhagen, was offered the study of the gold-arsenopyrite occurrences and the general geology in the Bindal area as a scientific project by A/S Sydvaranger. Arriving in the area he was asked in which rock types the scheelite hides since the mineral was only known from stream sediments. The search for scheelite by uv-light on solid rocks led to the first findings of scheelite in skarn on Finli-field and at five more localities in 1972. The rich scheelite mineralisation at Målviken was found the following year and traced up the hill side to about 350 m height. The findings were made at the end of the 1973 field season. Peter Skaarup happened to spend only three days on the occurrence before returning to Denmark. The results of the two years of work were published in the paper: *Strata-Bound Scheelite Mineralisation in Skarns and Gneisses from the Bindal Area, Northern Norway. Mineralium Deposita 9, 1974.*

Kåre M. Lande living in the small village Lande at Tosenfjorden was also involved in the investigations of A/S Sydvaranger. A personal friendship between the two authors was started in these years and led to the later prospecting activities.

A/S Sydvaranger only carried out a few further investigations of the Målviken occurrence. Ten years later A/S Sulfidmalm, an exploration firm under Falconbridge Ltd., Canada, went into the zone and found more mineralisation than previous known. After sampling and analysing, the occurrence was considered so promising that a drilling program was carried out and the ore tested by flotation at the AB Statsgruvors laboratory using the same method as for the Swedish scheelite ore at Yxsjöberg. A/S Sulfidmalm made the following reports on their work: *B.A. Sturt: Preliminary report of the regional geology of the Tosen-Moldvika Area, Northern Norway. A/S Sulfidmalm report 1982* dealing with the general geology; *F. Nixon and K. Kjærsrud: Målviken Scheelite Occurrence, Bindal region, Norway. A/S Sulfidmalm report 1983* dealing with the general geology and the analytical results; *M. Lindvall: A/S Sulfidmalm. Anrikningsundersökning av ett scheelitmalmprov. Delrapport 2 och slutrapport: Redovisning av satsforsök. Yxsjöberg 1983* dealing with the flotation test. A report on the diamond drillings, however, is lacking. No one seems to have heard of it. Kåre M. Lande has personally been told that the ore zones were located in three out of four drillings. The lowest placed on the road side near sealevel was the one without mineralisation. Moreover a visiting geologist has been presented with rich mineralisation in one drill-core by the A/S Sulfidmalm people.

The British geologist David Wyn James studied the occurrence. He received his Ph.D. for the thesis: *The Geology and Geochemistry of the Målvika Tungsten Skarn Deposit, Central Norway* at Sheffield University 1991. His thesis encompassed mapping, petrographic, mineralogical, geochemical investigations of parent rocks, skarn and mineralisation together with genetic considerations.

The present authors continued the search for scheelite in the region and in 1993 asked the Nordland Program under NGU for financial support to install remote camps in the area. Prior to such activity Ingvar Lindahl, head master of the Nordland Program and Peter Ihlen recommended us to carry out further work on the Målviken occurrence which we thought fully investigated. We received financial support from Brønnøy Kommune for such work in 1995 and 1997. In 1995 the known findings together with some new ones were mapped, sampled and analysed. The breakthrough, however, first came in 1997 simply by hacking and removing earth in cross sections. Today 89 exposures of good and excellent mineralisation in 13 bands of skarn are known.

The description of these exposures, geological map, analyses, ore evaluation, together with the above mentioned previous works and the market situation for tungsten are treated in the report: *Peter Skaarup and Kåre M. Lande: Målvikzonen – 1995 og 1997 kortlægning, prøvetagning, økonomisk vurdering af wolframmineraliseringen og dens genese, 1998.* The report (222 p.) will be translated from Danish into English as documentation for potential investors. The present prospectus refers to the report.

In order to evaluate the economic potential of the Målviken occurrence, a model for mining and mine construction has been set up by mining engineer Karl-Åke Johansson, Minexpert AB. The model is given as an appendix *p. 32-44* in this prospectus.

The prospectus is supported by Brønnøy Kommune, Nordland Fylke, the Norwegian Industrial and Regional Development Fund (SND) and the Norwegian Trade Council, Toronto. Contact persons and addresses are to be found on *p. 6*. Geologist John Bailey has kindly improved the English language.

3. THE MÅLVIKEN OCCURRENCE

3.1. Location

The Målviken tungsten skarn occurrence lies at Tosenfjorden on the main road connecting Brønnøysund with the E6 *(fig. 1)*. The road actually cuts through the lowest part of the occurrence. We are in Brønnøysund Kommune (the local municipality) part of Nordland Fylke (the local county) almost in the geographic centre of Norway, midway between Nordkapp the northernmost point and Lindesness the southernmost point. The distance from Trondheim is 370 km and from Oslo 860 km.

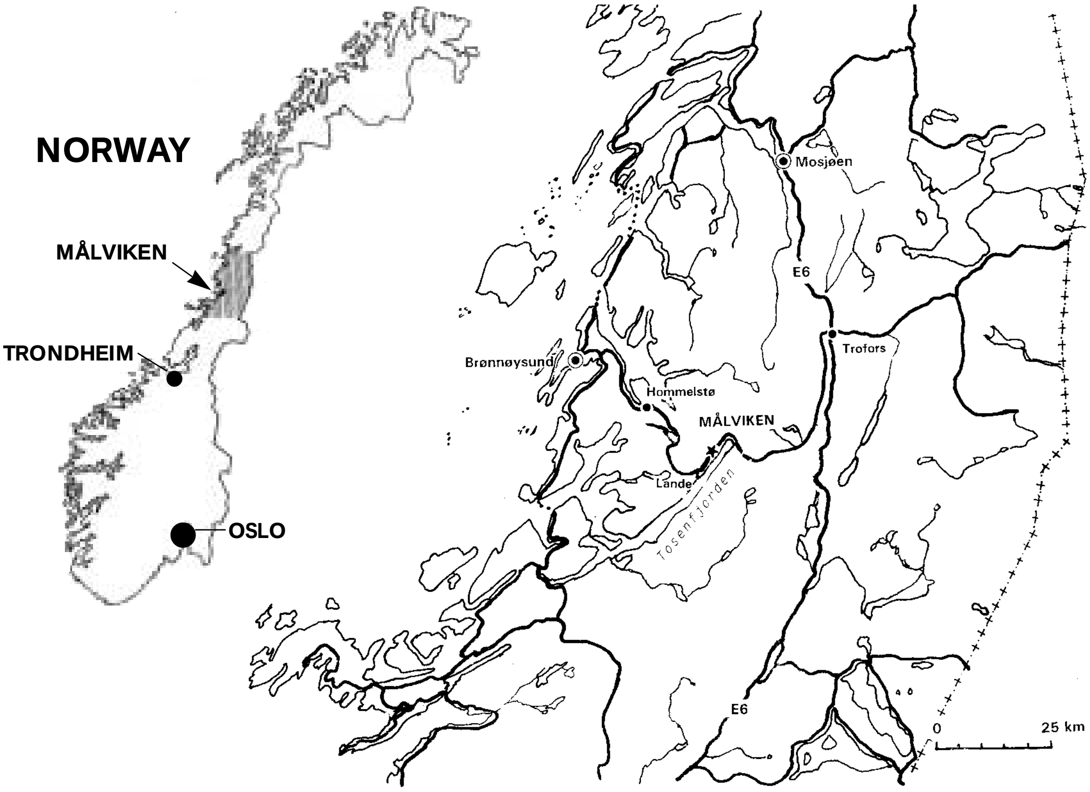

Fig. 1. Location of the Målviken occurrence

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3. 2. Dimensions

The scheelite ($CaWO_4$) mineralisation considered to be of economic value is found in skarn bands occurring over a horizontal distance of 900 m within a 30 to 60 m wide zone ranging from the fjord just south of the little bay Målviken up the steep mountainside NNW to Fjeldnubben at 500 m height. The mineralisation normally occurs in two and up to six subparallel bands of skarn within the zone.

3. 3. Terrain and exposure

The mountainside has a uniform steep slope of 29° on average. The only interruptions are narrow shelves where NE-SW trending fracture zones cut through. Up to 360 m above sea-level the terrain is irregularly rugged and covered by a relatively thin layer of soil except for boulder fans below the fracture zones. Above 360 m the glaciers have given more rounded forms and left behind a thicker cover of out-wash moraine.

Boulder fans and moraine prevent excavating to solid rocks over a distance of 320 m of the 900 m long zone. This gives a potential for further scheelite mineralisation in one-third of the zone. This potential has not been included in the ore evaluation.

The exposure over the remaining two-thirds of the zone is limited especially over the amphibole-biotite gneiss and the skarn forming the main part. Here the rocks are covered by 10 to 20 cm of fertile soil on which forest grows in the lower parts and grasses and herbs in the higher part. Up the zone there are only a few natural exposures of skarn. Most of the exposures marked on fig. 3 have been made by hand.

The biotite gneiss bordering the Målvik zone is far more wellexposed as the gneiss is more resistant to weathering and gives less fertile soil. The vegetation here is low bushes.

3.4. Geological setting

The gneiss and the marble of the area belong to the Cambro-Ordovician Helgeland Nappe of meta-sediments. In the Målvik zone these are mainly amphibole-biotite gneiss of presumed volcanic origin, minor semipelitic biotite gneiss and not more than three or four tiny layers of marble originally being limestone. The meta-sediments were metamorphosed under staurolite-almandine subfacies conditions and underwent widespread small-scale folding during the first main phase of tectonisation.

During the second main phase of tectonisation the sequence down-folds in the border-zone to the Bindal Massif around NNW-SSE to N-S trending axes to have an uniform steep eastern dip. The sequence was intruded in the initial phase of down-folding by semi-concordant dykes of granite and by more rare fine-grained grey dykes probably of quartz diorite. These 1-2 m wide dykes are numerous in the zone- near area. One lager 10 m wide granite dyke can be traced upwards within the zone. 3 km W of the Målviken occurrence a major sheet-formed diorite of regional N-S extent is supposed to have been intruded during the same phase *(fig. 2).*

Migmatisation of biotite-gneiss as well as veining of amphibole-biotite gneiss and granite dykes took place during the second phase of tectonisation – along with skarn formation at the contact between marble and wall gneiss or granite dykes and skarn formation in favourable tectonic zones.

The first formed small-scale folds were refolded during the second phase to have steep E and SE plunging axes. The deformation also gives newformed small-scale folds in dykes and gneisses with nearly horizontal NNW-SSE or N-S trending axes. The younger small-scale folding is much less widespread than the older. Deformation in the form of strong shearing, thinning and paralleling of dykes and gneisses in certain zones are, however, marked. Particularly marked is the zone forming the river bed of Rafossen east of the Målviken zone. Shearing zones are also found within the Målviken zone and west of it.

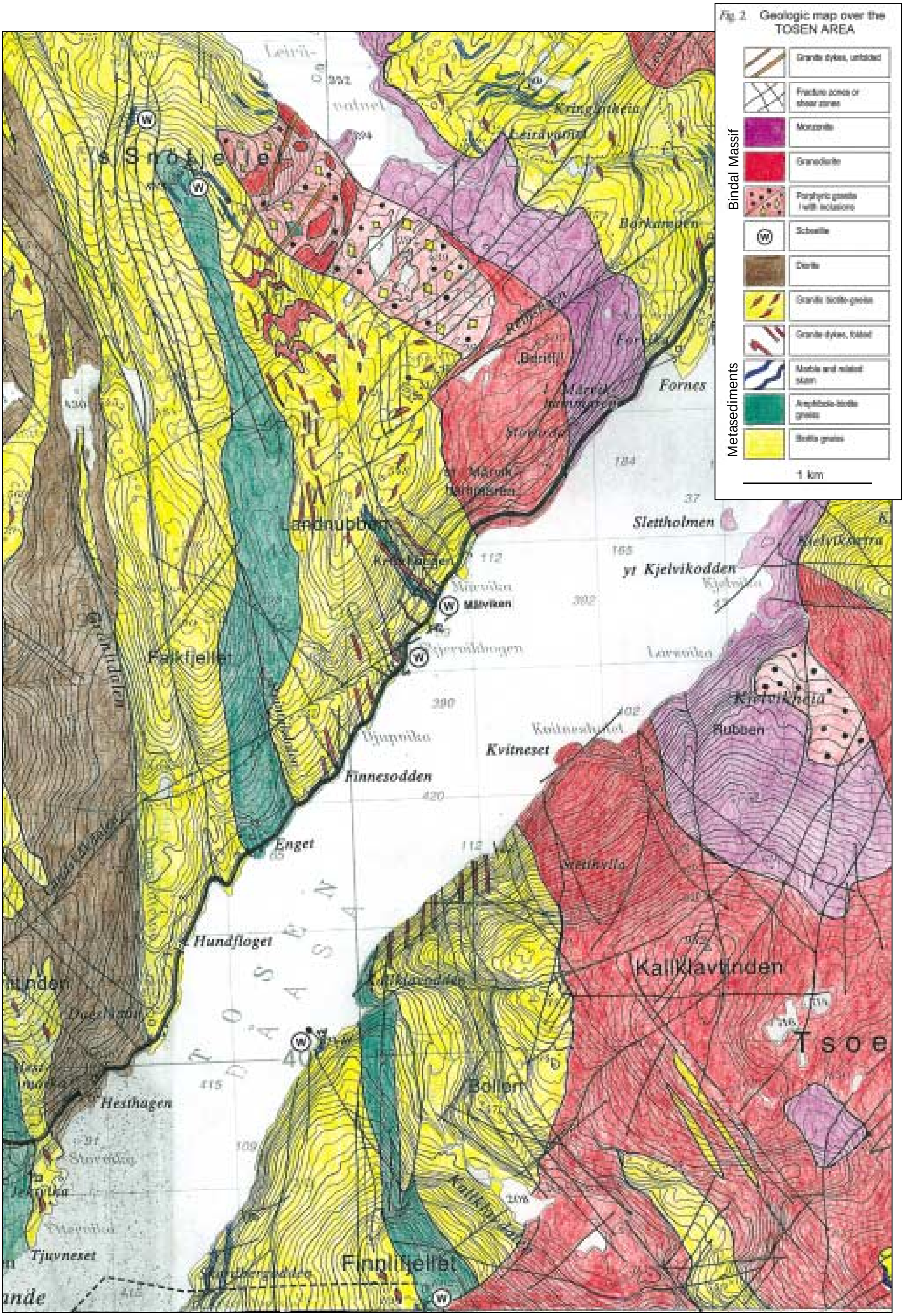


Fig. 2. Geologic map over the Tosen Area

The above mentioned strong deformation of gneisses and dykes in the border zone to the Bindal Massif, stands in sharp contrast to the apparently passive emplacement of the augen granite, the granodiorite and the monzonite forming the Bindal Massif *(fig. 2)*.

The granodiorite lying in contact to the meta-sedimentary sequence approximately 500 m NE of the Målviken zone is a massive white medium-grained rock with only a faintly developed steep foliation of random direction. The contact itself follows the main foliation of the gneiss and the same NE dip. A close-up inspection shows, however, that the granodiorite cuts the foliation of the gneisses. The granodiorite contact towards the meta-sediments continues across Tosenfjorden. On Finlifjeld which lies just in the southermost part of the map *(fig.2)*, the granodiorite has numerous sharp-edged inclusions of amphibole-biotite gneiss which are partly skarnitised and scheelite bearing. The sharp-edged inclusions can be traced over a 600 m distance further south. These localities show the scheelite mineralisation to be older than the emplacement of the granodiorite at its present position. The scheelite mineralisation in the Målviken occurrence is of the same type and is believed to have the same relative age.

Younger scheelite mineralisation or remobilisation of the older is present in one of the numerous so-called gold-quartz-arsenopyrite dykes in the area. One of these dykes is that in Kolsvik Ore-field which gave the first found scheelite (see *p. 7*). The gold-quartz-arsenopyrite dykes are not really dykes but silicified fracture zones along thrust-planes cutting through the meta-sediments and the granodiorite of the Bindal Massif.

Formations younger than the above mentioned second main phase of tectonisation includes - besides the gold-quartz-arsenopyrite dykes - young granite dykes and fracture zones. The young granite dykes are generally unfolded 1-2 m wide dykes cutting the gneisses and Bindal intrusives in an area around Leirå Vatn. The granite dykes have the common NE-SW fracture zone direction determining the direction of the Tosenfjorden, but they seem not to be connected to the same phase of brittle deformation. The fracture zones are from a few m and up to 10 m wide. Other generations of fracture zones with different direction are also present in the area.

3.5. Skarn

Skarn forms all over the area along the borders between marble and country rock. At some distance, 1 km or more from the Bindal Massif, the border zones are only a few cm wide. Closer by, the zones increase to 10-50 cm. At the contact and inside inclusions within the massif, skarn forms more meter-wide zones.

The increasing degree of skarn formation towards the massif is parallelled by the increasing degree of mobilisation of the biotite gneiss. At some distance from the massif the biotite gneiss has a pegmatitic neosome phase forming veins cutting the foliation of the gneisses semi-concordantly. Closer by, the biotite-gneiss grades into schlieren and finally into nebulitic gneiss.

Fig. 3 Scheelite-bearing skarn. Note the two sets of fracture-filled veins with recrystallisation along the borders. Tosenveien.

Besides the nearness to the Bindal Massif, skarn formation further depends on the rock replaced. The amphibole-rich variety of the amphibole-biotite gneiss originally being Ca-rich undergoes much more widespread transformation than the more biotite-rich types and the granite dykes.
This dependence is quite obvious in the Målviken zone where the transformation of the amphibole-rich gneisses may give way to 10 to 20 m wide skarn bands. The borders of the skarns are quite irregularly cutting the foliation of the gneiss and all transitions from local to complete replacement may be found.

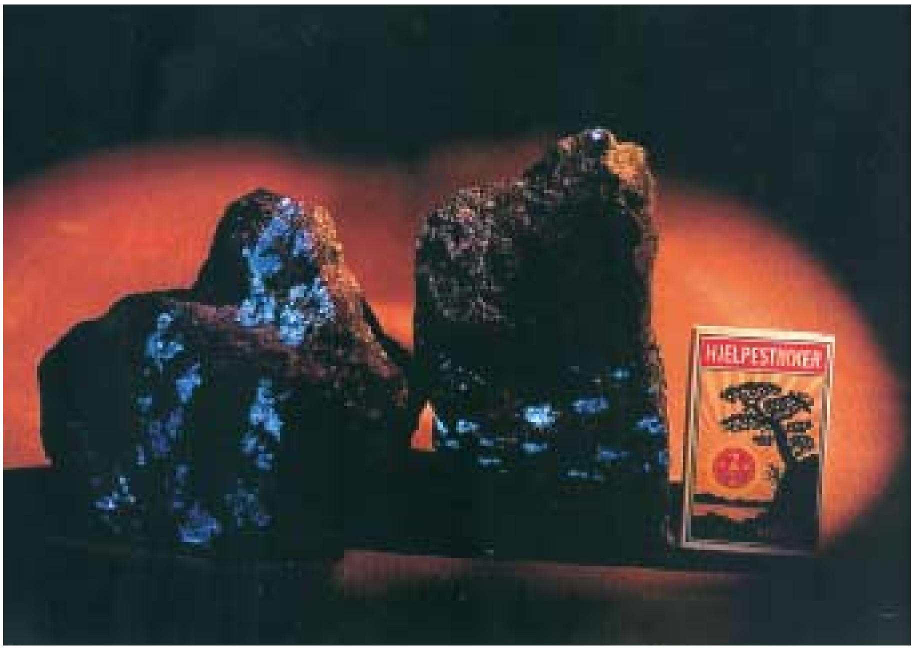


Fig. 4. Examples of scheelite schlierens. Photo: Roar Berg Hansen, Brønnøysund

When marble is present, skarns always form a wall zone. It seems obvious that the skarn forms from transportation of Ca-rich solutions from the marble into the wall rock. The skarn forms a so-called *reaction*-skarn, as mentioned first by Eskola. In the Målviken occurrence only tiny layers and scattered lenses of marble are present. The connection of skarn to certain levels in the sequence suggests, however, a relationship to three possibly four minor layers of marble in the original sequence. Certain parts of the wider skarn have, however, no obvious signs or relics of marble. This skarn is expected to form in zones of strong deformation where the deformation has given ingress to more long-travelled Ca-bearing solutions.

The mineral content of the skarn depends on the rocks replaced. Where the amphibole-rich variety of the amphibole-biotite gneiss is replaced, a nearly mono-mineralic skarn of diopside is formed. Replacement of the less amphibole-rich amphibole-biotite gneiss, the biotite gneiss and the granite dykes gives a skarn formed from diopside and plagioclase, with an increasing amount of plagioclase in the mentioned order of rock types. Skarn rich in grossular-andradite-rich garnet may occur. Such garnet skarn often appears close to relics of marble or centrally in the diopside skarns as elongated masses of limited extension and width up to 1 m.

3.6. The scheelite mineralisation

Scheelite appears in the skarns mainly as grains over 1 mm. Common grain sizes are from 0.5 cm to 1 cm. The scheelite grains occur disseminated forming a mineralisation which may be characterised as either scattered, rich or very rich mineralisation. More common, however, are scheelite forming schlierens or rows of single grains *(fig. 4)*. The schlierens may been characterised as either rich or very rich mineralisation. Moreover scheelite occur in veins, breccias and local pockets forming extremely rich mineralisation with more than 50 vol% scheelite. Finally scheelite occur as scattered grains on rare crosscutting fractures in the skarn.

The mineralisation is variable in strike. Rich schlierens at the top of a skarn band may be succeeded by rich schlierens in the middle or the bottom of the same band close by. The field work in 1997 showed this variability of the mineralisation. We learned to uncover the skarn bands in their whole width, since skarn which at first sight seemed without mineralisation was often found to be mineralised when it was uncovered in its full width. We also learned that the rich, the very rich and the extremely rich types of mineralisation are present at all levels of the Målviken zone from sea level until the end of the zone 500 m above.

Wyn James (see *p. 8*) found the scheelite mineralisation to be connected to a phase of pervasive hydrous alteration of the first formed reaction-skarn. This alteration is generally not seen by eye in the field. The mineralised skarn might be slightly coarser, in part pegmatoid, but usually one has to shine on the rock with the uv-lamp to be sure of the mineralisation.

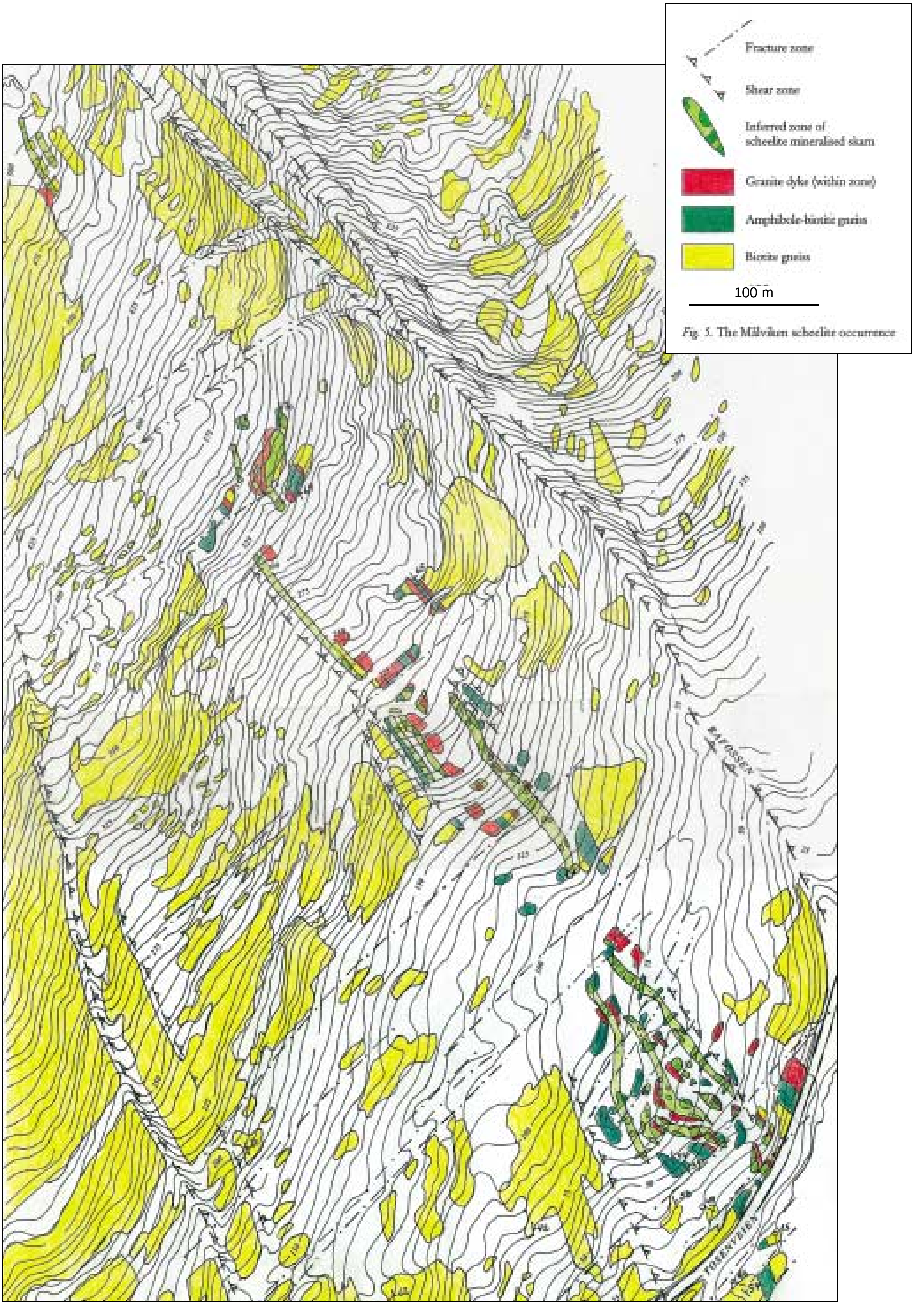


Fig. 5. The Målviken scheelite occurrence

The pervasive hydrous alteration shows up in thin-sections by the connection of scheelite to the hydrous minerals amphibole and clinozoisite, together with pyrrhotite, calcite and quartz. Almandine-spessartine-rich garnet may be present too.

Chemical analyses show that the mineralised skarn besides tungsten is enriched in molybdenium, gold, bismuth, zinc and manganese. The average contents of these elements are: Mo 24 ppm, Au 0.426 ppm, Bi 404 ppm, Zn 492 ppm and Mn 4619 ppm. There is always a constant relationship between gold and bismuth suggesting that gold follow bismuth in bismuth-sulphides.

The pervasive hydrous alteration of the skarn is succeeded by a later phase of hydrous alteration along quartz and calcite-filled fractures. Along these fractures the skarn recrystallised into very coarse grains of garnet, clinozoisite, amphibole and pyrrhotite in the up to 10 cm wide borders. These phase of hydrous alteration seem not to be connected with scheelite precipitation as the veins are never found to be scheelite bearing, but the scheelite may possibly recrystallise within the borders.

Two sets of vein-filled fractures occur. One set are tiny veins following tension fractures in curving skarn bodies oriented perpendicular to the border. The other set are broader veins running zig-zag across the skarns *(fig. 3)*.

Previously the scheelite mineralisation was belived to be of the stratabound type (Skaarup 1974). To day we agree with Wyn James (1991) that the mineralisation most probably is of contact metasomatic type formed from ascending hydrothermal solutions in the border zone to the Bindal Massif. Scheelite is a relatively common mineral in skarns in this zone, but is only found in small amounts. The rich mineralisation in the Målviken zone should probably be explained by the greater distance to the massif (500 m), and the presence of amphibole-biotite gneiss widely transformed into skarn in the strongly tectonized zone where the numerous granite dykes have created an anisotropy in the deformed body giving pathways for the ascending hydrothermal solutions. The scheelite-mineralised skarn bands are actually often found bordered by granite dykes.

3.7 Ore calculation

At present, rich scheelite mineralisation has been found in 89 exposures of skarn. These findings have been marked on the map *(fig. 5)* and drawn together into 13 zones representing the inferred ore. We expect the mineralisation to have such coherence and richness in these zones that the average grade is 0.7-0.9% WO_3 (see below).

The mineralised zones normally occur in two and up to six sub-parallel levels within the 30 to 60 m wide Målvik zone. The findings lie within the two-thirds of the 900 m long main zone where it is possible to remove soil and vegetation by hacking and digging by hand to reach the solid rock. Accordingly there is a potential for further findings within the remaining one-third (320 m) of the main zone.

The mineralised zones have variable length and width from 1 m to 4.5 m. Taken together, the total length of the 13 zones is 949 m and the average width 2.1 m.

The ore grade was determined in 1995 from sampling 10 kg specimens each representing a 1 m cross-section of skarn or wall rock. All together 12 profiles were analysed giving 1.38 % WO_3 for the 1 m cross-section with the highest tungsten content and 0.43 % WO_3 for the adjacent second highest 1 m zone. This result is in good correspondence with A/S Sulfid-malms 19 analysed profiles which in 1983 gave 1.53 % WO_3 for the richest zone-meter and 0.39 % WO_3 for the second richest zone-meter. Taken together, the two sets of analyses give an average content of 0.9 % WO_3 for a 2 m wide zone.

In 1997 many new exposures of mineralised skarn were found. These exposures have not yet been analysed but their grade has been evaluated visually. It was done by characterising the mineralisation in the field as either scattered, good, very good or extremely good and measuring the width. In accordance with the analyses from 1995, 40 cm of good, 20 cm of very good and 10 cm of extremely good mineralisation correspond to an average content of 1% WO_3 in a 1 m wide skarn zone. The visual evaluation gave an average grade of 0.7% WO_3 in a 2,1 m wide skarn in the 13 zones.

The ore area of the 13 exposed zones is 1898 m^2. Based on the assumption that the frequency of mineralisation and the grade will be the same at depth, the ore volume above sea-level may be calculated as the content of a triangular body with 900 m horizontal length, 2.1 m width and 577 m height. The heights correspond to the average dip of the mineralised zones of 60 degrees. The volume of such a body is 545,265 m^3. The density of the skarn is found to be 3.29 t/ m^3 leading to an inferred ore reserve of 1,794,000 t above sea-level.

We further expect the mineralisation to continue downwards to at least 100 m below sea-level. Corresponding calculations gives a volume of 217,350 m^3 or an inferred ore reserve of 715,000 t for a rectangular body being 900 m long, 115 m height and 2.1 m wide.

The total tonnage of the occurrence is therefore expected to be at least 2,509,000 t of tungsten skarn ore with a grade from 0.7 to 0.9% WO_3. Corresponding to 22,581 t of tungsten oxide (WO_3) or 17,908 t of tung-sten (W) with an ore grade of 0.9% WO_3 and 17,563 t oxide (WO_3) or 13,929 t of tungsten (W) with an ore grade of 0.7% WO_3.

If the grade is 0.9% WO_3 such reserve places the Målviken occurrence as no. 1 in Europe and no. 21 in the world. The comparison was made to the 65 of the world's demonstrated occurrences of tungsten in 1983 with a grade of 0.3% WO_3 or more described by Anstett et al. (1985) *(table 1).*

No.	Deposit	Location	In situ resource 1,000 t	In situ grade % WO_3	WO_3 content in t	Type	Year of discovery
1	Shizhuyuan	China	190,000	0.33	627,000	Stockwork, skarn	1950
2	Mactung	Canada	57,000	0.96	547,200	Skarn	1971
3	Tyrny-Auz	USSR	50,800	0.60	304,800	Skarn	1934
4	Qingliu	China	78,000	0.38	296,400	Porphyry	
5	Damingshan*	China	13,962	1.04	145,205	Stockwork,vein	1960
6	Vostok-2	USSR	22,025	0.58	127,745	Skarn	1968
7	Xihuashan	China	17,428	0.65	113,282	Vein	1908
8	King Island	Australia	7,600	1.03	78,280	Skarn	1911
9	Sangdong	S Korea	8,500	0.86	73,100	Skarn	1910
10	Uludag	Turkey	14,000	0.50	70,000	Skarn	1950
11	Huangsha*	China	3,294	1.75	57,645	Vein	1930
12	Guimeishan*	China	2,175	2.20	47,850	Vein	1918
13	Dzhida Feldt	USSR	10,910	0.43	46,913	Vein	1933
14	Piaotang	China	2,432	1.75	42,560	Vein, stockwork	1908
15	Cantung	Canada	2,600	1.32	34,320	Skarn	1959
16	Pangushan*	China	1,528	1.90	29,032	Vein	
17	Boguty	USSR	4,320	0.60	25,920	Vein, stockwork	1941
18	Dajishan	China	3,890	0.65	25,285	Vein	1918
19	Khao Soon	Thailand	2,500	1.00	25,000	Vein	
20	Dangping	China	2,749	0.88	24,191	Vein	1908
	Målviken	Norway	2,509	0.90	22,581	Skarn	1973
21	Mittersill	Austria	4,500	0.50	22,500	Stratiform	
22	Panasquiera	Portugal	6,100	0.36	21,960	Vein	1894
23	Maykhurinsk	USSR	5,000	0.40	20,000	Vein	
24	Chicote Grande	Bolivia	2,400	0.80	19,200	Vein	
25	Santa Comba	Spain	3,600	0.50	18,000	Vein	1943
	Målviken	Norway	2,509	0.70	17,563	Skarn	1973
26	Xiangdong	China	2,290	0.72	16,488	Vein	
27	Akchatau	USSR	2,741	0.50	13,705	Greisen	1936
28	Salau	France	900	1.40	12,600	Skarn	1960
29	Montredon	France	2,000	0.63	12,600	Vein	1942
30	Ingichika	USSR	2,866	0.43	12,324	Skarn	
31	lul'tin	USSR	1,505	0.80	12,040	Vein	1937
32	Dalongshan	China	1,135	1.00	11,350	Vein	1908
33	Choljlla	Bolivia	2,800	0.40	11,200	Vein	
34	Hungshuichai	China	1,274	0.77	9,810	Vein	
35	Antonovogorsk	USSR	1,179	0.80	9,432	Vein	1910
36	Yochang Dist,	China	488	1.90	9,272	Vein	1910
37	Yubilennoye	USSR	1,432	0.60	8,592	Vein	
38	Shangping	China	695	1.15	7,993	Vein	
39	Xialong	China	851	0.93	7,914	Vein	1908
40	Doi Mok	Thailand	1,000	0.75	7,500	Vein	
41	Kara	Australia	1,000	0.73	7,300	Skarn	
42	Belukha	USSR	1,181	0.60	7,086	Vein	1916
43	Doi Ngorem	Thailand	400	1.75	7,000	Vein	
44	Wengyuan	China	359	1.90	6,821	Vein	1910
45	Yxjöberg	Sweden	1,500	0.43	6,450	Skarn	
46	Verk.-Keyraktin	USSR	1,400	0.45	6,300	Vein	
47	Kami	Bolivia	600	0.97	5,820	Vein	
48	Bukuka	USSR	958	0.60	5,748	Vein, stockwork	1911
49	Bolsa Negra	Bolivia	500	1.04	5,200	Vein-manto	
50	Spokoinyi	USSR	1,000	0.50	5,000	Greisen	
51	Pasto Buero	Peru	1,000	0.46	4,600	Vein	1909
52	Bom Gorkhon	USSR	400	1.00	4,000	Vein	
53	Enramada	Bolivia	500	0.80	4,000	Vein	
54	Krantzberg	Namibia	900	0.40	3,600	Vein	
55	Tasna	Bolivia	400	0.84	3,360	Vein	
56	Borralha	Portugal	700	0.47	3,290	Vein	
57	Mawchi	Burma	600	0.50	3,000	Vein	
58	Chambilaya	Bolivia	500	0.58	2,900	Vein	
59	Kti-Teberda	USSR	475	0.60	2,850	Vein	
60	Lyangar	USSR	540	0.50	2,700	Skarn	
61	Balcany	USSR	386	0.60	2,316	Vein, skarn	1920
62	Yaokanghsien	China	235	0.82	1,927	Vein	
63	Kara Oba	USSR	204	0.80	1,632	Vein	
64	Pueblo Viejo	Bolivia	100	1.00	1,090	Vein	
65	Viloco	Bolivia	44	1.50	670	Vein	

* percentages of WO_3 exceeding 0,3% may not represent Chinese resources of information according to Wegner et al.

Table 1. Demonstrated resources in world's 65 greatest tungsten deposits grading 0.3% WO_3 or more. After *Anstett, T.F, D.I. Bleiwas & R.J. Hurdelbrink 1985: Tungsten Availability – Market Economy Countries. United States Department of the Interior. Bureau of Mines, Information Circular 9025, 50 p.* Year of discovery after *Wegner et al (op. cit. p.26).*

3.8. Recovery

In 1983 A/S Sulfidmalm tested the Målviken ore by flotation at the laboratory of AB Statsgruvor using the same method as for the Swedish scheelite ore at Yxsjöberg. The tests gave 90% recovery of scheelite, in a concentrate with 74% WO_3 corresponding to a 94% scheelite content. The concentrate is considered free of any damaging impurities and is as such a marketable product. It was noted in the report that the recovery was expected to be lower on a producing scale, presumably about 80%.

4. TUNGSTEN PRICE AND MARKET

4.1. Properties and use of tungsten

Tungsten has the highest melting point 3,410^{o}C among metals and is one of the heaviest elements with a density of 19.3 g/cm^3. It has high corrosion resistance, good thermal and electrical conductivity and a very low coefficient of thermal expansion. At temperatures over 1,650^{o}C tungsten has the highest strength of all metals.

These properties determine a long row of special uses and tungsten is regarded as a basic component in the machinery, oil, drilling, construction, electrical, electronic, chemical and armament industry. Due to the relatively high price, tungsten is used for special purposes and in high-quality products where other and cheaper substitutes are not adequate.

Mill products of tungsten are used for lamp filaments and electrodes in the electro-technical industry, plates for heat end radiation protection in the atomic energy industry. The specific weight makes it useful in ball-mills, earlier also as counterbalances in aeroplanes. The last use is now replaced by depleted uranium.

Tungsten is a high quality alloying agent in steels – especially when wear-resistance at high temperature is required. It is moreover used in super-alloys of many different types.

	W.Europe	Japan	USA	China
Hard metals (WC)	62%	45%	60%	40%
Steels/Superalloys	24%	25%	21%	48%
Mill Products (W)	6%	10%	15%	4%
Chemical comp./Others	8%	20%	4%	8%

Table 2. Estimated consumption by end-use sector 1996. *Met.& Min. An. Rev.1997.* In China tungsten instead of molybdenite is more widely used as an alloying component.

The greatest use of tungsten is, however, as a hard metal in form of the carbide WC with a hardness close to diamond. Tungsten carbide is cemented with cobalt into bits for cutting tools, mining tools and oil drilling equipment.

Chemical components of tungsten are used for their fluorescing properties in TV sets, for corrosion protection, pigments and catalysts in the refinery industry and many other purposes.

The future use of tungsten may increase due to the “Green Bullet” program set up by the US Fish and Wildlife Service. Recently the service has granted final approval of two tungsten-based shot products for hunting waterfowl and coots – tungsten-iron and tungsten polymer shot. One component of the “Green Bullet” program was to find an alternative for the lead-antimony cores in small-caliber ammunition projectiles. At present tungsten-nylon and tungsten-tin were identified as potential substitutes for lead-antimony in the cores of 5.56 mm projectiles.

Golf clubs and cores in golf balls are other new uses of tungsten.

Tungsten has a strategic significance since it is used in armour steels and in armour penetrating ammunitions. From 1986 depleted uranium has widely replaced the last mentioned use of tungsten in US. Depleted uranium is waste from atomic energy production and would except from its use as counterbalances in aeroplanes, otherwise have been stored.

Ammunition with cores of depleted uranium was widely used during the Gulf war in 1990 for the first time. Depleted uranium is a classified material. It is said to be harmless, but is found to contain plutonium and more of the radioactive uranium isotopes. The Gulf war syndrome, i.e. cancer among American and British soldiers and the local population in Iraq who were or live in the battlefields, has been ascribed to the spread of depleted uranium. According to an article in Sunday Times, Sept. 3 2000, Dr. Asaf Durakovic, who is professor of nuclear medicine at Georgetown University, Washington, and the former head of nuclear medicine at the US Army’s veterans’ affairs medical facility in Delaware, has together with his team of American and Canadian scientist discovered life-threatening high levels of depleted uranium in Gulf veterans 10 years after the desert war. Durakovic concluded that troops inhaled the tiny uranium particles after American and British forces fired more than 700,000 shells of depleted uranium during the conflict. The findings begin to explain for the first time why medical orderlies and mechanics are the principal victims of Gulf war syndrome. British Army engineers who removed tanks hit by depleted uranium shells from the battlefields and medical personnel who cut off the clothes of Iraqi casualties in field hospitals have been disproportionately affected. In the UK more than 400 veterans are estimated to have died from “Gulf war syndrome”.

In Europe the danger of depleted uranium has recently gained focus with the high proportion of blood cancer (leukaemia) among Italian soldiers stationed in Croatia in areas where Serbian tanks were attacked by ammunition containing depleted uranium. Moreover, the impression is gained that the US Army instructions to soldiers treating tanks and victims from such attacks are very restrictive.

Besides radiation, depleted uranium has an environmental poisoning effect damaging drinking water and vegetation.

WHO and NATO have decided to consider the danger of depleted uranium in systematic investigations. If the investigations lead to the conclusion that depleted uranium should be replaced by tungsten in ammunition, a considerably increased demand for tungsten would be the consequence.

4.2. Production and consumption of tungsten

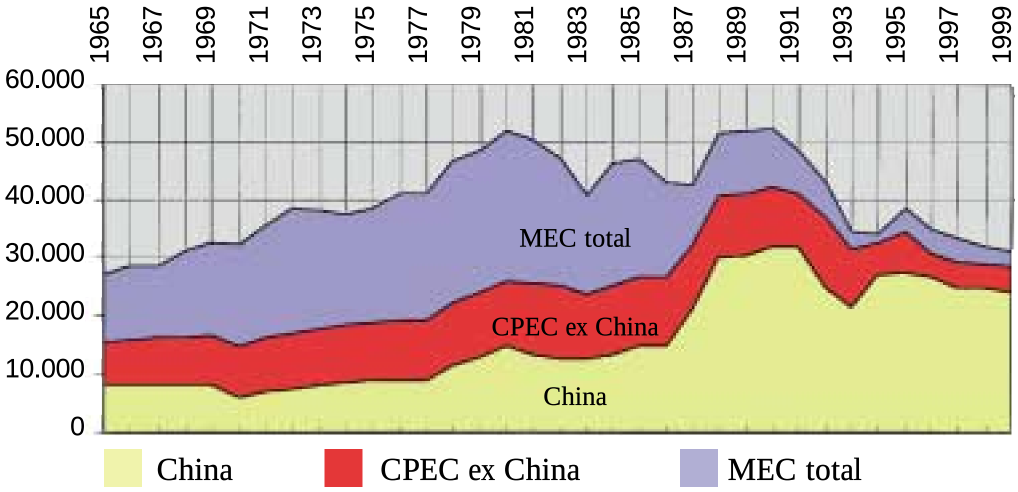


Fig. 6. World tungsten production in tW 1965 – 1999 in Market Economy Countries (MEC), Centrally Planned Economy Countries (CPEC) exclusive China and China.

Previously the world statistics on production and consumption of tungsten in Centrally Planned Economy Countries (CPEC) were based on estimates. From 1992 China has submitted its own figures for production.

The world's production of tungsten concentrate rose from 27,000 tW in 1965 to 52,000 tW in 1980. It varied from 41,000 tW in 1983 to 47,000 tW in 1985 before the increase to a constant level of 52,000 tW in the period 1988 – 1990. After this it dropped steeply and reached the present level of about 31,000 tW – 35,000 tW *(fig. 6).*

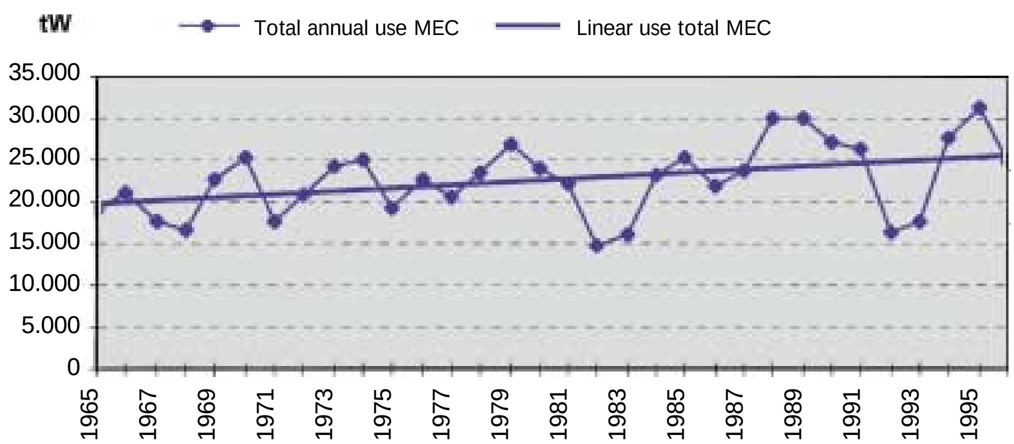


Fig. 7. Tungsten consumption in Market Economy Countries (MEC). *Source: 1965-1984 Minerals Yearbook, 1985-1996 Mining Annual Review.*

During the 1965-1999 period there has been a marked shift in producers. The production within the Market Economy Countries (MEC) decreased from a maximum of 26,000 tW in 1980 to a present level of 2,600 tW. China having a production of about 8,000 tW in the sixties and the seventies raised production at the beginning of the eighties to 15,000 tW and further to 30,000 tW at the end of the eighties to become the worlds single dominant producer. After 1991 the production has dropped to about 25,000 tW. The production within the remaining CPEC has been about 10,000 tW. It decreased to about 4,500 tW, which is the present level *(fig. 6).*

The consumption of tungsten within the MEC has slightly increased from 19,000 tW in 1965 to 26,000 tW in 1996. The small average increase of 219 tW per year is easily overlooked by the dramatic change in consumption from one year to the next *(fig. 7).*

Today China is thought to be the world's leading consumer with a consumption of about 11,000 tW per year.

The greatest drop in consumption is in the former CPEC from about 19,000 tW before 1990 to about 2,000 tW to day.

According to the estimates in Metals and Mining Annual Review, there is a deficit in production compared to consumption from 1993 to 1996 *(table 3).* This deficit is expected to have been covered by sales from stocks in Russia, China and Kazakstan. Today USA also sells tungsten from stocks *(see p. 25).*

4.3. Tungsten price

Tungsten goes into world trade as concentrates either of scheelite ($CaWO_4$) or of wolframite (Fe,Mg WO_4), as ammonium paratungstate (APT) and as ferrotungsten (FeW). The Metal Bulletin of London regularly submits prices on these items. APT is an intermediate product for tungsten carbide (WC). The APT price is said to best reflect the demand in the market, as it has the highest amount in open trade

Here we only look at the price variation of concentrates. Tungsten concentrates trade as standard in US$ c.i.f per mtu (mtu = metric tonne unit = 10 kg) contained WO_3 in concentrates with more than 65% WO_3. The price of high value and low value concentrates is normally given. Concen-

Figures in tW	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996
Production MEC	19,026	15,151	10,857	10,998	10,927	10,100	7,550	6,100	2,600	1,695	3,530	3,870
Production CPEC	29,380	28,913	32,377	40,962	41,235	39,755	39,350	31,889	28,125	22,752	31,790	26,520
World production	48,406	44,046	43,234	51,960	52,162	49,855	46,900	37,989	30,725	24,447	35,320	30,390
Consump. MEC	25,262	21,839	23,619	30,118	30,038	27,200	26,400	16,250	17,600	27,593	31,350	24,850
Consump. CPEC	19,500	19,500	19,500	19,500	19,500	17,950	16,000	15,500	15,500	14,560	13,250	12,450
World consump.	44,762	41,339	43,119	49,618	49,538	45,150	42,400	31,750	33,100	42,153	44,600	37,300
Balance	3,644	2,725	115	2,342	2,624	4,705	4,500	6,239	-2,375	-17,706	-9,280	-6,910

Table 3. Production and consumption of tungsten in Market Economy Countries (MEC) and Centrally Planned Economy Countries (CPEC) 1985-1996. *Source: Metals and Mining Annual Review.*

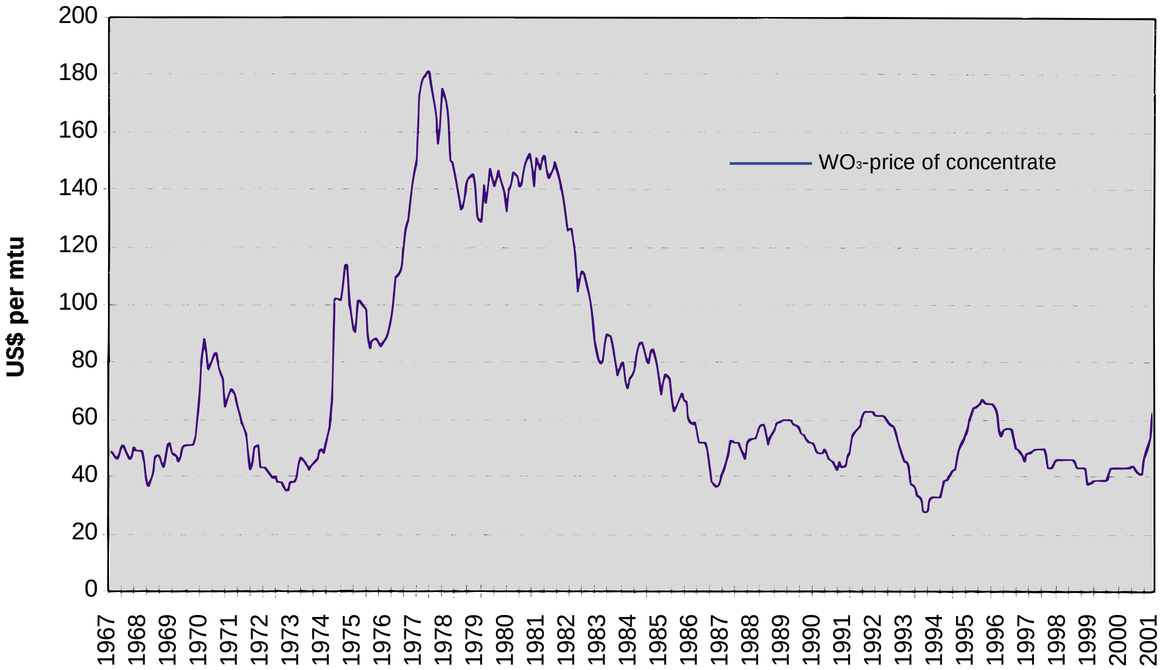


Fig. 8. Price variation 1967 – 2001 in tungsten concentrates on European Market. US$ c.i.f. per mtu (10 kg) contained WO_3.

trates of scheelite would correspond to the high value price due to the purity compared to concentrates of wolframite.

Figure 8 shows the price variation of tungsten concentrate on the European market 1965-99 as a simple arithmetic average of high and low price.

At the beginning of the period from 1965 to 1972, the selling of concentrates from the US defence stock (built after the Korean War) determined the price. It varied from $45 to $51 until 1970 when a short stop in the export to Europe (during the Vietnam War) caused the price to rise to $77. Sales were regained and the price dropped to $39 in 1972.

China stopped the export of tungsten to marked economic countries in 1972. This happened after the export to the Soviet Union was regained in 1970. The oil crises in 1973 led to increasing demand of tungsten in drilling and nuclear power industry, as well of growth in the military industrial complexes during the Cold War under Brezhnev (1964-82) in the Soviet Union and Johnson, Nixon, Ford and Carter in US (1963-81). The price of tungsten rose from $44 in 1973 to $180 in June 1977 on the London Market, the highest recorded. It led to the opening of new mines in market economic countries. The price dropped shortly after but kept at a high level of $140 to $145 until 1982.

Under the Chinese Spring after the death of Mao in 1976, the economy was liberated and tungsten production was increased from numerous mines. The USA established diplomatic relations with China in 1978 and the export of tungsten was maintained. In 1982 recession started in the western world. The tungsten price dropped to $106 in 1982 and further to $80 in 1983. USA mines closed down. A new method for recycling tungsten carbide from tools was developed. Today 36% of the USA demand is inferred to come from recycling. The armament industry in USA started in these years to replace tungsten in missiles and ammunition by depleted uranium.

In 1985 the tungsten market suffered from oversupply. Twelve Chinese trade organisations were in mutual competition offering tungsten and tungsten products to the market. China's dominant position came from 1985 to 1989 when ferrotungsten and APT were offered at prices corresponding to the price of concentrates, which was from $56 to $47. This led to the closure of mines, ferrotungsten and APT industries in Europe, Japan and other marked economic countries. The USA maintained, however, APT industries under protection.

In 1987 China established a state owned monopoly for tungsten trade in an attempt to stabilise the prices. The breakdown of Soviet Union and Eastern European Countries from 1989 to 1991 created a drastic drop in consumption from 52,000 tW in 1989 to 32,000 tW in 1992, explaining the fall in tungsten price from $56 in 1991 to $34 in 1993, the lowest so far.

Russia gained the status in 1993 of a most favoured trading partner to USA. This allowed Russia to export tungsten without the same import tariff as on the exports from China. Before 1989 the Soviet Union imported 5,000 tW per year from China. After 1989 Russia and other CIS countries (Commonwealth of Independent States) became exporters in competition to China In Russia five mines have been in production but now all are either closed or producing on a very low level.

According to statistics in Metals and Mining Annual Review, there has been a major deficit in production of 36,253 tW from 1993 to 1996 *(table 3)*. This deficit is expected to be covered by sales from stocks in China, Russia and Kazakhstan. Many authors have expected an increase in the tungsten price due to emptying the stocks, but this has not been the case yet.

The price of tungsten reached $64 in 1995. China forced the closure of mines for a short period. The purpose was to influence the price and gain control over the production. After stronger demand in 1995, the market maintained its normal depressed level in 1996 and the price dropped to $44. Such a low price is below the production costs even in China. Subsidies have kept the production going. The owners of the mines are either the state, the military or the local authorities.

The 18 state-owned mines in China have been the biggest and richest wolframite deposits in the world. They have, however, been in production for a long period. Their average age is 40 years. Only half can maintain production for more than 10 years. They face various problems – some mines are running out of resources, some are short of funds, and some have excessive operating costs. The mines carry a heavy social burden to support schools, hospitals and a large number of retired workers. Today two-thirds of the production comes from locally operated mines.

In 1999 the Chinese Government took several steps to control the release of tungsten. Illegal mines and mines causing environmental damage or resource destruction were ordered to halt. Permits for new tungsten projects were not given before December 31, 2000. In June 1999 the Government reduced the number of export licenses and in July set minimum prices for tungsten products which incrementally are to be raised during a 3-year period.

The rise in concentrate prices reflects these decisions. The low –high price increased from \$33 –\$44 in July 1999, to \$34/40 -\$48 until August 2000, succeeded by \$36 – \$50 September, \$40 – \$53 in October, \$42 - \$55 in November, \$50 - \$55 in December 2000 and finely \$55 – \$73 in January 2001 according to Metal Bulletin.

In spite of the low prices at that time, the USA reopened for quoted sales from the National Defence Stockpile in 1999. The sales have been suspended since 1985 when the stock contained 37,300 tW. Uncommitted inventory authorised for disposal amounts of 33,800 tW in September 2000.

4.4. World tungsten resources

The International Strategic Mineral Issues. Summary Report - Tungsten by Werner, A.B.T., W. David Sinclair, and Earl B. Amey. U.S. Geological Survey circular; 930-O; 1998 estimated the demonstrated tungsten resources of type R1E to 379,000 tW. At the 1995 world production level of about 31,000 tW, the resources appear to be sufficient to last only until the year 2007. However, this figure does not adequately reflect the economically exploitable parts of tungsten resources in major deposits in China, the former Soviet Union and other nonmarket – economy countries. If economic conditions are suitable, the total tungsten resources in the inventory (4,728,000 metric tons of tungsten) would likely suffice to satisfy levels of world demand that existed in 1995 for tungsten well into the 21st century.

There is no doubt that China will be an important tungsten producer in the future, but the production has been high for a long period. If we compare the resources in the original rich deposits in production 1983 listed by Anstett et al. *(op cit. p. 18)*, these resources should have been used up by now. In the future China will have to produce from occurrences with a lower grade from 0.3% - 0.5% WO_3. This is in accordance with reports of many state-owned mines being exhausted about 2005.

Also the former CPEC have an important part of the resources, but they never succeeded in becoming self-sufficient. They had to rely on imports from China. Their occurrences are generally reported to grade about 0.5% WO_3.

4.5 The economic forecast for tungsten production

After a long period of low prices due to the Chinese subsidies of production and selling from stocks in other countries, the price increase since July 2000 is most probably the first part of the way up the mountain.

The forecast for tungsten production is promising at least over a period of 5 years for the following reasons:

The world's stocks of tungsten are depleting. When the stocks are used up the present over-supply situation will change into under-supply as the production capacity is lower than the consumption.

The world's demonstrated tungsten resources would be exhausted in 2007 by a consumption of 31.000 tW corresponding to production in 1995.

China has apparently gained control over tungsten export and production amounting 78% of the world in order to make it profitable in a narrow economic sense. Paradoxically this is to be seen as a sign of liberation of the Chinese economy leading to closure of unprofitable mines. Today China is the world's single greatest consumer of tungsten. Continuing economic growth should increases the domestic demand. Only half of the main Chinese tungsten deposits may maintain production for more than 10 years. Gradually the production should be supported by new workings in deposits with a lower grade of 0.3-0.5% WO_3. Development of these will demand investments from funds which can no longer be expected to come from the state. Tungsten production has not the same importance to the Chinese economy as previously.

Reconstruction of the economy in CIS especially the Russian should raise the demand for tungsten for all purposes. Previous consumption was 8,000 tW, 15% of the total.

The USA have in these years increased budgets for armour production after the fall since 1991. The present and the coming NATO partners have the same needs.

WHO and NATO investigations of the health consequences and the environmental impact of the use of depleted uranium in the Gulf and Balkan Wars may lead to a return to tungsten in armour penetrating ammunition and war-heads in missiles. This, together with the new use of tungsten in golf clubs and green bullets for hunting, may increase the demand considerably.

5. DEVELOPMENT PLAN AND INVESTMENT IN THE MÅLVIKEN OCCURRENCE

5.1. The Målviken prospect

The following conditions makes the Målviken tungsten skarn project a promising prospect.

The ore grade seems fairly high at 0.7-0.9 % WO_3 and the inferred ore reserves place the occurrence in the best part of the medium-sized deposits possibly as no. 1 in Europe and no. 21 in the world. By means of flotation the scheelite may be concentrated into a marketable product with an recovery of 80%. Gold and bismuth together with construction material may be by-products. The ore is free of any contaminating components. Tungsten is considered to be a non-toxic component.

The occurrence has a unique location in relation to infrastructure. It is crossed by a main road and power line. Water can be taken close by. A landing quay for deep-seated vessels may be constructed at Tosenfjorden at the main entrance to the mine. The nearest living house is at a distance of 2.5 km. But the area is urbanised and a skilled labour force is to be recruited locally. The airport is in Brønnøysund the local main town with 3.600 citizens, and is 70 km away.

The development plan for the occurrence includes continuation of the ongoing exploration phase and a later phase of mine construction and production, when the market price of tungsten is at profitable level.

5.2. Exploration phase

The exploration phase in continuation of the present investigations comprises:

a) Drawing of detailed topographic map over the zone.

b) Tracing the known mineralisation and searching for new by trenching and uncovering to solid rocks as far as possible. Geological mapping. Structural analyses. Blasting. Systematic sampling of cross-sections for each 5 m for assays of W, Au, and Bi. Inspections of thin sections.

   The aim of this part of the program is to obtain secure information on the ore grade, the size and the form of the ore bodies over approximately two-thirds of the expected zone of mineralisation. Comparatively detailed information may not be obtained from drillings as the steep slopes would prevent closely spaced drilling sites and the mineralisation is coarse grained and irregularly spaced.

c) The recovery of ore is to be tested by relevant methods on a laboratory scale. The tests should also show if it is possible to concentrate the low content of presumed gold-bearing bismuth sulphides into a definite phase.

d) Exploration drilling from four positions throughout the zone including two cores at different angles at each position.

The drillings should test the mineralisation at deeper levels. They are parts of the exploration drilling program, which is to be carried out before the final decision on mining and mine construction is to be made *(see below)*.

e) Final account on planning and environmental questions in order to receive a mining permission.

The investments during this first phase of exploration have been estimated to about US$ 505.000 or 4.5 million NOK (Norwegian kroner). The sum includes the above-mentioned points except for point d) the drilling program. The drillings (1,200 m) of about US$ 110.000 or 1 million NOK are part of the total exploration drilling program set up by Karl-Åke Johansson *(see below)*.

The Norwegian development funds may give financial support for 30%, in certain cases 50% of the exploration costs.

5.3. Model for mining and scheelite production in Målviken

In order to place the investors on a realistic decision ground, a model for mining and scheelite production in Målviken has been set up by mining engineer Karl-Åke Johansson, Minexp AB, Sweden. His report is included as an appendix *(p. 31 – 44)*.

On the basis of the presented geological knowledge, Karl-Åke Johansson has evaluated mining methods and mine construction and calculated the capital expenditure and operating cost at the actual price-level. The cash flow is compared to a revenue from scheelite concentrates at a market price of US$ 50 per mtu.

The total investment including the above-mentioned first phase of exploration is calculated to US$ 15,151,000 or 134,850,000 NOK in a 300,000 t per year production of ore grading 0.8% WO_3 per t in situ. The break even point after 10 years of production at a rate of 0% return lies at US$ 53 per mtu of contained WO_3. The present high price $73 has a 18% rate of the investments per year. A market price of US$ 80.5 has a 25% rate of return.

5.4 Risk factors

The two main factors which make the Målviken project promising are the relative high WO_3 content in the ore and the unique location of the occurrence.

We might have over estimated the ore grade at 0.7% - 0.9% WO_3, as the mineralisation is quite irregular and difficult to evaluate. We have been aware of that risk and tried to be realistic. This was done by giving the contribution from the extremely rich mineralisation, the breccias, veins and pockets, only minor importance. Accordingly we may actually have under-estimated the ore grade. The whole economy of the project is strongly depending on the ore grade. If the in situ ore grade is 0.5% WO_3 the 0% rate of return would be at US$ 85 per mtu, the 15% rate at US$ 111 and the 25% rate at US$ 129. Therefore the first main aim in the development plan is to gain secure information on the ore grade *(see p. 27).*

The exposures of rich, very rich and extremely rich mineralisation in all levels from 0 m and up to 500 m above sea-level make it relatively safe to assume a continuation of the mineralisation to only 100 m below sea-level.

Tungsten occurrences all over the world are relatively small. The larger ones are the accumulations of smaller ones. Only minor occurrences have reported grades above 1.1% WO_3. They are all older findings known for many years. Therefore competition from new rich occurrences is not to be expected. Other scheelite skarn occurrences have the same irregular distribution of mineralisation as the Målviken prospect.

The world's tungsten reserve is either contained in old workings suffering from bad economy or in virgin deposits. In both cases investment has to be done to make production going.

Tungsten is a relatively expensive metal. Substitution from other metals has been going on as far as possible. New composite ceramic materials, polycrystalline diamonds and so on are said to be in competition with tungsten carbide. However, this is not visible from the statistics. Tungsten carbide has the greatest increase in consumption among all the tungsten products.

Mining activities in the Målviken area require permits from various authorities regarding prospecting, development, mining, taxation, employment standards, occupational health, waste disposal, land use, environmental protection, mine safety and other matters. It can not be guaranteed that the project will obtain these permits. A priori, however, none of the matters seems to exclude mining activities. The ore has the advantage of being free of any contaminating components. The present project is supported by the local and regional authorities as a matter of economic development in the region.

5.5 Claims

The Målviken occurrence is held by "mutings" owned by Kåre M. Lande. Mutings are a type of pre-claims giving the right to prospect, survey and carry out drillings and limited test operations. The pre-claims may be held for seven years, with a possible 3-year extension. The pre-claims are succeeded by claims. A claim may be held for ten years, with a possible 10-year extension. Claim holders can carry out development including test mining of up to 10,000 t of ore annually. New mining operations require permission from the agriculture and forest authorities in relation to the Planning and Building Act concerning pollution of the environment.

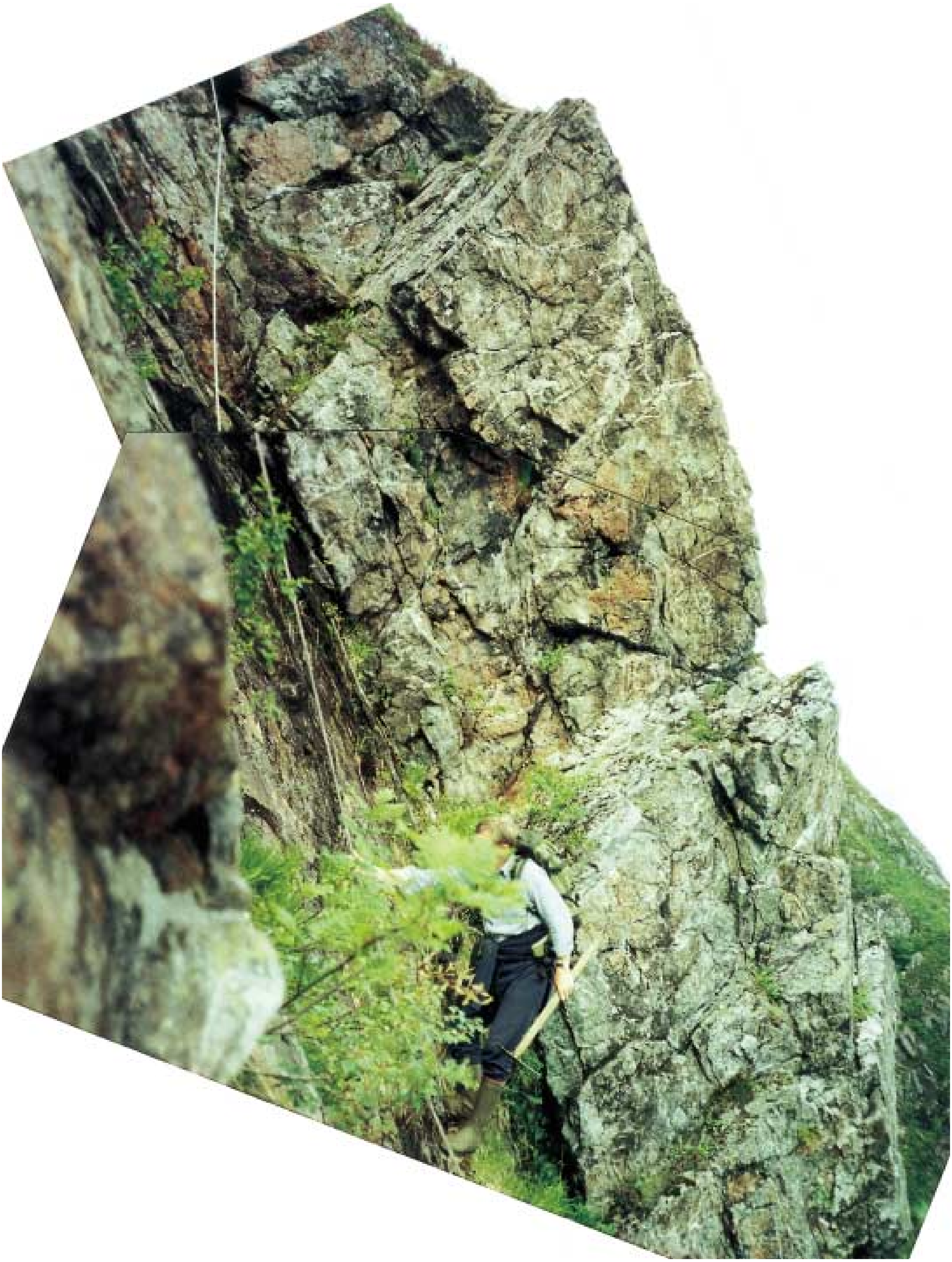


Fig. 9. Scheelite-bearing skarn in “Peter zone” 350 m above sea-level

Minexp AB

ME-2001-P-01

Målviken Tungsten Project

Mining report

Made for

Peter Skaarup and Kåre M. Lande

2001-02-01

Mining study of the Målviken tungsten project.

SUMMARY

Geological information today (January 2001) has resulted in a possible mineralization of 2,500,000 tons at 0.7 to 0.9 % WO_3. The mineralization consists of 13 mineralised zones containing ore lenses with an average width of 2.0 m. The mineralization is known at surface through extensive geological mapping and sampling of the outcropping host rock. The tonnage is calculated with the assumption that the same configuration and grades are present from –100 m below sea level to 500 m elevation and along a 900 m strike. (Målvikzonen of Peter Skaarup and Kåre M. Lande).

The following study is based on an assumed mineable ore of 2,500,000 tons at an average grade of 0.8 % WO_3. In order to develop the existing "inferred ore" to "ore reserve" of the categories "proven and possible" additional prospecting and interpretation has to be performed.

Underground mining for indicated areas and with indicated complexity of the mineralization is limiting production rates to the neighbourhood of 300,000 tons per year.

Production rates, ore grades, tonnage and ore geometry are major factors in mining together with rock mechanics considerations. Production rates are highly dependent on time-consuming stoppages, that is on the time that has to be spent on rock support and grade control activities.

Mining is proposed to be done with blast hole stoping starting from sea-level and going up. The mine is proposed to be developed from an access drift (stoll) from the NW side of the fjord passing under the road and following the strike of the ore. This drift can also be used for additional exploration drilling. Mining will be done on a small to medium scale and is heavily dependent on ore geometry.

Table of contents

Background

The following study is made on behalf of Peter Skaarup and Kåre M. Lande in order to be included in a prospectus for investors for further development of the property.

The project area of the Målviken Tungsten Project is situated on the NW shore of Tosenfjorden on the slopes of Landnubben.

There is presently no operation or development going on in the property and all evaluations are based on a grassroots scenario where all capital investments has to be included.

Ore reserve and geological setting

The geological setting and estimation of the ore reserve used in the following evaluation is taken from the report "Målvikzonen – 1995….". No additional evaluations are attempted in this study since it is outside the scope of work.

The mineralised zones are sub-parallel zones in a highly tectonized skarn environment. Typical zones are 1.0 – 4.5 m wide, 15 to 60 m long and 20 to 40 m thick at a density of 3.29 t/m^3 and a grade of 0.7 – 0.9 % WO_3. Tonnage from one ore lens using these numbers is 1,000 to 35,000 tons. The total number of ore lenses for an ore reserve of 2,500,000 tons is in the range of one hundred.

For this mining evaluation a simulated ore configuration, based on existing geological reports, is used (appendix 1)

For initial calculations a mineable grade giving a mill feed of 0.72 % (0.8 % + 10% mining dilution) WO_3 is used to make an initial estimate of possible ore reserves. It should be noted that 10% dilution is the case with a regular and well-defined ore body. See Appendix 2.

Geological information so far is not sufficient for calculation of a proven reserve. All ore reserve figures are in the category of inferred.

Grade control

It is essential in order to control the revenues to have a good grade control system in place not to lose WO_3 containing materials or to dilute the ore with barren waste rock. This is especially important in a situation with narrow ore lenses and a highly tectonized setting.

The site geologists can probably make a grade control from exposed rock surfaces and from drill cuttings by using uv-lamps.

If the mineralization is such that the mineralised zone contains reasonable grades outside the main defined ore a larger scale of mining can be considered since "dilution" will contain some additional value.

The following approach is proposed:

1. Exploration drilling.

Long holes from surface or underground exploration positions, average length 150 m.

The main purpose is to define mineralised positions and to give planning information for the development ramps. Initial drilling should preferably target the area for the initial development, that is sea-level and 50 to 100m up.

2. Definition drilling

Shorter holes from the underground drifts, 60 to 120 m long with a pattern of ~20 x 20 m.

The main purpose is to give information for planning of drifts in ore and to establish an ore reser-ve.

Geo-mechanical considerations

Geo-mechanical background information

Information from the geological work performed up to today indicates good rock conditions with predominantly homogeneous rock and absence of chlorite and talc. There are a number of NE-SW striking fracture zones with widths of 1 to 10 m. These highly fractured rock zones could be a pro-blem especially if larger spans are considered to be excavated.

The present information gives no indication whether the host rock for ore is either more or less competent than the surrounding waste rock.

No in situ stress has been measured.

Geo-mechanical considerations

The present information does not give any concern for stability and drifting problems, although a number of factors must be evaluated for a final design of stopes and development drifts. If larger excavations are to be made, for example for an underground mill, considerable additional studies will have to be made.

Proposals

For future work, some geomechanical mapping needs to be performed such as RQD logging and other work in order to develop a rock mass index.

Choice of mining method

Choice of mining method should be made with consideration of the following factors:

- Start of production
- Rock stability.
- Dilution.
- Stress direction and size.
- Production rate.
- Plunge of the ore body.
- Irregular ore configuration in both horizontal and vertical extensions.
- Grade control possibility.
- Metal grades in surrounding waste/dilution.
- Metal losses in fill or draw-pints.
- Low cost at the expense of high recoveries with low value ores.
- Mine production capacity

Considered mining methods.

Sub-level stoping, long hole stoping

Sub-level stoping is a low cost method but it results in large open stopes during the draw out phase. In a small application like Målviken, a variation is proposed with development and draw drifts mainly in the ore.

The irregular ore zones are troublesome for dilution and ore losses. It is also expected to be hard to fine-tune the layouts to fit the irregular ore geometry. See appendix 2.

In order to limit the ore losses in crown pillars, the evaluation is based on dividing the ore into stopes each 75 m high

Cut and fill mining

Cut and fill mining is a somewhat more expensive method but gives a higher degree of control over the drifts and layouts to fit the irregular ore geometry.

Overall the mill feed grade with cut and fill mining should in this type of application be about 10% higher than with sub-level caving.

In this application back filling with waste rock from development is considered.

Comparison of different mining methods	Cut & Fil mining	Sub-level stoping
No apparent horizontal stress direction	-	-
Production rate and start of ore production	+	
Plunge of the ore body		
Narrow ore bodies	--	
Irregular ore configuration in both horizontal and vertical extensions.	-	- -
Grade control possibility.		-
Metal grades in surrounding waste/dilution is rather low	?	??
Metal losses in fill or draw-points.		
Vertical ores.	+	+
Low cost at the expense of high recoveries with low value ores.	-	+
	3 (-)	2 (-)

Conclusion

Based on the mentioned conditions, sub-level stoping is recommended. The final choice of mining method should, however, be made based on the conditions experienced during test drifting before detailed layouts are made for development.

Costs and revenues

All costs and revenues are calculated in US$, totals in US$ and NOK (Norwegian kroner) using an exchange rate of 8.90 NOK/US$. Metal prize used is US$ 50 /mtu.

For comparison purposes an exchange rate of 1.06 NOK/SEK (Swedish kroner) is used.

All relevant cost except cost for underground mining is based on actual figures from ongoing production. The cost for underground mining is derived by comparison with Finnish and Swedish operations with similar methods.

Mining cost accuracy is in the range of order of magnitude.

Mill operation and capital costs are estimated only to give some background for a preliminary evaluation.

Technical recovery is based on the AB Statsgruvor study.

Commercial recovery is a guesstimate for sensitivity purposes. The term commercial recovery is used to cover all costs and deductables after the product leaves the mining area.

Development

Scandinavian experience indicates cost for development to be 10,000 –13,000 NOK per m. The higher figures include cost for drifting in poor rock.

Swedish contractors indicate a cost of 400 SEK per m^3 plus rock support.

Dilution and loss of metals

Considerable efforts have to be taken in order to avoid dilution and loss of metal during mine production. If the section in appendix 2 should be representative for ore loss and dilution, actual ore loss should be 16% and dilution should be 69 %. Dilution will contain some metal that will give a better final result but dilution grades must be evaluated more closely.

Mining cut & fill

Mining productions of 150,000 tons per year and level are normal with ore bodies of similar size and irregularity. Productivity in C&F mining is normally consistent throughout the full mining cycle until ~90% of the stopes are recovered. Typical costs for large regular ore lenses are 100 NOK/ton including cost for ongoing development that accounts for approximately 40% of the volume. This cost is indicating a direct mining cost of 100 to 150 NOK per ton; the higher is considering high costs for small ore bodies.

Sub-level stoping, long hole stoping

Mining productions of 150,000 tons per year and level are normal with ore bodies of similar size. Productivity in sub-level stoping is normally decreasing when ~ 60% of a mining level is mined out. Typical costs are 70 NOK per ton.

Sub-level stoping

Development

The main access (stoll) is developed from the small flatter area on the sea side of the main road. Development starts going down with a slope of 1:6 to get safely below the road. After the road is passed development is done slightly uphill about 2% for about 900 m until the expected north end of the ore is reached. From this main access at about 400 m N the main ramp is developed at an average slope of 1:8 to the upper parts of the mineralised zone. See appendix 4.

Sizes of development and transportation drift are 25 m^2. This is sufficient for transportation with normal highway trucks. Ramp slope of 1:7 or 14% can easily be managed by contractor equipment. Curves are drifted with a flatter slope giving an overall slope for costing purposes of 1:8. For exploration and definition drilling a number of drilling bays must be made in conjunction with drifting of the main development. The volume of these bays is estimated to 5% of main ramp volume, but at a lower unit price. The main ramp should be designed to fit in both C&F mining and sub-level stoping in order to allow a final choice of method as late as possible.

The ramp access should be equipped with a portal and doors at surface to avoid freezing problems in the mine.

Ventilation can be supplied from the portal during the initial period by ventilation ducts until the system is completed with a ventilation shaft reaching the surface at 260 m elevation. After this, ventilation air can flow freely into the drift assisted by a surface fan at the top of the shaft. A system for heating the air must be in operation to avoid freezing in the winter months. The ventilation system needs to follow the main ramp as it is progresses upwards. Heating is preferably done with propane heating direct in the stream of air or by using a heat pump taking heat from the sea-water.

Mine water must be pumped to the main ramp from where it can flow by gravity to the portal from positions above sea-level. From deeper levels, mine water is pumped by submersible pumps in small pump sumps. The quantities of water from the mine can only be establised from observations and test pumping in drill holes.

Mine water is discharged into a surface pond with an oil trap. After quality control mine water can be either used for process water or discharged in the fjord.

Conclusion and recommendation

Underground mining with sub-level stoping is, based on present information, a viable option for further development of the property. Mining can be done with sub-level stoping or cut & fill methods.

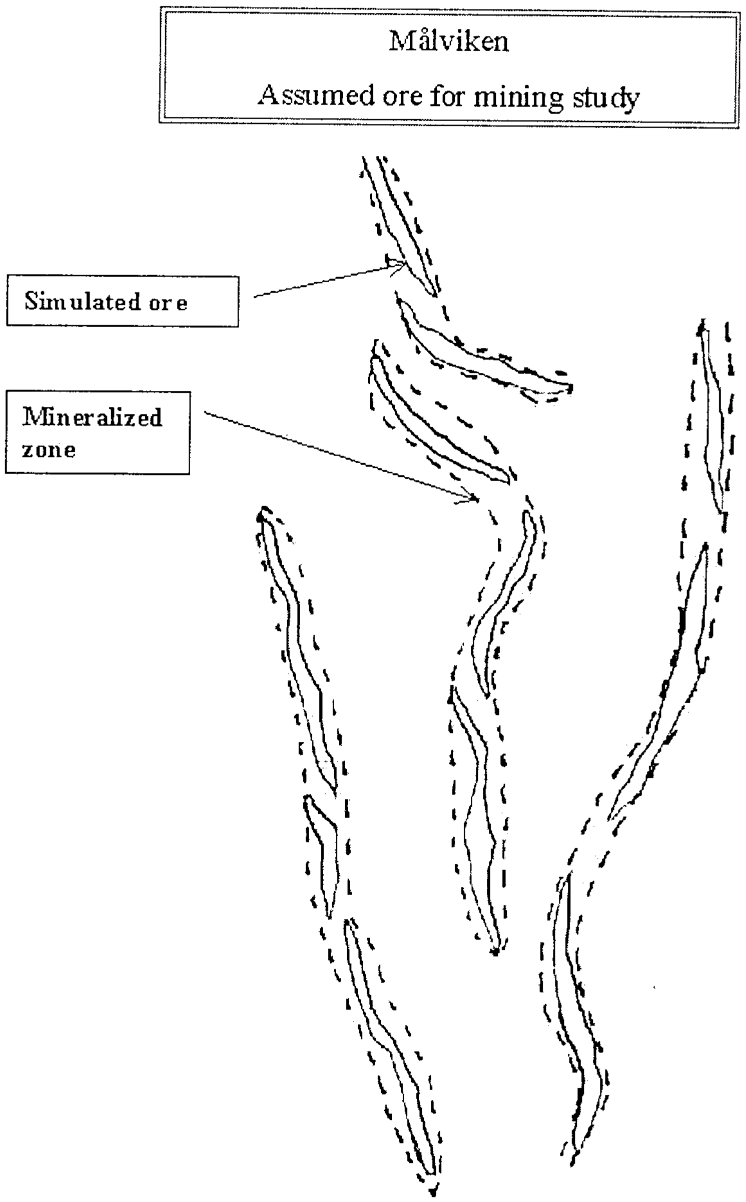
SLS gives a slightly lower cost but a lower recovery than C&F. However C&F mining is more suitable than SLS for poor rock conditions and very irregular ore bodies.

As the initial development can be made in a similar manner for both methods and the costs are comparable, the final decision about mining method can made when more information is available from the initial drifting of the access ramp.

Milling and industrial areas have not been studied. However, it seems unlikely that there is a sufficiently large area in the mines immediate area for a mill and office. Alternatives could be Tosbotn village or further SW of the mine. A mill installed on a barge could be an alternative.

Disposal of tailings is another area that needs to be considered further. Disposal directly on the bottom of the fjord is to be preferred from a cost and technical point of view but may not be permissible from an environmental point of view.

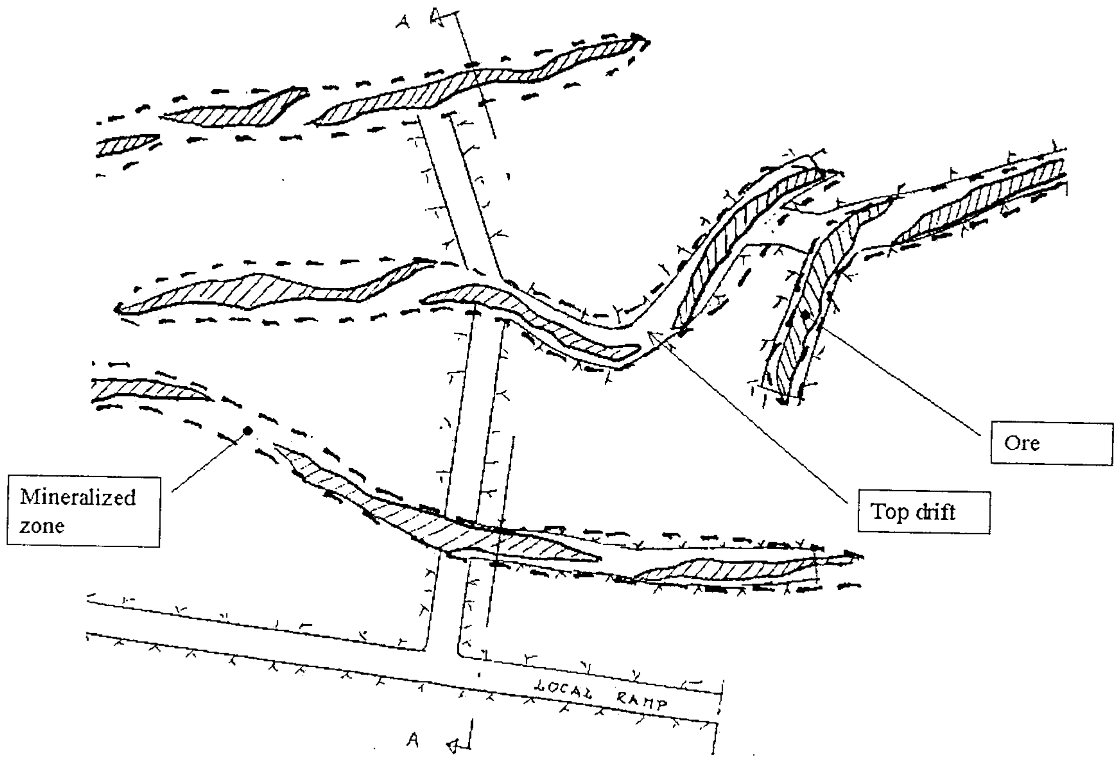
Appendix 1. Simulated ore configuration



Ore simulated from geology descriptions. The same configuration is used for both vertical and horizontal projections. Simulation is only used in order to get a feeling for the consequences for mining with this complexity of the ores.

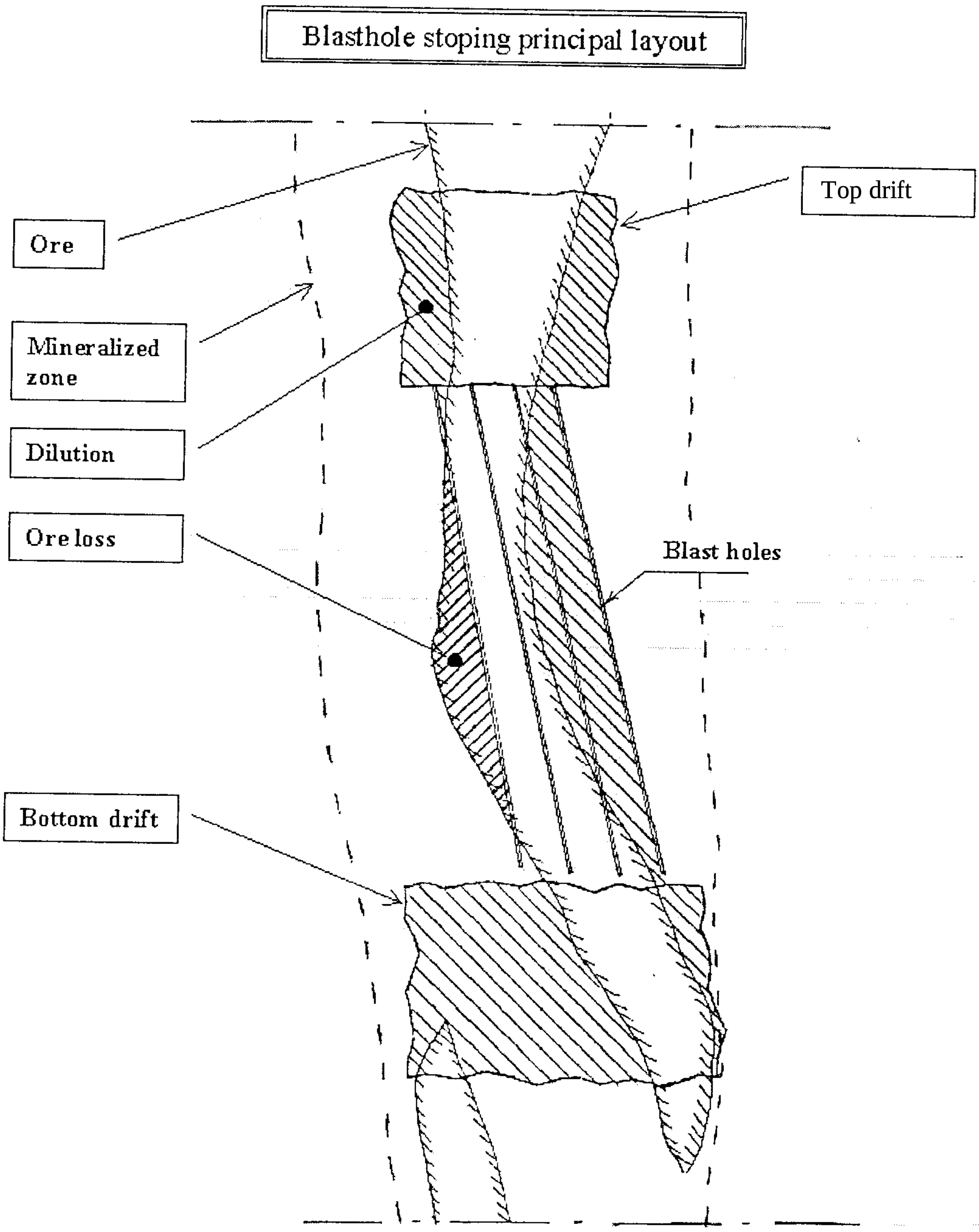
Appendix 2. Access to the ores and development

Simulated horizontal projection with development of the different ore lenses.



Appendix 3. Dilution and ore losses

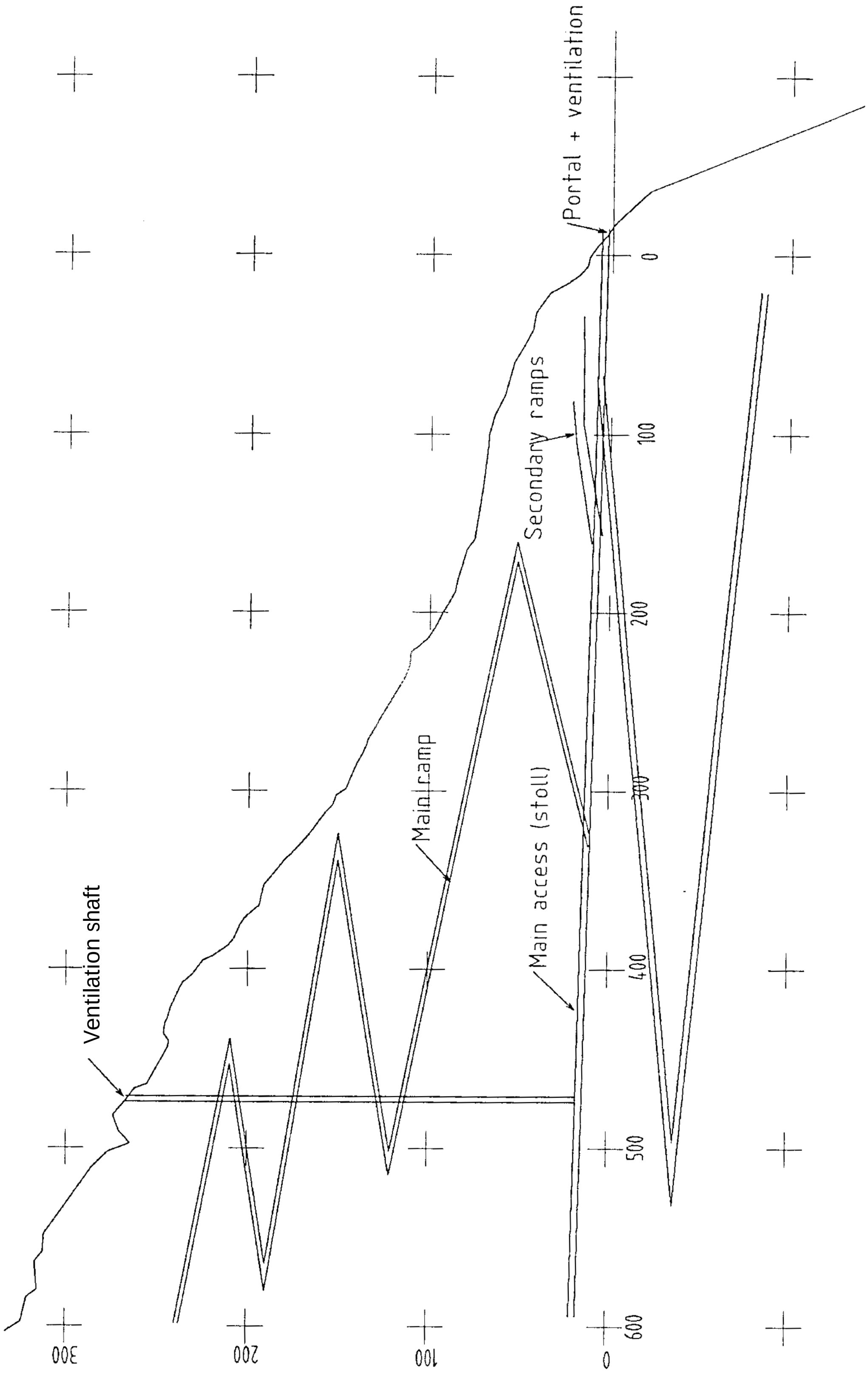
Simulated cross section with blast hole stoping indicating ore losses and dilution.



Section A – A in Appendix 2.

Appendix 4. Development

Main profile along the base line.



Appendix 5. Cash flow estimation US$

MÅLVIKEN UNDERGROUND
SUB-LEVEL STOPING
CASH FLOW ESTIMATION
All numbers in US$,

= estimates

				Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Sum
PRODUCTION PARAMETERS														
Mine	Exploration drilling	m	Investment	3,250	6,000									9,250
	Definition drilling	m	Op cost	0	3,000	8,000	8,000	8,000	8,000	8,000	8,000	4,000	1,000	56,000
	Development	m	Investment	1,700	700									2,400
	Development	m	Op cost	0	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	0	10,400
	Ore production	t		0	125,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	2,525,000
Mill	Mill feed	t		0	125,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	2,525,000
	Head grade	WO3 %	0.72	0	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	**0.72**
	Technical recovery	%	80											
	Kg W produced			0	720,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	**14,544,000**
	Commercial recovery	%	95											
	Payable mtu			0	68,400	164,160	164,160	164,160	164,160	164,160	164,160	164,160	164,160	**1,381,680**
CAPITAL EXPENDITURES			US$/unit											
	Exploration phase 1,	Ea	505,617	505										505
	Exploration drilling	m	67	219	404									624
	Development	m	1,348	2,292	944									3,236
	Powerline, Transf, Pumps	Ea	112,360	112										112
	El, Power	mth	11,236	112										112
	Mill, office, workshop, site prep	Ea	10,112,360	10,112										10,112
	Mine dep, Admin,	mth	16,854	169										169
	Målviken G&A	mth	28,090	281										281
	Subtotal	**1,000 US$**		**13,803**	**1,348**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**15,151**
OPERATING COSTS														
Mine	Development	US$/m	1,236	0	1,607	1,607	1,607	1,607	1,607	1,607	1,607	1,607	0	12,854
	Ore production	US$/t	7.9	0	983	2,360	2,360	2,360	2,360	2,360	2,360	2,360	2,360	19,860
	Definition drilling	US$/m	79	0	236	629	629	629	629	629	629	315	79	4,404
	Mine Dep, Admin	US$/mth	16,854	34	202	202	202	202	202	202	202	202	67	1,719
	Subtotal			**34**	**3,028**	**4,798**	**4,798**	**4,798**	**4,798**	**4,798**	**4,798**	**4,483**	**2,506**	**38,837**
Crushing & FEL Feeding		US$/t	0.79	**0**	**98**	**236**	**236**	**236**	**236**	**236**	**236**	**236**	**236**	**1,986**
Mill		US$/t	5.62	**0**	**702**	**1,685**	**1,685**	**1,685**	**1,685**	**1,685**	**1,685**	**1,685**	**1,685**	**14,185**
Målviken G&A		US$/mth	28,090	**56**	**337**	**337**	**337**	**337**	**337**	**337**	**337**	**337**	**112**	**2,865**
TOTAL OPERATING COSTS		**1,000 US$**		**90**	**4,166**	**7,056**	**7,056**	**7,056**	**7,056**	**7,056**	**7,056**	**6,742**	**4,539**	**57,874**
TOTAL COSTS		1,000 US$		13,893	5,514	7,056	7,056	7,056	7,056	7,056	7,056	6,742	4,539	73,025
REVENUE														
	W prize	US$/mtu	50											
	Exchange rate	US$/NOK	8,90											
	Net smelter return	1,000 US$		0	3,420	8,208	8,208	8,208	8,208	8,208	8,208	8,208	8,208	69,084
	Royalties													
NET REVENUE		**1,000 US$**		**0**	**3,420**	**8,208**	**8,208**	**8,208**	**8,208**	**8,208**	**8,208**	**8,208**	**8,208**	**69,084**
CASH FLOW		1,000 US$		-13,893	-2,094	1,152	1,152	1,152	1,152	1,152	1,152	1,466	3,669	
CUMULATIVE CASH FLOW		1,000 US$		-13,893	-15,987	-14,835	-13,683	-12,531	-11,379	-10,228	-9,076	-7,609	-3,941	
SENSITIVITIES														
CASH FLOW +10% op costs		**1,000 US$**		*-13,902*	*-2,511*	*446*	*446*	*446*	*446*	*446*	*446*	*792*	*3,215*	
CUMULATIVE CASH FLOW		*1,000 US$*		*-13,902*	*-16,412*	*-15,966*	*-15,520*	*-15,074*	*-14,627*	*-14,181*	*-13,735*	*-12,943*	*-9,728*	
CASH FLOW -10% mill feed grade		**1,000 US$**		*-13,893*	*-2,436*	*331*	*331*	*331*	*331*	*331*	*331*	*646*	*2,848*	
CUMULATIVE CASH FLOW		*1,000 US$*		*-13,893*	*-16,329*	*-15,998*	*-15,667*	*-15,336*	*-15,005*	*-14,674*	*-14,343*	*-13,697*	*-10,849*	

Appendix 6. Cash flow estimation NOK

MÅLVIKEN UNDERGROUND
SUB-LEVEL STOPING
CASH FLOW ESTIMATION
All numbers in NOK

= estimates

				Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Sum
PRODUCTION PARAMETERS														
Mine	Exploration drilling	m	Investment	3,250	6,000									9,250
	Definition drilling	m	Op cost	0	3,000	8,000	8,000	8,000	8,000	8,000	8,000	4,000	1,000	56,000
	Development	m	Investment	1,700	700									2,400
	Development	m	Op cost		1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300		10,400
	Ore production	t		0	125,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	2,525,000
Mill	Mill feed	t			125,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	300,000	2,525,000
	Head grade	WO3 %	0.72		0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	0.72	**0.72**
	Technical recovery	%	80											
	Kg W produced			0	720,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	1,728,000	**14,544,000**
	Commercial recovery	%	95											
	Payable mtu			0	68,400	164,160	164,160	164,160	164,160	164,160	164,160	164,160	164,160	**1,381,680**
CAPITAL EXPENDITURES			NOK/unit											
	Exploration phase 1,	Ea	4,500,000	4,500										4,500
	Exploration drilling	m	600	1,950	3,600									5,550
	Development	m	12,000	20,400	8,400									28,800
	Powerline, Transf, Pumps	Ea	1,000,000	1,000										1,000
	El, Power	mth	100,000	1,000										1,000
	Mill, office, workshop, site prep	Ea	90,000,000	90,000										90,000
	Mine dep, Admin,	mth	150,000	1,500										1,500
	Målviken G&A	mth	250,000	2,500										2,500
	Subtotal	1,000 NOK		**122,850**	**12,000**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**134,850**
OPERATING COSTS														
Mine	Development	m	11,000	0	14,300	14,300	14,300	14,300	14,300	14,300	14,300	14,300	0	114,400
	Ore production	t	70	0	8,750	21,000	21,000	21,000	21,000	21,000	21,000	21,000	21,000	176,750
	Definition drilling	m	700	0	2,100	5,600	5,600	5,600	5,600	5,600	5,600	2,800	700	39,200
	Mine Dep, Admin	mth	150,000	300	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	600	15,300
	Subtotal			**300**	**26,950**	**42,700**	**42,700**	**42,700**	**42,700**	**42,700**	**42,700**	**39,900**	**22,300**	**345,650**
Crushing & FEL Feeding		t	7.00	**0**	**875**	**2,100**	**2,100**	**2,100**	**2,100**	**2,100**	**2,100**	**2,100**	**2,100**	**17,675**
Mill		t	50.00	**0**	**6,250**	**15,000**	**15,000**	**15,000**	**15,000**	**15,000**	**15,000**	**15,000**	**15,000**	**126,250**
Målviken G&A		mth	250,000	**500**	**3,000**	**3,000**	**3,000**	**3,000**	**3,000**	**3,000**	**3,000**	**3,000**	**1,000**	**25,500**
TOTAL OPERATING COSTS		**1,000 NOK**		**800**	**37,075**	**62,800**	**62,800**	**62,800**	**62,800**	**62,800**	**62,800**	**60,000**	**40,400**	**515,075**
TOTAL COSTS		**1,000 NOK**		**123,650**	**49,075**	**62,800**	**62,800**	**62,800**	**62,800**	**62,800**	**62,800**	**60,000**	**40,400**	**649,925**
REVENUE														
	W price	US$/mtu	50											
	Exchange rate	US$/NOK	8.90											
	Net smelter return	1,000 NOK		0	30,438	73,051	73,051	73,051	73,051	73,051	73,051	73,051	73,051	**614,848**
	Royalties													
NET REVENUE		**1,000 NOK**		**0**	**30,438**	**73,051**	**73,051**	**73,051**	**73,051**	**73,051**	**73,051**	**73,051**	**73,051**	**614,848**
CASH FLOW		**1,000 NOK**		-123,650	-18,637	10,251	10,251	10,251	10,251	10,251	10,251	13,051	32,651	
CUMULATIVE CASH FLOW		1,000 NOK		-123,650	-142,287	-132,036	-121,785	-111,533	-101,282	-91,031	-80,780	-67,729	-35,077	
SENSITIVITIES														
CASH FLOW +10% op costs		**1,000 NOK**		*-123,730*	*-22,345*	*3,971*	*3,971*	*3,971*	*3,971*	*3,971*	*3,971*	*7,051*	*28,611*	
CUMULATIVE CASH FLOW		*1,000 NOK*		*-123,730*	*-146,075*	*-142,103*	*-138,132*	*-134,161*	*-130,190*	*-126,219*	*-122,247*	*-115,196*	*-86,585*	
CASH FLOW -10% mill feed grade		**1,000 NOK**		*-123,650*	*-21,681*	*2,946*	*2,946*	*2,946*	*2,946*	*2,946*	*2,946*	*5,746*	*21,306*	
CUMULATIVE CASH FLOW		*1,000 NOK*		*-123,650*	*-145,331*	*-142,385*	*-139,439*	*-136,493*	*-133,546*	*-130,600*	*-127,654*	*-121,908*	*-100,602*	