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Tittel

Prefeasibility study of the Skiftesmyr and Godejord copper zinc prosjects, Grong, Central Norway

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Bernard, Louis M	29.09 1997	Braddick, Resources Inc

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Økonomisk		Skiftesmyr Godejord
Råstoffgruppe	**Rastofftype**	
Malm/metall	Cu Zn	

Sammendrag, innholdsfortegnelse eller innholdsbeskrivelse

At todays base metal prices, the project can achieve Capital Payback within the 6.6 year life of the main orebody, the Skiftesmyr, as it requires at about 5,3 years if New Equipment is utilized and 4 year payback if capital equipment for mine/mill is substantially all used equipment. Net profit (before taxes) is Cdn$ 7 million over the remainder of the mine life for used equipment.

Given a 10% increase over todays base metal prices makes the project very profitable and yield a 3,4 year bay back. The profit picture becomes very attractive when used equipent is costed, yielding a 2,8 year payback and a potential Net profit (before taxes) of Cdn$ 17 million over the remainder of the mine life.

A long time base metal price increase of 20% effects a rapid payback of 3 years (new equipment) or 2,3 year for used equipment wich yields av a net profit (before taxes) of Cdn$ 23 million over the remainder of the mine life

The Project can become more profitable by finding more of the Higher Grade ore at Godejord to achieve Capital Payback in tless than One Year and a total Net Profit of $19 million (Cdn) over the remainder of the life of both orebodies at todays base metal prices.

See recomandations.

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Vi oversender herved rapport "PREFEASIBILITY STUDY OF THE SKIFTESMYR and GODEJORD COPPER/ZINC PROJECTS GRONG AREA, NORWAY" utarbeidet for Braddick Resources av L.M. Bernard..

Med vennlig hilsen

Boye Flood

Vedlegg

PREFEASIBILITY STUDY
OF THE
SKIFTESMYR and GODEJORD
COPPER/ZINC PROJECTS
GRONG AREA, NORWAY

PREPARED FOR
BRADDICK RESOURCES LTD
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Master Of Engineering- Metallurgy
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December, 1996

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EXECUTIVE SUMMARY

Braddick Resources Ltd. has acquired base metal properties in the Grong area of Norway which consist of two Copper/Zinc deposits, the Skiftesmyr and the Godejord, whose claims cover 20.4 square kilometers. Previously held by the Finnish mining company, Outokumpu, the in-place ore reserves at the two properties have been estimated at 2,724,000 tonnes by Outokumpu in 1992 and 3,850,000 tonnes by the Geological Survey of Norway.

The present study was performed at the request of Braddick Resources Ltd. for Mining the two orebodies by an underground Longhole method similar to that employed at the Juma Mine, which is in the same type of geological environment . Separate Copper and Zinc concentrates would be produced in a plant with an annual capacity of 360,000tonnes. A comparison of New and Used equipment indicated substantial reductions in Capital Costs with used equipment, lowering the costs from Cdn$13 million to $9.73 million. Operating costs were very reasonable at Cdn$31.00/tonne of ore mined and milled.

The viability of a mine/ mill operation is expected to be very attractive with a 10-20% increase over current base metal prices. Paybacks of 2.3 to 4.2 years may be achieved at various metal prices forecast for the short and longterm. Payback time could be substantially reduced to a year or less if 4-500,000tonnes of ore with a higher Zinc grade was available at the start of production, potentially from the Godejord Orebody.

The study concluded that further Exploration and Development work was justified, with the target to establish more ore reserves of a higher grade at the Godejord, which already has indications of more high grade ore resulting from work done in 1996 by the NGU(Geological Survey Of Norway).

1.0 Introduction

Braddick Resources Limited has acquired two copper/zinc deposits in the Grong district, Norway, covering a claim block(Figure 1) of 20.4 square kilometers. Geologically, the district, which hosts the currently active Cu/Zn Joma Mine in Norway, extends into western Sweden where the Stekenjokk copper/zinc deposit has been mined .The two Grong deposits, known as the Skiftesmyr and the Godejord, have been explored by the Finnish mining company Outokumpu over the past several years and which had determined ore reserves of over 2.7 million tonnes of copper/zinc ore for the two properties as of 1992. It has been recognized that both deposits are open to depth while the Godejord is open also along a strike length of 2.9 km, of which only 900m have been explored, leaving some potential for increasing the ore reserves .

Present plans by Braddick Resources Limited include the consideration of placing the two deposits into production at a rate sufficient to sustain an operation for 8 to 10 years.The intent of this study is to define the Capital and Operating costs of mining the narrow (3m) orebodies by underground mining methods and concentrating the minerals to produce separate Copper and Zinc concentrates in order to determine the Projects economic viability. Consideration of a Pyrite Concentrate for future production has been addressed briefly as the market for this product is uncertain at this time.Cost and logistical data were gathered during a field trip to the Grong area in Norway in October, 1996. The study marries a combination of local Norwegian costs with Capital costs for equipment to be purchased in Canada/USA and shipped into Norway. A conservative approach has been taken when estimating costs and the philosophy of establishing a cost-effective Operation has been used throughout the study when considering both Capital and Operating costs. For this reason, used equipment and modular buildings are used wherever possible. Equipment costs have been estimated to include prefabricated bases, enabling it to be quickly and inexpensively installed on simple concrete pads. Modularization will also enable the mine/mill facilities to be easily dissembled at the end of the mines life, effecting an inexpensive Site Reclamation plan , a requirement that is currently in effect in Norway. While some ideas taken from the Outokumpu mine planning for the Grong deposits have been incorporated in this study, they have been modified to lower both Capital and Operating Costs.

Outokumpu based their plans on their current operation Grong Gruber A/S Mine at Juma , which the author visited and determined that it was a "cadillac" type of operation typical of most Scandinavian mines.The mill size is rather large(~2000 tonnes/day) for the size of operation required at Braddicks Grong property, which has been estimated to be able to support a 1000 tpd operation.

Although the Grong Gruber mine will close in the early part of 1997, the mine/mill equipment will not be available for purchase by Braddick, as Outokumpu have already made arrangements with a "group" to take over the equipment, buildings and mine. For the purpose of this study, costs of the required equipment/buildings will be obtained independent of the Outokumpu operation. For this reason, equipment has been costed from Canadian and U.S. sources with the provision to ship it to Norway, where it can easily be landed at the port of Namsos sufficiently close to the site to minimize overland trucking costs.

2.0 Geology

2.1 Regional Geology and Mining History

The Grong District covers 3000 square kilometers in central Norway, bounded to the East by the Swedish border, to the west by the Namsen River, in the South by the Sandola Valley and to the North by Lake Namsvattnet and the Borgefjell National Park (reference Appendix No.1- Geology of The Skiftesmyr and Godejord Deposits).

"The Grong District is underlain dominantly by Lower Paleozoic metavolcanic, metasedimentary and intrusive rocks of Mid Ordivician age, that comprises the Gjersvik Nappe, part of the larger Koli Nappe of the Upper Allochthon tectanostratigraphy within this part of central Scandinavian Caledonides.These nappe sheets contain thrust emplaced terrains that are far transported slices of volcanic, intrusive and sedimentary rocks of ocean floor, rifted-arc and back-arc marginal basin infill that have been thrust eastward onto the Baltoscandinavian basement(Baltic Shield).Mafic volcanites dominate the island arc-rifted arc complex with felsic volcanites forming only a minor component. The felsic volcanites occur at several stratigraphic levels, often associated with massive sulphide mineralizations that are generally overlain by thin layers of banded iron formation, which regionally can form extensive marker horizons throughout the district.The whole sequence has undergone extensive folding and shearing deformation related to thrusting and Nappe emplacement.The rocks are generally moderate to strongly sheared (well foliated) and have undergone Upper Greenschist, grading into Lower Amphibolite facies metamorphism within the western part of the district."

Volcanic hosted massive sulphide mineralization is common but deposits are generally small, containing less than one million metric tons. There are several major deposits in the Grong Districtthat have been or are being mined ;

JOMA- 20 million tonnes, 1.3%Cu, 1.7%Zn, 32%S, mined since 1972, scheduled to close mid-1997.

SKOROVASS- 10 million tonnes mined since 1952-1984, first mined for Pyrite with 1.1%Cu&39%S, last mined grade of 1.15%Cu, 2.71%Zn.

GJERSVIK- 1.6million tonnes, 25km from Joma, 1.60%Cu, 0.90%Zn. Currently being mined, ore processed at Joma.

The deposits are generally low in Lead content and contain some recoverable Gold and Silver.

The two deposits that are the subject of this study, the Skiftesmyr and Godejord, lie in the southwest corner of the Grong District 20-30 kilometers east of the village of Grong. The Skiftesmyr deposit occurs in the same stratigraphic level as the previously mentioned Skorovass and Gjersvik deposits, overlain by mixed felsic/mafic tuffsand/or volcanoclastics , grading into mafic tuffs and massive/pillowed lavas. The country rocks adjacent the massive sulphide zone are extremely sheared.

2.2 Geology of the Skiftesmyr Deposit

"The orebody at Skiftesmyr consists mainly of Zn-Cu rich massive pyritic ore that occurs as thin layers or as a continuous series of lenses forming a relatively thin , plate-like orebody varying in thickness from 2-20m, averaging 4-6m. The massive ore contains many fragments of country rock near its contact with the host rock, especially within the upper and eastern parts of the orebody. These fragments appear to be remants of fold hinges that have been ripped apart and now occur as loose fragments , floating within the strongly sheared orebody. The massive sulphide layers and lenses are enclosed within a quartz-sericite , albite and chlorite rich schistose country rocks that contain variable quantities of disseminated and veined sulphides, dominantly Pyrite. Minor quantities of Chalcopyrite and Sphalerite are also present within these altered and sheared rocks, Chalcopyrite being mostly confined to the darker chlorite rich rocks and Sphalerite in the pale , quartz-sericite and albite rich rocks.The massive ore is dominantly Pyritic with varying subordinate quantities of Chalcopyrite and Sphalerite and minor amounts of Pyrrhotite. The Silver and Gold mineralogy at Skiftesmyr has, to date, not been studied and the distribution of these precious metals within the the orebody is little known because of the sparse amount of analytical data presently available. The main gangue minerals are quartz, chlorite and calcite. Copper and Zinc are antipathetically related to each other and show a clear zonal distribution within the massive Pyrite orebody, which is typical for most volcanic hosted massive sulphide deposits. The Copper rich ore dominates the eastern and upper levels of the orebody and Zinc rich ore is concentrated in the western part and at depth in the orebody. The deposit is open at depth and towards the west , where the orebody also becomes distinctly thinner." The pyritic ore is a compact, homogeneous type of medium-grain size in the range of 1-5mm. Without the benefit of a mineralogical study, the pyrite grains appear to be granular, with few sulphide inclusions.Chalcopyrite and Sphalerite form grains at the boundaries between the larger Pyrite grains, indicating that there should be little cause for problems in the selective flotation of the copper and zinc minerals frpm each other and the host Pyrite.The Skiftesmyr ore is felt to be very different to the complex fine-grained pyritic ores of the Skorovass and Joma ore deposits and as such is expected to be easier to process by flotation.

2.21 Skiftesmyr Ore Reserves

"The earliest ore reserve calculation for the Skiftesmyr deposit was carried out in 1977 by Grong Gruber A/S and gave a geological ore reserve of 3.5million tonnes grading 1.16% Copper and 1.79%Zinc. Later drilling (1980-92) has not changed this figure to any degree , as much of the drilling was confined to filling in details within the upper levels of the orebody." In 1992, NorsulfidA/S in planning for mining the Skiftesmyr deposit by a combination of open-pit and underground methods, estimated in-place ore reserves at 1 and 2% Copper equivalent cut-off as follows:

@ 1% Cu equivalent cut-off- 2,746,470 tonnes, 1.23%Cu, 1.86%Zn, 11.37ppm Ag, 0.35ppmAu, 37.52%S

Ore zone 400m long, 400m depth, 2-21m thick.

@ 2% Cu equivalent cut-off- 1,759,417 tonnes, 1.38%Cu, 2.13%Zn, 12.99ppm Ag, 0.37ppm Au

Mineable Ore(1% cut-off) - *2,684,000 tonnes, 1.08%Cu, 1.63%Zn, 8.65ppmAg, 0.31ppmAu, 34.6%S .

*Includes dilution and ore left for pillars.

2.3 Geology of the Godejord Deposit

" The Godejord deposit lies 3-4km SSW of the Skiftesmyr deposit in a slightly different geological environment.The rocks at Godejord consist of a complex dominated by mafic volcanites and minor tuffite/sediments that have been strongly deformed and metamorphosed under Lower Amphibolite facies conditions. The whole sequence seems to be inverted at Godejord. Mafic volcanites dominate. The lower part of the sequence is dominated mostly by thick layers of massive flows , dykes and subvolcanic high level doleritic intrusions or sills and the upper part by pillowed flows. These two units are separated by a very persistent Banded Iron Formation/tuffite horizon that forms a prominent marker horizon throughout the district.

At Godejord, this unit forms the Hanging Wall(HW) to the main(East) orebody. This sequence of mixed mafic volcanites/tuffite rocks is distinctly different from those found at Skiftesmyr. Trace element characters of the volcanic rocks are also distinctly different from those at Skiftesmyr and those in the Gjersvik Gp in general". ----------------- "The Godejord ore zone lies at a level in the thick volcanic sequence that is dominated by calc- rich tuffites with iron formations and chert intercalated with mafic volcanites and minor felsic units. The total strike length of the mineralized zone is in excess of 2km. However, the most interesting mineralization is confined to a 500m long zone centered around the main Godejord showing". The thickest part of the orebody appears to plunge steeply to the northeast , which was confirmed by Outokumpu in 1992 by down-hole geophysics on the whole eastern ore zone.

"Semi-massive to massive mineralization is only found in the eastern part of the zone, around the main showing at Godejord. Here the ore zone is closely associated to a magnetite-bearing quartzite lens(recrystallized chert), that for the most part forms the hanging wall to the deposit and locally can reach thicknesses up to 10m. The mineralized zone at Godejord is confined to the footwall of a prominant quartzite(chert) horizon that trends roughly East-West and dips 60-70 degrees to the north. The orebody is thickest and richest around the main showings(called the John Godejord skjerp) and the most interesting mineralization plunges steeply to the NE. Ore mineralization occurs within a zone containing a variety of host rocks ranging from quartz, carbonate, quartz-sericite and actinolite-tremolite rich layers. Pyrite dissemination is most common and quite variable and interlayered with bands of semi-massive to massive pale, honey-colored sphalerite and chalcopyrite-rich disseminations in dark hornblend-actinolite rich layers. The individual layers vary from 10-30cm thick layers of massive Sphalerite. Gold mineralization may be found associated with quartzite lenses and layers within the mineralized zone, as is the case to the west. Gold may have been derived through remobilization from chert/banded iron formation layers that have been tectonically reworked and hydrothermally altered during the period of sulphide deposition.The richest parts of the deposit occurs within a zone up to 60m long and 15m thick, where grades can reach up to several percent Cu, 0.7%Pb, 80ppm Ag, 5ppmAu, 25%Zn, which are atypical for the Godejord sulphide zone.

Another anomalous trait of the Godejord deposit is its large quantities of extremely Iron-poor, pale honey-colored Sphalerite. ----------- From the East Orebody(main showing), the mineralized zone at Godejord continues to the West for about 1.5km and to the East about 1.0km. The thin mineralized zone corresponds to a prominant IP anomally in the heavy overburden-covered terrain. To the west, the mineralized zone has a maximum thickness of 4-5m consisting of strongly altered quartz-sericite-albite rich rock carrying variable quantities of Pyrite and minor Sphalerite that occur as dissemination and veins. The orebody beneath the the quartzite horizon at Godejord is interpreted as being a calc-silicate skarn mineralization, occurring as actinolite-tremolite, quartz, carbonate and quartz-sericite rich rocks that contain variable disseminations to semi-massive to massive mineralization rich in Pyrite-Sphalerite-Chalcopyrite ,minor galena".

2.31 Godejord Ore Reserves
Ore reserves were calculated by Outokumpu(NorsulfidA/S) in their 1992 mining study(Appendix 2) for mining from surface(307m ABSL) 67 meters to the 240m level. **1992 Mineable ore reserves** were calculated at **76,221 tonnes**, grading 0.76%Cu, 7.76% Zn, 24.47ppm Ag, 0.83ppm Au. Dilution was calculated at 16.5% and is included in the mineable reserves.

The Geological Survey of Norway(NGU) performed further ore reserve calculations based on the prior 31 drill holes and an additional 3 new deep holes that intersect the ore zone 250meters below the main showing.The orebody is called the East Orebody and lies beneath the main showing and has a strike length of 500m. The following is the ore reserve estimates ;

@1% Cu-equivalent cut-off : 250-300,000 tonnes over 150m strike length, 200m depth, grading 0.6%Cu, 4.2%Zn, 0.1%Pb,15ppmAg, 0.4ppmAu.

@2% Cu-equivalent cut-off : 100,000 tonnes over 100m strike length , 100-120m depth, grading 0.8% Cu, 6.9%Zn, 0.2%Pb, 20ppm Ag, 0.8ppm Au.

3.0 Infrastructure

3.1 Minesites and Locale

The two Mine Properties are about 5km from each other and are well located in a rural area about 15km East/northeast of the small town of Grong(Figure 2), served by paved roads and within one or two kilometers of a major electrical power transmission line(Figure 3). The primary industry in the area is a combination of farming and lumbering, the later being carried on mainly by lumber companies.The roads into each property are gravel, used mostly for hunting and by the lumber companys, and would require upgrading to sustain the heavy truck traffic anticipated for a mining operation.

Initially, the Skiftesmyr Orebody, being the larger of the two deposits, would be developed into a Mine and Mill operation, requiring electrical power from the main line 1 kilometer away. The Godejord deposit will be mined towards the end of the mine-life at the Skiftesmyr. A new stretch of road about 0.5 km long will be required to allow ore from Godejord to be trucked to the mill at Skiftesmyr located about 5km away. Power for the Godejord operation is located only 1 kilometer away. It is anticipated that the Hydro Company will build all electrical transmission facilities, providing transformers and switch gear, the cost for which will be built into the power charges for the operation , eliminating these items as Capital Costs.The main road will have to be upgraded for about 6 kilometers and the property access roads reconstructed for about 1 kilometer over a surface area that can be swampy in places but is generally rolling terrain supporting stands of birch and pine trees.There appears to be sufficient sand and gravel for use as road material, however, coarse road ballast will be best obtained from rock excavated from the Mine Access Tunnel. Land used for the minesite will be subject to payments to the landowner for damage and loss of trees.

Fresh water is readily available from several nearby lakes and the ground water appears to be another potential source. Given the potential for heavy rainfalls, creeks will have to be diverted around the minesite and tailings area to avoid damage to facilities.Sewage disposal will be by conventional septic system.

3.2 Regional Infrastructure

Concentrate produced from the operation will be trucked in 30 tonne loads to the nearest port, which is Namsos, located about 80 kilometers away(Figure2). Outokumpu presently ship concentrate from their own dockside storage facility at Namsos, using a conveyor to load the ship.It is assumed that this facility will be available from either Outokumpu or the Port Authority, for use by Braddick Resources for shipping the Skiftesmyr concentrates.

Staff and mine/mill operators are expected to be available as the result of the closure of Outokumpu's mine/mill operation at Juma in early1997. Given the time lapse between a possible start-up of operations at Skiftesmyr and the closing of the Juma mine, not all of the personnel required will be available from Juma as some will have relocated to other employment. However, a core of experienced personnel can be assembled and used to train inexperienced personnel which are expected to come mainly from Grong, the closest town to the Skiftesmyr orebody and which is also expected to house workers moving to the area from other parts of Norway. Mine personnel are expected to report to the minesite using their own means of transportation, eliminating the necessity for a Company bus system. Except perhapsfor the mill construction period,there will be no need for a camp at the mine or to provide housing as the town of Grong is expected to have sufficient housing available.

4.0 Mining Plan

4.1 Skiftesmyr Deposit

The mining plan in Outokumpu's Study of April 23,1992 has been used as the basis for mining the Skiftesmyr orebody which was estimated to have an in-place ore reserve of 2,648,000tonnes grading 1.08%Cu, 1.63%Zn, 0.31g/t Au, 8.65g/t Ag, using a 1% Cu equivalent cut-off grade. However, mine development , which in the Outokumpu Study was performed for the complete mine, will in this plan be done in stages to minimize the start-up Capital requirements and disperse some of the development costs into the Operations budget. In order to accomplish a staged mine development, the following sequence of work is anticipated:

1. Underground Exploration

Access to the mineralized zone will be by way of a 650 meter long tunnel, 5m. wide x 4m. high, commencing at the 160meter level(above sea level) and declining at a slope of 25%(14 degrees) to Sea Level elevation. This places the major part of the orebody above the Decline, requiring a Ramp to access the ore zones.Exploration of the ore zones will take place via two drifts(sub-levels) placed in the mineralization and following the strike of it, allowing the rock/ore structures to be geologically mapped and further diamond drilling of the ore zone to depth. An ore sample can also be taken for Metallurgical testing.

A single ventillation raise would also be required during this phase, with all the work being performed by a contractor which eliminates the need to purchase equipment.

Costs for this phase is expected to be financed as Exploration and is separate from the costs presented in this study. However, as a rough estimate, based on Outokumpu's 1992 cost estimates(escalated to 1996 basis) for 650 meters of tunnel, 200 meters of sub-level drifts and 250 meters of ventillation raise , the cost could be upward of $2 million (Canadian) exclusive of Diamond Drilling and bulk sampling.It may be possible that costs will be in the order of $3 million once the exploration requirements are taken into account.

2. Mine Development

Upon completion of the Exploration Phase, mining can commence with minimal development, ie. sufficient to enable the development work to keep ahead of production.Preproduction work will consist of the installation of all mine facilities,eg.ventillation fans, dewatering pumps, jaw crusher, tunnel conveyor belt, dump pockets, extension of sub-levels, power lines etc. It should be noted that heating of the ventillation air has not been included on the basis of the Juma mine which stopped heating their mine air some time ago.

Mining of the ore will be by long-hole blasting from sub-level drifts put into the ore zones and drifting along the mineralized zone. In order to supply the mill with 360,000 tonnes of ore per year, the mine must produce 1200 tonnes per day on two 8 hour shifts during a 6 day week.. Assuming an average ore width of 3 meters, a single blast of an ore block 30 meters high, 3 meters wide and 5 meters deep will provide the required tonnage, based on an ore weight (in place) of 4 tonnes per cubic meter(high s.g. due to +60% sulphide content). Since the ore body is steeply dipping(plunging~70 degrees), it is planned to blast the ore down to the lower level where a 7 cubic yard LHD Loader will load it onto a 30 tonne Underground Truck for transporting to the Jaw Crusher, located underground near the Access Tunnel. A second truck with a capacity of 15 tonnes and using two"containers" will handle development ore and waste produced from Ramp and Sub-level Drift installation. A second LHD with a 2.5 cubic yard bucket will work on Ramp and Sub-level Drift installation, loading into the Container Truck. Production drilling will be by an electric-hydraulic Self-propelled Long-hole Drill designed to work in a 3 meter high drift and capable of drilling 89mm(3.5 inch) holes to a 250 meter depth. Drift drilling will use a conventional two-boom mobile electric-hydraulic Drill Jumbo. Blasting for production and development shall use ANFO loaded via a pneumatic Pot carried to the location by one of the LHD Loaders.

Ore shall be trucked to a Jaw Crusher having a capacity of 75 tonnes per hour at a closed-side setting of 51mm(2 inches). A Feeder Grizzly shall feed the ore to the crusher. Discharge from the crusher will be conveyed to a Haulage Conveyor located in the 650 meter long Access Tunnel , by which the ore will be brought to surface and into the Mill, feeding directly into the Cone Crusher. The choice of a haulage conveyor eliminates the need for several 30 tonne

trucks and the additional ventillation required to vent the fumes from the haulage trucks.Table 2 lists the mine equipment .
Mine life of the Skiftesmyr has been based on a recovery of 90% of the stated in-place ore reserves of 2,648,000 tonnes(cut-off 1% Cu equivalent), or 2,383,000 mineable tonnes, yielding a mine life of 6.6years.

4.2 Godejord Deposit

The Godejord Deposit is located about 5 km away from the Skiftesmyr mine property and according to Outokumpu's report filed with the Department of Mines, contains 76,221 tonnes of ore grading 0.76%Cu, 7.76%Zn, 0.83g/tAu, 24.47g/tAg. Although the Zinc grade is higher than the Skiftesmyr, the low tonnage cannot support it's development until the Skiftesmyr is nearing depletion of it's ore reserves. At that time, equipment can be diverted from the Skiftesmyr to the Godejord to commence mine development and production.Access to the mineralised zone is much simpler than for the Skiftesmyr, requiring only a short drift of about 100 meters to reach the ore. Mining will be by sub-level long-hole mining as for the Skiftesmyr, but with the sub-level drifts 20 meters apart. Ore blasted from each level will be loaded onto trucks for transport to the Skiftesmyr mill. There, facilities will be in place to allow truck dumping into a small bin providing feed to the Jaw Crusher relocated from underground at the Skiftesmyr to the surface adjacent the mill cone crusher.

Although the zinc grade of the Godejord is almost 5 times higher than the Skiftesmyr, the metal price for zinc at the time when mining the Godejord is being planned will determine if it will be viable. The small size of the orebody may also dictate if it can support the cost of mine development.For this Study, ore from the Godejord has not been contemplated when considering the economics. However, the ore value is compared to Skiftesmyr when considering revenue.

5.0 Ore Processing

5.1 Metallurgy

5.11 Metallurgical Testing

Ore samples ranging in size from 7 kilograms to 35 kilograms were taken from surface outcrops at both the Skiftesmyr and the Godejord in October 1996 and shipped to Lakefield Research near Peterborough, Ontario, for preliminary metallurgical testing.Two samples of each ore were taken, ie. **Godejord**: (i)sphalerite rich massive to semi-massive(15%Zn,1%Cu) and (ii)semi-massive with higher Copper content(19%Zn,3.2%Cu), **Skiftesmyr**: (i)copper-rich disseminated, semi-massive(1.6%Cu,0.35%Zn) and (ii)massive pyrite ore(0.97%Cu,1.9%Zn) .Sample weights/description and multi-element ICP Analyses are presented in Appendix 3 .

The tests were performed in order to gain a better perspective of the metallurgy(results in Appendix 3), but were not intended to be used for Plant Design as the samples were not representative having been taken at the surface where they could be adversely affected by exposure to air and water which in turn could affect the metallurgical results. In general, the tests, which are summarized below, were successful in outlining the parameters for producing separate Copper and Zinc concentrates, indicating the following trends:

Godejord:

Ore type(i); (sample GO-1) A very high grade of Zinc concentrate (+60%) from this type of Godejord ore(1%Cu, 15%Zn) can be anticipated at high(88%) Zinc recoveries while the copper concentrate will contain 20% copper,13% zinc,5% lead with an 84% copper recovery. Lead and zinc contents in the copper concentrate may be too high and will have to be lowered through differential flotation, which has yet to be tested and will probably be best performed on a more representative ore sample.Gold and silver recoveries into the copper conc were 58 and 70% respectively at grades of 23 and 83g/tonne. No difficulties are anticipated in achieving acceptable grades and recoveries in actual operations. Pyrite concentrate produced from this ore type appears to be very clean, containing about 0.3% copper, 0.1%zinc, 0.04%lead. About 15 tonnes of pyrite can be produced for every 100 tonnes of ore processed.This is important for environmental reasons, allowing Pyrite to be removed from the ore and sold as a product, thus eliminating potential acidification of tailings in the storage area.

Ore type(ii); (sampleGO-2) Given the higher zinc and copper grades of this ore type(3%Cu, 19%Zn) which is not expected to be typical of the average ore, minimal testing was performed. The results indicated good metal recoveries of 83% of the copper into a concentrate analysing 23%Cu , 10%Zn,1.7%Pb. Zinc recovery was 85% into a concentrate assaying 52%Zn, 0.5%Cu, 0.16%Pb.

Skiftesmyr:

Ore type(i); (sample SKM-1) From a head grade of 1.6% Copper, a good grade copper(23% to 27%) concentrate can be obtained at recoveries of 84to 89% containing less than 1% zinc and lead. Zinc recovery/grade were not satisfactory due to the very low(below 1% cut-off)zinc headgrade of 0.35% which would ordinarily not be subject to a Zinc recovery step.

Ore type(ii); (sample SKM-2)This ore sample more accurately resembled the "average ore", having a grade of 1%Cu, 2%Zn. A copper recovery of 87% into a concentrate grading 23%Cu and 5%Zn appears achievable and is expected to be improved upon in a large scale operation. Zinc concentrate grade was 53% with a recovery of 77%, the main contaminant being pyrite which was easily floated, indicating pre-activation due to copper solubilization from oxidation.

SUMMARY OF METALLURGICAL TESTING(Best Results)

	ORE SAMPLE			
	GODEJORD		SKIFTESMYR	
PRODUCT	**GO-1**	**GO-2**	**SKM-1**	**SKM-2**
HEADS,%	1.0Cu,15.7Zn	3.2Cu,19.3Zn	1.6Cu,0.35Zn	0.97Cu,1.9Zn
Cu Conc.	(test F3)	(test F8)	(test F4)	(test F7)
Grade%; Cu	20.1	22.8	23.4	22.4
Zn	13.4	10.1	0.3	4.8
g/t Au	23.2	-	-	(5.35)
g/t Ag	835.0	-	-	(93.0)
%Recovery; Cu	84.1	87.3	87.4	83.8
Zn	4.2	7.2	6.17	8.5
Tons conc/1000t. ore	46.0	134.0	57.8	36.0
Zinc Conc.				
Grade%; Zn	62.8	52.3	0.85	53.8
Cu	0.16	0.54	0.24	1.47
%Recovery; Zn	88.1	85.8	76.6	77.7
Cu	3.0	4.8	4.11	4.5
Tons conc/1000t.	205.0	307.0	no conc.	29.8
Pyrite Conc.	356 t./1000 t. ore 0.26Cu, 2.29Zn	0	196 t./1000t ore, <0.2Cu/Zn/Pb	0

5.12 Metallurgical Process

The ore, consisting mainly of Pyrite, Chalcopyrite and Sphalerite, will be wet ground to liberation size and subjected to differential flotation under conditions similar to those used in the testing by Lakefield Research.The flowsheet in Figure 4 outlines the steps taken to achieve separate Zinc and Copper Concentrates. Through the use of ph control with lime (calcium hydroxide) and the possible use of other pyrite depressants such as Sodium Sulphite or Zinc/Cyanide complex, both Pyrite and Sphalerite will initially be prevented from floating, allowing the copper mineral Chalcopyrite, to be concentrated using specific selective collectors for Chalcopyrite. Stabilization of the air bubbles will be accomplished by mixing small amounts of Frother such as Methyl Isbutyl Carbinol into the pulp prior to subjecting it to Flotation.The flotation will take place in two stages at a pulp density of 35% solids; the "rougher" stage where most of the copper mineral will be recovered, and the "scavenger" stage that will ensure the recovery of all the flotable copper mineral. Both concentrates will be "cleaned" together in a single flotation stage to produce the final copper grade of a sulphide concentrate containing at least 25% copper.

Following the copper float, the zinc mineral Sphalerite will be "activated" with copper sulphate and conditioned with frother and Collector prior to floating . Once again, a "rougher" and "scavenger" flotation will produce a concentrate that will be subjected to "cleaning" in a separate flotation circuit to produce a zinc concentrate containing at least 53% zinc.The possibility of requiring a Pyrite depressant, such as sodium sulphite, has been demonstrated with regard to achieving both copper and zinc grade in the final concentrates.

On the understanding that the sulphide minerals are relatively coarse , no provision has been made to regrind either of the concentrates. Should future testing demonstrate the requirement that either the copper concentrate or the zinc concentrate require regrinding and further flotation to achieve the desired metal grade, a small regrind mill and recleaner circuit will be included in the mill design.

A Pyrite concentrate has not been planned at this early stage as the market for this product is speculative and depends on several developments in the near future, however tests have been performed in order to assess the potential grade of the product.

The need for Pyrite rests with two real possibilities:

1. The Borregaard Sulphuric Acid Plant in Sarpsborg near Oslo requiring 250,000 tonnes per year in the near future.
2. The new development in Norway using Pyrite directly in Agricultural applications requiring a source of sulphur for soil modification.

5.2 Mill Equipment

The mill equipment as listed in Table3 is conventional and has been sourced from both new and used equipment suppliers in Canada and the United States. A South African supplier was also considered as the exchange rate is presently favourable.

Sizing of equipment allowed for small future increases in production rates where possible. For instance, the Cone Crusher was sized with a 150 HP motor to yield a 110 tonne/hour capacity. By installing a larger motor and a different "bowl", through-put capacity can be increased by 10-15%. There are some pieces of equipment that cannot tolerate increased through-put without performance being adversely affected, such as Flotation Cells as they are directly related to volume flow in a certain period of time. An increased flow would decrease flotation time and result in lower metal recoveries.

The Cone Crusher, fed with 50mm ore size from the underground Jaw crusher, will easily crush the daily mill feed requirements of 1030 tonnes per day in one and a half shifts, approximately the time that the Mine is operating their Jaw crusher. Should more crushing be needed, the third shift would then be available. The cone crusher product is planned to be 12mm(0.5inch) in size and will be stored in a bin with 2000 tonnes in capacity from which it will be drawn by a feeder conveyor at the required feed rate controlled by a belt scale, feeding into the 3m.diameter x 4.25m.long Ball Mill.

Grinding in the ball mill with 75mm (3inch) steel balls will provide the proper feed size to the flotation circuit. Classification of the Ball Mill discharge will be by a single 500mm (20 inch) hydrocyclone capable of processing a 250% circulating load using a 5 x 5 SRL Pump.The mill has been selected almost exactly for it's size and horsepower, leaving little room for increased through-put, unless the feed size is decreased from 12mm to 9mm, in which case the mill capacity could possibly be increased by 10% using the same size motor. Pulp from the Cyclone (overflow) will flow by gravity to a conditioning tank where reagents will be added at controlled rates using Clarkson Feeders.

The Rougher and Scavenger flotation will require 56 Denver DR100 cells(each 2.8 cu.m) for both the Copper and Zinc flotation stages.The upgrading of the concentrates by refloating in "cleaner" cells will use 14 of the smaller Denver DR18SP cells.Internal pumping of the various flotation products will require 12 -1.5 x 1.25 SRL Pumps. Final tailings will be pumped to the tailing disposal area by a 5 x 5 SRL Pump. The final concentrate products will be pumped into separate 7m diameter thickeners for each of the Copper and Zinc concentrates from which they will be pumped at 65% solids to the separate Drum Filters each having a filter area of 14 square meters(157sq.ft.). It is anticipated that the Copper Concentrate will reach the required moisture on the filter. However, the Zinc Concentrate will require further dewatering using a rotary kiln-type dryer. Both the Copper and Zinc concentrates will be conveyed to holding bins in the concentrate truck-loading bay.There the individual products will be loaded onto highway transport trucks and transported in 30 tonne loads to the seaport of Namsos some 80km away.

5.3 Mill Site Facilities and Tailing Disposal

The structure housing the crushing, grinding , flotation and dewatering sections is unique and has been chosen for it's ease in both assembling and disassembling , requiring only 60 days to erect without the need for large foundations. The **pre-fabricated** structure consists of Aluminum structures supporting a PVC Membrane that has been designed to include 8 inches of insulation and withstand winds of 125-130 miles per hour. Snow load problems are minimal as the Membrane is very slippery, with the structure sloping at 26 degrees off horizontal. The structure has a completely unsupported span of 40 meters(130 feet) and covers a length of 87 meters(285 feet). Built in Canada and used in arctic conditions, this type of structure provides a less expensive building than a conventional steel column and I-beam structure. The main drawback is the fact that the frame cannot be used to support a Bridge Crane commonly found in conventional mill buildings. In this instance, it is anticipated that once the heavy equipment is in place using hired mobile cranes, motors or liners etc. can be easily moved by smaller mobile cranes hired for the occasional heavy lift. Otherwise, winches and come-alongs will be used by maintenance crews in their regular work. to move heavy parts. Piping will be supported from the floor where required and power cables will be be accessed from floor trenches . Motor control centers will be located adjacent the equipment they serve and shall be fully enclosed. There will be 6 man-doors for

There will be 6 man-doors for worker access and 6 roll-up(insulated) bay-doors for equipment access. Similar structures will be used for each of the Warehouse and Maintenance Shop facilities.A simple concrete pad is all that is required for these buildings.

Tailing disposal will take place adjacent the Skiftesmyr Mine , using two small lakes designated by Outokumpu in their 1992 study as being suitable.
It is planned that the first lake will be dammed to provide sufficient capacity for the first half of the mines life. The diversion of a small creek around the lake will be necessary . Should a market develop for the Pyrite contained in the tailings(up to 60%), the volume required may decrease by 20 to 30%,or more depending on the amount of pyrite recovered. Conceivably, if the potential markets mentioned in section 5.1, are realized, almost all of the pyrite content of just over 200,000 tonnes per year could be removed from the tailings, leaving about 100,000 tonnes per year of tailings to be impounded. This would greatly reduce the volume which in turn would mean that the initial site chosen for tailing disposal would last the several years of the mine's life.It would also be desirable from an environmental viewpoint, lowering the costs for site Reclamation at the end of the mine's life.

6.0 Environmental Planning

Present environmental regulations in Norway appear similar to those in Canada, with limits on water/air quality and the requirement to provide a reclamation plan for the mine and mill site. There is no requirement for a Bond to be posted, but a company is expected to have the funds available at the end of the mines life to effect the proper reclamation.
The closure of Outokumpu's Juma mine in early 1997 will cost about 18 million kroner, equivalent to about Cdn$3.6 million.The Closure Plan was required one year before the mine closure. There are three areas at Juma to clean up; plant , mine , tailing area. All foundations must be removed down to the surface level and the area covered with topsoil and grass. The concentrate loading area must have the soil removed and replaced with clean, uncontaminated soil. The contaminated soil will have to be moved back into the mine. Anything that is susceptible to creating Acid Mine Drainage has to go back into the mine . Waste rock with up to 5% pyrite is acceptable.

This means that the Skiftesmyr Mill Tailings containing 60% pyrite would have to be replaced back in the mine.Thus the incentive to remove the pyrite is very

Markets for the pyrite will have to be developed early in the project, particularly with the Boregaard Sulphuric Acid Plant near Oslo which currently receives Pyrite from Finland and is considering a switch to Elemental Sulphur as their source of sulphur when the Finnish mine closes in the year 2000.
Should the Skiftesmyr Mine approach them very early, they may well be persuaded to continue using Pyrite from the Skiftesmyr Mine, which could add some additional revenue to the project as well as minimizing the environmental costs upon closure of the mine.

7.0 Cost Basis

7.1 Basic Criteria and Assumptions

This study has assumed certain parameters which may change with further development of the Project .**Capital costs** were a combination of actual Supplier budget prices for equipment and an estimate of the cost required to install equipment.Equipment suppliers were mainly Canadian, necessitating the estimating of shipping costs to Norway. Where possible, used equipment was costed on a "reconditioned" basis.A saving of 30 to 40% can normally be expected when using second-hand, reconditioned equipment. **Operating costs** were estimated based on experience for both the mine and the mill , using Norwegian costs for salaries and supplies.Established mining methods used by Outokumpu on a similar type of ore was used to cost mining equipment and operating costs. **Metal production** in concentrate was based on estimates using prior experience from other operations factored to reflect the laboratory testing. Metallurgical testing was performed on recently taken surface samples but the results were only used as a guide to estimate contaminant metals in concentrates and the general reaction of the ore to flotation.**Net Smelter returns** were a combination of actual treatment charges/refining charges for the Copper concentrate while a percentage of the concentrate value as Net Smelter revenue was estimated for the Zinc concentrate, all based on Outokumpu input.

7.2 Capital Cost Estimate

A summary of the Capital Costs using essentially all new equipment is presented in Table 1 which,for the total Project, amounted to Cdn$13,015,000.

7.21 Mine Capital

The total capital costs for the Mine were Cdn**$3,577,000**, using almost all new equipment costs.Excluding the single piece of used equipment (table2), a saving of 30% for all **used /reconditioned** mine equipment would reduce the costs by Cdn$1,037,000 to yield a cost of **$2,540,000.**

7.22 Mill Capital

The equipment costs for the Mill totalled Cdn$2,302,000(Table3), of which about 70% was priced on the basis of new equipment. If all of the new equipment were available as "**used/rebuilt**", a saving of Cdn$ 500,000 would apply, assuming a 30% discount off the "new" price. The Mill equipment capital cost would then be about **$1,800,000 .**

The Mill Building (table 4) is priced on a "new" basis, costing Cdn$2,221,000 . If this structure was available as a "**used**" facility, a further saving of perhaps $600,000 may be achieved.However, the realty is that mill buildings are rarely available alone, and the disassembling of one and re-erecting it at a far away site is not cost effective.

However, there are available two used Mills, complete with buildings situated in the U.S.A.and priced (on an **as-is / where-is** basis) as follows :
850 TPD Copper/zinc Plant -US$950,000 (Cdn$1,275,000)
1000TPD Copper Concentrator - US$1,250,000 (Cdn$1,675,000)
Assuming that the larger plant (1000tpd) is acceptable(see Appendices) and can be upgraded to the required tonnage through-put of 1030 tonnes/day(1135 s.tons) without much additional cost, the following cost comparison can be made:

Canadian Mill(per this study)**Cost**	**:Equipment**(new)	= **Cdn$2,302,000**
	Building(new)	= **Cdn$2,221,000**
	Total Cost	= **Cdn$4,523,000**
U.S. Copper Concentrator Cost(used 1970 vintage)		= **US$1,250,000**
	Disassembly/moving	= **US$ 500,000**
	Total Cost	= **US$1,750,000**
	Canadian $ Cost	=**Cdn$2,350,000**
	(Allow)Cost to Upgrade to 1030tonnes/day	=**Cdn$ 173,000**
	Grand Total(approx.)	=**Cdn$2,523,000**

SAVINGS USING U.S. PLANT , MOVED&UPGRADED=Cdn$2,000,000
NOTE: This assumes that there are no further costs re: reclamation at the U.S. site as a condition of sale. It should also be noted that there is a zinc

It should also be noted that there is a zinc concentrator in Europe for sale(no price available) which may be of interest if the project proceeds in the future.

7.23 General Support/Services

The remainder of the Capital Cost estimate (after Mining&Milling) amounts to Cdn$7,136,000 as summarized in Table 1 and detailed in Tables 4&5 for Buildings, Services and Systems.

The bulk of these costs can be attributed to Buildings(Cdn$2,716,000) and Services (Cdn$1,270,000) which among other things, includes site vehicles.For **used equipment**, the building cost would be decreased by $2,220,000 to $495,000 to reflect the elimination of the new prefabricated Mill Building if the used U.S.Copper Concentrator is purchased. Since the "used Mill" comes with it's own drawings for equipment installation, a reduction of the Engineering Costs can also be expected, say 50%, from $500,000 to $250,000.

The remainder of the costs of Cdn$4,170,000 are estimates of peripheral costs surrounding most projects. At this point, there is no basis for lowering these costs until a detailed Feasibility Study is performed that would define the actual costs associated with the various categories, such as Electrical, Piping, Instrumentation, etc.

Capital Cost Summary (Used equipment basis):

Used Mining Equipment	**= Cdn$ 2,540,000**
Used Mill Equipment & Building	**= Cdn$ 2,520,000**
Buildings/Services	**= Cdn$ 495,000**
Services/Systems/peripheral costs	**= Cdn$ 4,170,000**
TOTAL CAPITAL(used) COST	**= Cdn$ 9,725,000**

7.3 Operating Costs

Operating costs were based on actual Norwegian costs in the mining industry today and an estimate of the Supplies and Consumable items associated with a typical mine/mill operation. Prior experience and the Outokumpu mine/mill operation at Juma were used as models for planning the operational structure.

A total of 100 personnel will be required to run the mine/mill operation, costing Cdn$6,350,000 annually in wages/benefits alone, with supplies and consumables estimated at an additional $3,748,000 to yield a total cost of $10,098,000 per year or a unit cost of Cdn$28.21 per tonne of ore.

As a part of the Operating costs, freight charges for moving the separate Copper and Zinc concentrates to the respective smelters are estimated to total Cdn$1,034,000 annually which includes trucking to the port of Namsos and sea freight to Finland and Oda, Norway. The unit cost per tonne of ore is estimated at Cdn$2.87. Total operating costs are then Cdn$31.08 per tonne ore.
Table 6 summarizes the unit operating costs according to the main areas of Mine, Mill, Administration. Tables 7&8 detail the various annual costs for the mine/mill areas.

8.0 Revenue and Payback

8.1 Skiftesmyr Ore

Net smelter returns were estimated for concentrates from the Skiftesmyr Ore using information received from Outokumpu which specified terms for smelting/refining the Copper Concentrate and an estimate from Outokumpu's Juma mine of the net revenues received from the sale of their Zinc Concentrate. These estimates are expected to be conservative and will vary depending upon various penalities for unwanted impurities in the concentrates. The terms apply to 1997 contracts.Tables 10&11 outline the Net Smelter Return calculations for various metal prices but do not address penalties for impurities which are unknown at this point.

Three "scenarios" were investigated using three Metal Price levels ;

1. "current prices"; copper=us$1.00/lb, zinc=us$0.50/lb, gold=us$380/oz, silver= us$5.00/oz
2. "possible future prices" ; copper=us$1.10/lb, zinc=us$0.55/lb, gold=us$400/oz , silver= us$5.00/oz
3. "longterm future prices"; copper=us$1.15/lb, zinc=us$0.60/lb, gold=us$400/oz, silver =us$5.00/oz

The results, summarised in Table 9, showed that current metal prices would yield a Net Profit(before taxes and interest on capital) of Cdn$2,926,876/year and a payback of **5.3 years** . Payback decreases to **4.2 years** on the basis of **Used Equipment**.

An increase of about 10% in base metal prices was sufficient to increase the Net Profit to Cdn$4,401,300/year and lower the payback time to **3.5 years(new equipment basis)** or **2.8 years for Used Equipment**. The metal price increase is regarded as a possibility since there are indications that the zinc price is moving up and there are good possibilities that copper could regain some of it's losses in the next year or so, as LME warehouse stocks have been on the decline.
Over the longterm, an increase in base metal prices of 20% over todays prices may occur which would boost Net Profits to Cdn$5.35 million annually and yield a capital payback of about 3 years or **2.3 years for Used Equipment**.
Any large increases in future Precious Metals prices are regarded as speculative, hence a longterm price ceiling of US$400/oz reflects a conservative approach with regard to Gold prices while silver price was held at the current price of us$5.00.Precious metals in the ore only contribute about 10% of the revenue from metal sales.

8.2 Godejord Ore

As a comparison, if there were sufficient ore with the grade of Godejord and it was mined and milled on the same basis as Skiftesmyr, a quick calculation using todays metal prices, indicates a revenue of about Cdn$32 million per year and a net profit after operating costs (before taxes)of about Cdn$ 21 million. Payback could possibly be realized in less than one year(9 months) using essentially all new equipment.

9.0 Conclusions and Recommendations

9.1 Conclusions

At **todays base metal prices**, the project **can achieve Capital Payback** within the **6.6 year life** of the main orebody, the Skiftesmyr, as it requires at about **5.3 years if New Equipment** is utilized and **4 years payback** if capital equipment for mine/mill is substantially all **used equipment.Net Profit(before taxes) is Cdn$7 Million** over the remainder of the mine life for used equipment.

Given a 10% increase over todays base metal prices makes the project very profitable and yields a **3.5 year payback**. The profit picture becomes very attractive when **used equipment** is costed, yielding a **2.8 year payback** and a potential **Net Profit(before taxes) of Cdn$17 Million** over the remainder of the mine life.

A longterm base metal price increase of 20% effects a rapid payback of **3 years** (new equipment) or **2.3 years** for used equipment which yields a **Net Profit (before taxes) of Cdn$23 Million** over the remainder of the mine life.

The Project can become more profitable by finding more of the Higher Grade ore at Godejord to achieve Capital Payback in less than One Year and a total Net Profit of $19 million (Cdn) over the remainder of the life of both orebodies at todays base metal prices.

9.2 Recomendations

Skiftesmyr Orebody- Exploration should be carried out from Underground to determine the structure of the mineralised zone and whether the ore grade (or tonnage) can be increased . However, achieving a higher production rate from such a narrow orebody is questionable as the mining costs will escalate sharply. Hence more ore of the same grade is not a solution to achieving a higher profit.

Godejord Orebody - The higher **Zinc and Gold grade** make this orebody a prime target for an exploration programme aimed at finding more ore of the same grade. **A target of at least 0.5 Million Tonnes is desirable from a mining viewpoint and there appears to be potential for more ore at depth in the Main or East Orebody, and to the west along strike. Should this target be confirmed, it is recommended that the Godejord Orebody be developed simultaneously with the Skiftesmyr Orebody.**

TABLE 1
GRONG PROJECT
SUMMARY CAPITAL COST ESTIMATE

	Cdn.$ COST
MINING and MINE SUPPORT EQUIPMENT	3,577,000
MILLING and MILL SUPPORT EQUIPMENT	2,302,000
BUILDINGS/SERVICES MILL,SHOP,WAREHOUSE,LAB,OFFICE	2,716 ,000
SERVICES/SYSTEMS POTABLE WATER, ROADS, TAILING DAM,POWER.	955,000
ELECTRICAL: Mine / mill ; wiring & controls	400,000
INSTRUMENTATION	200,000
PROCESS PIPING	100,000
SITE VEHICLES	315,000
Engineering/Procurement/Construction Management	500,000
CONSTRUCTION(Indirect costs)	400,000
SPARES and INVENTORY	150,000
FREIGHT and CUSTOMS(10% of equipment + bldg.)	800,000
CONTINGENCY (10% of equipment)	600,000
TOTAL	$ 13,015,000
OWNERS COST	$_______?

TABLE 2

GRONG COPPERZINC PROJECT

MINE CAPITAL / EQUIPMENT COSTS

ITEM	No.	SIZE	Motor HP / KW	Cdn.$ COST new/ used
ORE BIN	1	100 tonne, 5m.longx3m.wide x3m.ht.	/0	9,000 /0
GRIZZLY FEEDER	1	1X4m	40/30	22,000/0
Hyd. Rock Hammer	1	16ft. boom/S-26Rammer	/28	67,000/0
JAW CRUSHER	1	24X36	100/75	75,000/0
CONVEYOR	1	650m long	156/117	650,000/0
COMPRESSOR	2	200 cfm	2 x 50/75	32,000/0
SCOOP TRAM	1 of ea	2.5 & 7 yd.(remote control)	0	710,000/0
u/g TRUCK(s)	1of ea.	1x30 t.&1x 15 t(container)	0	905,000/0
DRILL(mobile)	1	Longhole,(elec/hyd)	30/22.5	250,000/0
DRILL JUMBO	1	2-Boom (elec./hyd)	50/37.5	525,000/0
MAIN VENT FAN	2	3-4000cfm	2x100/150	36,000/0
U/G Vent Fans	6	200 cfm	6x10/45	24,000/0
U/G Vent Tube	10m lengths	600&800mmFlex(1500m) c/w coupling		30,000/0
Dewatering Pump	2	Gorman Rupp S-4J1	2 x 60/90	30,000/0
U/G Maint. Equipt.	?	various tools(allowance)		30,000/0
Survey Equipment	1	total station		15,000/0
Cap Lamp Charger	1	20 Lamps , 6volts	/0.2	2,000/0
Cap Lamps	20	normal		5,000/0
PIPE	n/a	750 m. of 4in steel c/w couplings		20,000/0
ANFO Hole Loader	1	Mobile,using Scoop Tram, 500 lb. capacity		10,000/0
Loader(surface)	1	Model 60Kawasaki,2Yd Bucket		0/120,000
u/g Transformer	2	500kva(to serve elct/hyd.drills)		10,000/0
TOTAL			**670 Kw**	**$3,577,000**

Mining Rate = 1200 tonnes/day ore

NOTE: All Electrics to be 50 HZ for European Use

* Steel @$12.50/sq.ft fabricated to tanks/bins, bolt together sections.

** Concrete estimated @ Cdn.$175/m3 based on Trondheim local price.factored by 1.2

TABLE 3

GRONG COPPER/ZINC PROJECT

MILL CAPITAL / EQUIPMENT COSTS

			Motor	Cost Cdn$
ITEM	No.	SIZE	H.P / KW	New/Used
Coarse Ore Bin	1	100 tonne(5 x 3 x 3m)	0	*9,000/0
CONE CRUSHER	1	Allis 4000 MF	150/113	0/160,000
SCREEN	1	3x6m	15/11	45,000/0
Dust Collector System	1	10,000 cfm	5/3.75	0 /70, 000
CONVEYORS	5	0.6m W X 110 m. L	5 x 5/19	130,000/0
Fine Ore Bin	1	2000 tonne(6 x 12 x 16m)	0	*71,000/0
Weightometer	1	40-50 tph.cap.	0	9,000/0
BALL MILL	1	3x4.25 m.(includes 62tonnes balls)	900/675	0/275,000
CYCLONEs	1	500mm. Dia.	0	5,000/0
Process Pumps	9	1.5x 1.25 SRL	8x 2h.p./12	41,000/0
	3	1.5x1.25 SRL	2x3h.p./6.75	14,000/0
	2	5 x 5 SRL.	2x40hp/60	15,000/0
	4	5 x 5 SRL	2x30hp/45	31,000/0
	3	Vertical Sump-2inch	5/3.75	26,000/0
CONDITIONER	3	2.5m x 2.5m.tanks	0	*12,000/0
AGITATOR	3	small	3 x 3/6.75	8,000/0
Float. Cells	56	Denver DR100	15hp/630kw	590,000/0
Float Cells	14	Denver Dr 18sp	5hp/53kw	105,000/0
Vacuum Pump	2	Nash CL730	50 hp/36kw	15,000/0
Air Blower	2	4000 scfm ea.	5/3.75	10,000/0
Conc. Thickeners	2	7m.diam.(no tank)	2 x 1/1.5	0/54,000
Thickener Tanks	2	7m. diam. ,9.5mm steel	0	31,000/0
Filters, Drum	2	14 sq.m (157sq. ft.) each	2 x 3/ 4.5	150,000/0
DRYER	1	rotary kiln (electric/diesel fired)	10/7.5	0/ 60,000
Conc. Hold. Tanks	2	1 day-(4m.dia.x4m. high)	0	*17,000/0
Conc.Tank Agitator	2	small	2 x 5/7.5	7,000/0
Conc. Conveyor	2	0.6m.W x20m. L.	5/3.75	75,000/0
Conc. Loading Bay	2	Bins c/w feeders ,for Cu& Zn Concs.(allow)		* 28,000/0
Samplers	10	Vezen-type	10 X 1/7.5	72,000/0
Lime Slaker	1	2 Ton/day	40/30	100,000/0
Reagent Feeders	10	Clarkson	10 X 0.25/2	21,000/0
Return Water Tank	1	40,000 gal., 6m.dia.x6m.high	/0	* 20,000/0
Fresh Water Tank	1	25,000 gal.,5m.dia.x 5m.high	/0	* 18,000/0
Reagent Mixers	3	Barrel Type	3/2.25	8,000
TOTAL			**1,745.25 Kw**	**$2,302,000**

Mill Feed Rate = 1030 tonnes/day ore, 3.5% Zn, 1.0% Cu , 63% Pyrite

Products : Cu conc. =40 tpd, Zn conc. =60 tpd (Future Pyrite conc. = 650 tpd)

TABLE 4
GRONG PROJECT
BUILDING/ SERVICES CAPITAL COSTS

ITEM	DESCRIPTION	Cdn.$ COST
MAINTENANCE SHOP	15 x 20 m. building,c/w welders,drills,grinder	**98,000**
WAREHOUSE	Consists of 20 X 20m Fenced Yard,15x20m. Building	**110,000**
ADMIN.BUILDING	Use two 40ft. Trailers	**8,000**
CHANGE HOUSE	Use one 52ft. Trailer	**21,000**
REAGENT STORAGE	Two 40ft.Containers	**5,000**
EXPLOSIVE STORAGE	Powder magazine(2x40 ft. container)cap and igniter sheds	**5,000**
MILL BUILDING	*One 40 X 86m Pre-engineered Structure	**2,221,000**
	Erection: Labor, crane, boom lifts, fork lifts, scissor lift.	**100,000**
MILL LAB(allowance)		
	Sample Prep. Equip.	
	Crusher, Pulverizer,Drying Oven, Dust Hoods,Tables	**50,000**
	Analytical Equip.	
	X-Ray (quickies), Ph Meters, Scales.	**40,000**
	Atomic Adsorption Product assays of Cu,Zn,Au,Ag.	**50,000**
	Lab Building (use two- 40ft. Trailers)	**8,000**
TOTAL		**$2,716,000**

TABLE 5
GRONG PROJECT
SERVICES/SYSTEMS CAPITAL COSTS

ITEM	DESCRIPTION	Cdn.$ COST
FRESH WATER SYSTEM		
One Well, equipped with Submersible Pump.		5,000
Surface Tank , chlorinator ,TransmissionPump,0.5km.of 2 inch PVC Pipe		20,000
TAILING DAM		
	Grubbing, Excavating, By-pass & Dam Construct.	550,000
	Return Water Pump(size 3X3 SRL)& Float Assembly includes 300meters 3 inch PVC Pipe.	20,000
ACCESS & SITE ROADS		
	7 Kms with compacted base, rolled & graded surface.	250,000
COMMUNICATION SYSTEM		
	Telephone & Radio Systems	50,000
SANITARY SYSTEM		
	Consists of 5 cu.m.Septic Tank&Bed at Mill /Admin Offices,and Change house-one 20 cu.m.tank/bed	30,000
SITE POWER DISTRIBUTION	1Km. of 220v. Line	30,000
MAIN POWER LINE	Cost included in cost/kw hr as rental charge.	000
SITE VEHICLES		
Personnel Van (1)	Required to pick up visitors, Transport Staff to/from Grong	30,000
Pick-up Trucks (3)	One for each of Mine , Mill , Maintenance	75,000
3 Ton Truck (1)	Required to transport Supplies& Parts/Equipment.	40,000
Road Grader (1)	Used, Required to maintain site& access roads	70,000
Loader (1)	Used Kawasaki model 60, 20 cu.yd.	75,000
Ambulance (1)	Used	25,000
TOTAL		**$1,270,000**

TABLE 6
GRONG PROJECT
SUMMARY OPERATING COSTS
Cdn.$/tonne, 360,000 TONNES/YEAR

	MINE	MILL	ADMINISTRATION
PERSONNEL	8.55	8.05	1.04
POWER	0.94	4.06	0.05
FUEL/LUB.			
VEHICLES	0.78	0.19	0.30
SUPPLIES			
EXPLOSIVES	2.08		
MAINTENANCE	0.06	0.03	0.02
BALLS		0.74	
REAGENTS		1.32	
CONCENTRATE			
MARKETING			2.87
TOTAL	12.41	14.39	4.28

MINE + MILL + ADMINISTRATION Costs,$/tonne=$31.08

TABLE 7
GRONG PROJECT
STAFF/LABOUR COST ESTIMATE

CATEGORY	NUMBER	ANNUAL WAGE, Cdn.$	YEARLY TOTAL
STAFF			
Manager(ex-pat)	1	120,000	120,000
Mill Superintendent	1	60,000	60,000
Mine Superintendent	1	60,000	60,000
Maintenance Supt.	1	60,000	60,000
Mine Foreman	3	54,000	162,000
Mill General Foreman	1	54,000	54,000
Mine Geologist	1	55,000	55,000
Mine Surveyor/draft.	1	48,000	48,000
Warehouseman	1	40,000	40,000
Secretary	1	34,000	34,000
Warehouse Clerk	1	34,000	34,000
Social Costs(30%)	—		218,000
Sub-Total	13		945,000
LABOUR			
Miners	24	23.40/hr.(incl. bonus)	1,402,000
Mechanics(mine)	15	19.25	721,000
Mechanics(surface)	20	19.25	818,000
Mill Operators (includes sample/assay)	28	22.60	1,383,000
Social Costs(25%)	—		1,081,000
Sub-Total	87		5,405,000
TOTAL	**100**		**6,350,000**

* Norwegian Pay Scale

TABLE 8
GRONG PROJECT
SUPPLIES/CONSUMABLE COST ESTIMATE 360,000 tonnes/year

		ANNUAL COST,Cdn.$
ELECTRICAL POWER(0.9 power factor)		
MINE ; 670 kw installed , 3,400,000kw.hr./yr.@ $0.11/kwhr x 0.9		340,000
MILL ; 1745.25kw installed, 14,700,000 kw.hr./yr.@$0.11/kwhr x 0.9		1,460,000
ADMINISTRATION;20 kw,allow170,000kw.hr./yr@$0.11/kwhr x0.9		17,000
SERVICE VEHICLES(3)	Lubrication/gasoline/diesel(allow)	21,000
UNDERGROUND VEHICLES(7)	Lub./gasoline/diesel (allow)	210,000
SITE VEHICLES(5)	Lubrication/gasoline/diesel (allow)	100,000
DRYER FUEL	Allow 106 liters/day @$0.44/l.	62,000
EXPLOSIVES	ANFO@ $1.60/kg ,	
	- Ore Blasting, 1.1 kg/t.ore	635,000
	- Waste Blasting 0.8 kg/t. waste	20,000
	-Auxiliary Explosives(caps.diesel)	95,000
STEEL BALLS	3 inch@ $900/tonne, 0.81kg/ t. of ore	265,000
LIME STORAGE	Silo 4m D x 7m Ht., 21t. cap.,+Feeder (rent)	15,000
MILL REAGENTS		
	Sodium Bisulphite@$0.75/kg,122g/t ore	33,000
	Lime,$0.25/ kg.,3kg/t.ore	270,000
	MIBC @ $2.50/kg,50g./ t. ore	45,000
	Collector(A3418&R208) @ $2.50/kg, 100g./ t. ore	90,000
	Copper Sulphate @ $1.50/kg., 50g./ t. ore	27,000
	Flocculant @ $4.00/ kg., 5 g. / t. ore	8,000
MAINTENANCE **CONSUMABLES**	MILL (ALLOW)	10,000
	MINE (ALLOW)	20,000
	ADMINISTRATION	5,000
TOTAL		3,748,000

TABLE 9
GRONG PROJECT
SUMMARY OF REVENUE V.S. METAL PRICES AND CAPITAL PAYBACK

CAPITAL:New =Cdn$15,555,000   OPERATING COST =Cdn$11,190,000/YEAR
Used =Cdn$12,265,000

SCENARIO#1 Current Metal Prices
Copper@ us$1.00/lb, Zinc@us$0.50/lb, Gold@us$380/oz. Silver@us$5.00/oz.
NET SMELTER RETURN = US$10,534,982 or Cdn$14,116,876 per year
NET PROFIT AFTER OPERATING COSTS=Cdn$2,926,876 per year

PAYBACK(New) = 15,555,000/2,926,876 = 5.3 years OR 64 Months
(Used)=12,265,000/2,926,876= 4.2 years OR 50 Months

SCENARIO#2 Possible Future Metal Prices
Copper@us$1.10/lb, Zinc@us$0.55/lb Gold@us$400/oz Silver@us$5.00/oz.
NET SMELTER RETURN = US$11,635,305 or Cdn$15,591,309 per year
NET PROFIT AFTER OPERATING COSTS =Cdn$4,401,309 per year

PAYBACK (New)= $15,555,000/4,401,309 = 3.53 Years OR 42 Months
(Used)= $12,265,000/3,471,000 = 2.79 Years OR 33 Months

SCENARIO#3 Longterm Future Metal Prices
Copper@us$1.15 Zinc@us$0.60/lb Gold@us$400/oz. Silver@us$5.00/oz.
NET SMELTER RETURN = US$12,343,117 or Cdn$16,539,777 per year
NET PROFIT AFTER OPERATING COSTS = Cdn$5,349,777 per year

PAYBACK(New) = 15,555,000/5,349,777 = 2.9 Years OR 34 Months
(Used)= 12,265,000/5,349,777 = 2.29 Years OR 27 Months

TABLE 10
GRONG PROJECT
ANNUAL COST and REVENUE SUMMARY
360,000 TONNE/YEAR MINE& MILL OPERATION

		Cdn. $/Year
CAPITAL EQUIPMENT/INSTALLATION		13,015,000
WORKING CAPITAL Initial 3 Months		2,540,000
	Total Capital	**15,555,000**
OPERATING Annual Cost Mine,Mill&Admin.		10,156,000
Concentrate Shipping		
Trucking to Port 80 km		350,000
Port Loading Fee		40,000
Cu conc. Ocean Freight/Finland		350,000
Zn conc. Ocean Freight/ODA		294,000
	Sub-total	1,034,000
TOTAL OP. COSTS		**11,190,000**
REVENUE (Canadian $) Current Metal Prices		
COPPER: 3192t payable Cu., US $0.88/ pound		7,853,200
ZINC : 4693t.metal, @US $0.47/lb LME 3 month Zn price US $1019/tonne		4,644,400
SILVER :62,437oz. Ag @ US $ 5.15 tr.oz. LME 3month spot		384,800
GOLD ; 2463 oz.gold,@$380u.s./oz (current)		1,234,300
TOTAL REVENUE (ANNUAL)		**14,116,700**
NET PROFIT = Total Revenue - Total Op. Cost=		**$2,926,700**

TABLE 11
GRONG PROJECT
NET SMELTER REVENUE ESTIMATE

SCENARIO#1 Current Metal Prices

COPPER PRICE=us $1.00/lb. ZINC =us$0.50/lb. GOLD = us$380.00/oz SILVER=us$5.00/oz

(1)Payable Copper content=13,300t/yr (25%-1) =3192 tonnes Copper/yr =7,037,037 lbs./yr.

Copper value = 7,037,037 x $1.00 = us$7,037,037

Less: treatment charge(TC)us$115/dmt = us$367,100

refining charge(RC)us$11.5cents/lb. Cu = us$809,300

NET PROCEEDS = us$5,860,637

(2) ZINC CONC. , 8856 tonnes/year@53% Zn = 4693 tonnes Zinc/yr .

Zinc value = 4693t x 2204.6 lbs.x us$0.50/lb. x 67%(smelter return) =us$3,465,973

(3) GOLD Content(copper conc.) 13,300tpy x (6.76g/t- 1g/t TC) = 76,608 g/ yr OR 2463oz./yr

PayableGold =2463 oz x (us$380/oz - $6/oz RC) = us$921,162

(4) SILVER Content(copper conc.) 13,300 t/yr x (176g/t -30TC) = 1,941,800g. OR 62,437 oz.

Payable Silver = 62,437 x (us$5.00/oz-$0.4/oz RC) = us$287,210

TOTAL REVENUE of Payables = us$10,534,982

OR Cdn$14,116,876

SCENARIO#2 Possible Future Metal Prices

COPPER PRICE= us$1.00/lb. ZINC = us$0.55/lb GOLD =us$400/oz. SILVER = us$5.00/oz.

Copper Revenue =7,037,037lb/yr x us$1.10 -$1,176,400TC =us$6,564,340

Zinc Revenue = 10,346,119lbs. x us$0.55 x 67%(est. smelter return) = us$3,812,545

Gold Revenue = 2465oz/yr x (400 - 6Rc) = us$971,210

Silver Revenue = (as in scenario#1) = us$287,210

TOTAL REVENUE of Payables = us$11,635,305

OR Cdn$15,591,309

SCENARIO #3 Longterm Future Metal Prices

COPPER PRICE = us$1.15lb. ZINC =us$0.60/lb GOLD & SILVER same as scenario#2

Copper Revenue = 7,037,037lbs. x us$1.1 - 1176400TC =us$ 6,916,192

Zinc Revenue = 10,346,119 lbs x us$0.60 x 67% =us$4,159,140

Gold Revenue = (as in 2) = us$971,210

Silver Revenue = (as in 2) = us$296,575

TOTAL REVENUE of Payables=us$12,343,117

OR Cdn$16,539,777

NOTE :(a) Basis for Treatment/Refining Charges
From OUTOKUMPU Harjavalta Metals quoteNov.1/96

(b) Currency Exchange: US$1.00 =1.34$Canadian

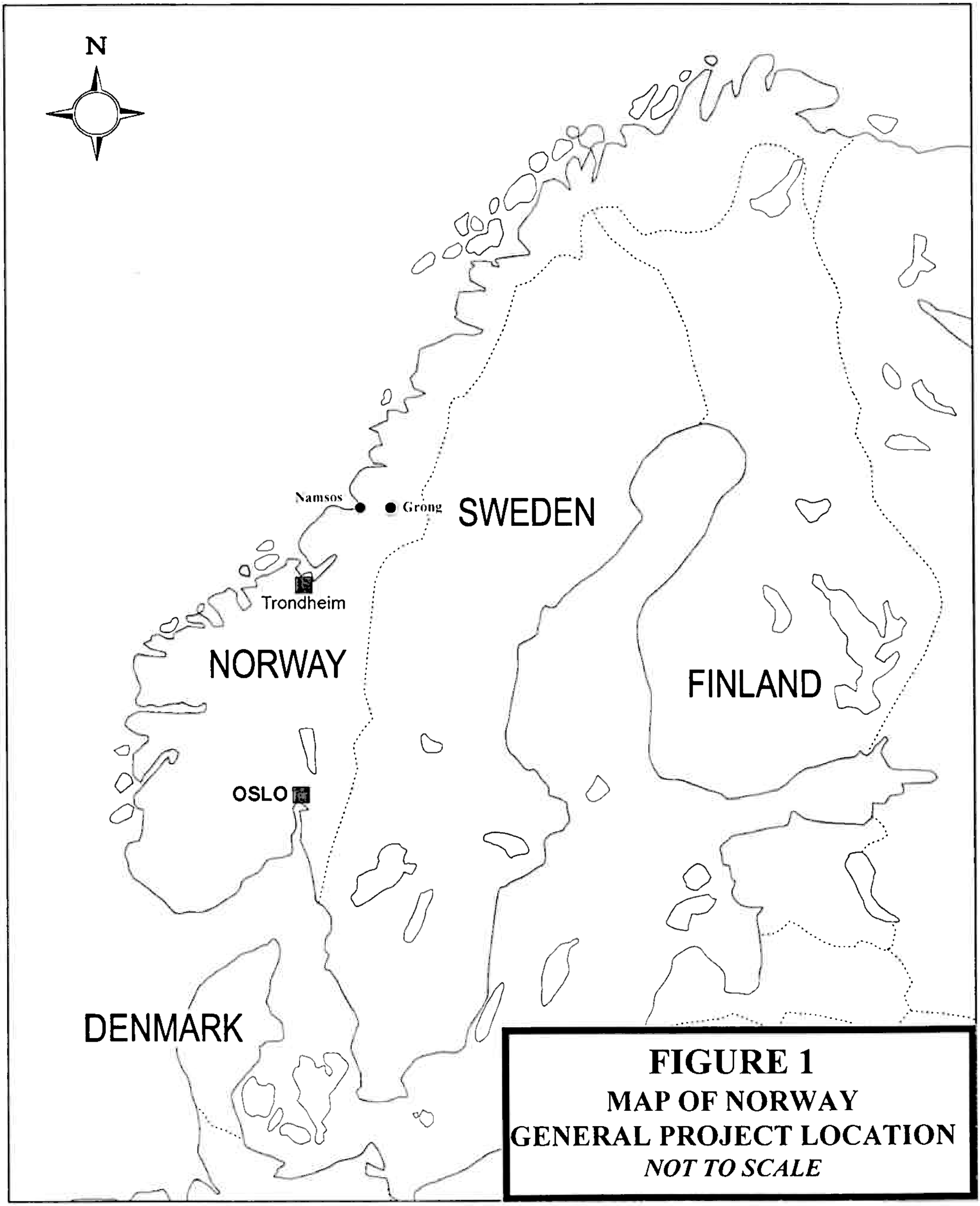

FIGURE 1
MAP OF NORWAY
GENERAL PROJECT LOCATION
NOT TO SCALE

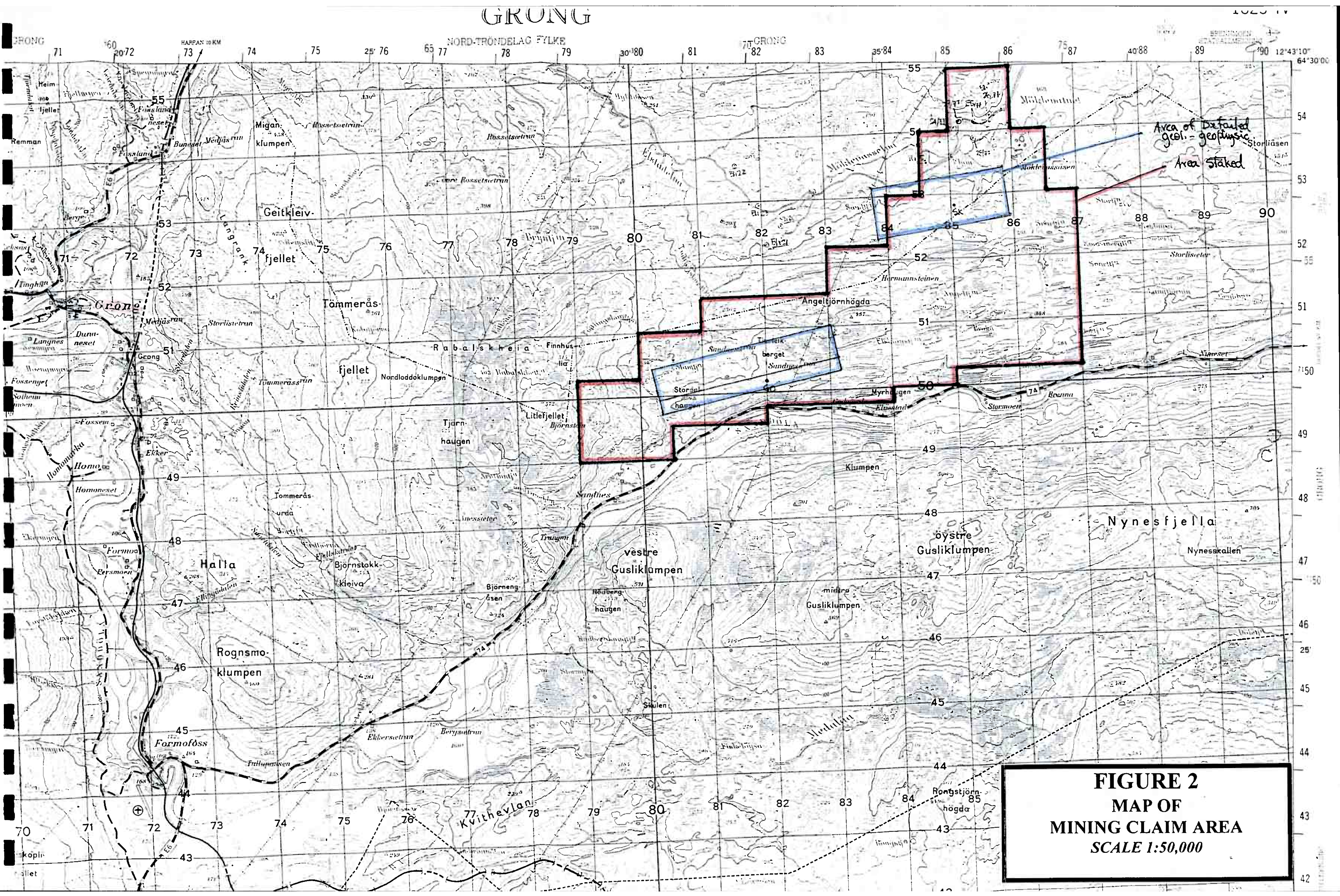

FIGURE 2
MAP OF
MINING CLAIM AREA
SCALE 1:50,000

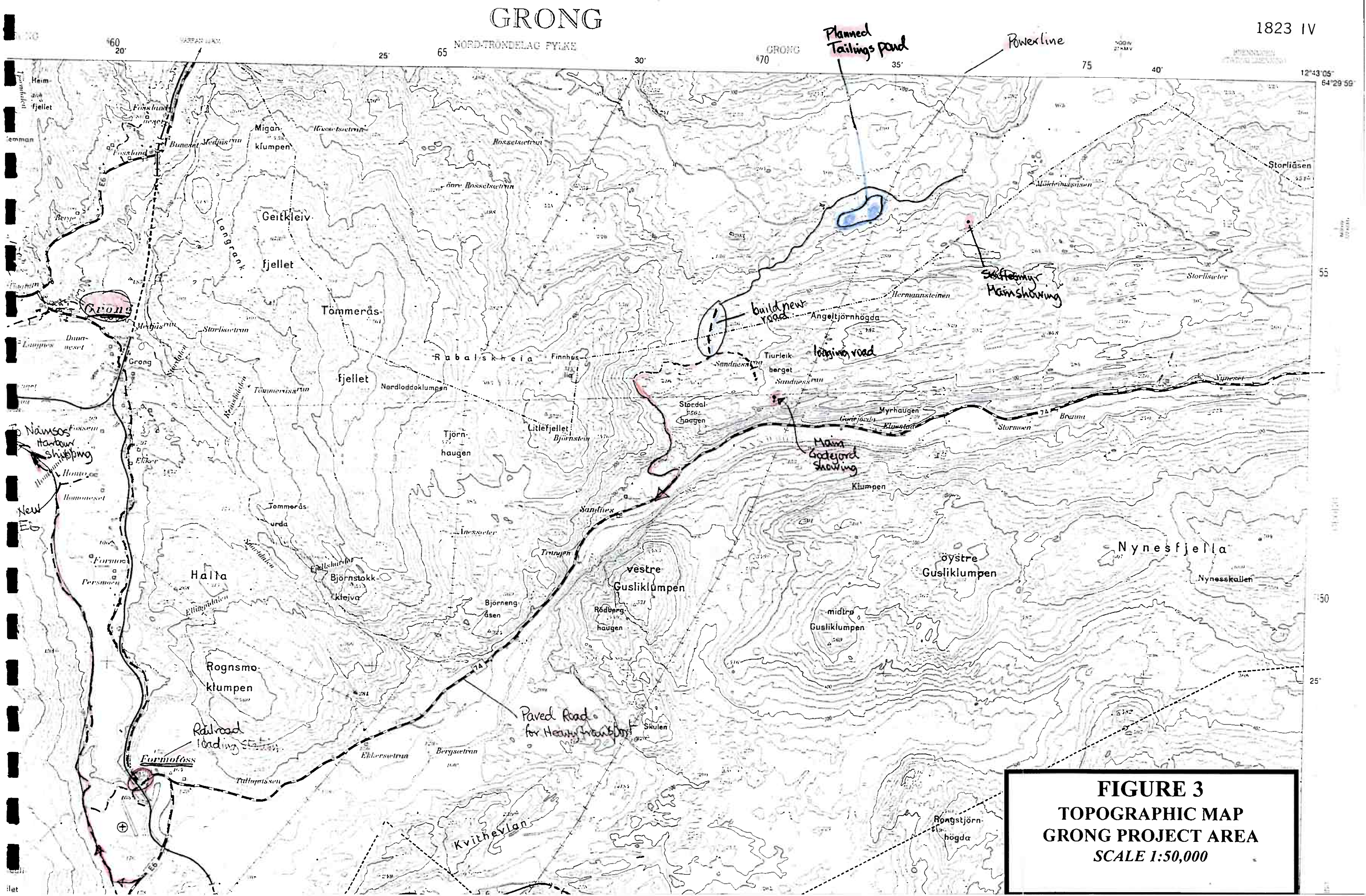

FIGURE 3
TOPOGRAPHIC MAP
GRONG PROJECT AREA
SCALE 1:50,000

A Laboratory Investigation of
The Flotation Behavior of Samples
from the Grong Deposit
submitted by
Braddick Resources Ltd.

Progress Report No. 1

Project No. LR5031

NOTE:
This report refers to the samples as received.

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February 4, 1997

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Abstract

Four samples from the Grong copper/zinc deposit in Norway were tested for flotation separation into three products: a copper concentrate, zinc concentrate and iron concentrate. Two separation schemes were tested on two samples, GO-1 and SKM-1. The tests using sodium sulphite as the iron sulphide depressant yielded better results than the tests using SO_2. Therefore, all tests after Test F4 used the sodium sulphite reagent scheme.

The best results for each of the four samples are listed in Table 1.

After initial screening tests, the majority of the flotation tests concentrated on sample SKM-2, as this sample best represents the bulk of the deposit. Optimization and repeat tests on SKM-2 were hampered by changing sample chemistry, likely due to oxidation, despite freezer storage of the sample. Test results deteriorated over time and required increasing dosage rates of collector and zinc activator.

The objective of the test program was to determine the flotation response in a limited number of bench tests. Only a limited reagent optimization was performed; whereas no changes were made to grinding conditions, which were maintained at a fairly aggressive level of a K_{80} = 50-55 μm in the primary grind. More testing will be required to optimize conditions.

Table 1: Test Summary

Test Number		GO-1 F3		GO-2 F8		SKM-1 F4		SKM-2 F7,F12	
		Assay	***Distrib.***	**Assay**	***Distrib.***	**Assay**	***Distrib.***	**Assay**	***Distrib.***
Head	Cu %	1.00		3.20		1.60		0.97	
	Zn %	15.7		19.3		0.35		1.90	
	Au g/t	1.82		0.76		1.39		0.38	
	Ag g/t	54.3		46.0		19.5		22.5	
	Pb %	0.39		0.41		0.10		0.052	
Copper Concentrate	**Cu %,%**	**20.1**	***84.2***	**22.8**	***87.3***	**23.4**	***87.4***	**22.4**	***83.8***
	Zn %,%	13.4	*4.20*	10.1	*7.20*	0.31	*6.17*	4.89	*8.50*
	Au g/t,%	23.2	*58.5*	4.59	*73.9*	11.8	*49.1*	3.73	*35.4*
	Ag g/t,%	835	*70.5*	257	*68.4*	170	*50.4*	137	*21.9*
	Pb %,%	5.05	*60.1*	1.70	*53.4*	0.76	*46.9*	0.15	*10.00*
Zinc Concentrate	**Zn %,%**	**62.8**	***88.1***	62.1	*83.3*	0.85	*76.6*	56.4	***76.3***
	Cu %,%	0.16	*3.00*	0.33	*2.40*	0.24	*4.10*	1.45	*4.20*
	Au g/t,%	*0.16*	*1.60*	0.23	*7.00*			1.42	*10.4*
	Ag g/t,%	*13.3*	*4.50*	54.8	*27.4*			66.0	*8.20*
	Pb %,%	0.063	*3.40*	0.100	*5.70*	14.7	*14.7*	0.21	*11.0*
Iron Concentrate	**Fe %,%**	**41.3**	***84.2***			**48.3**	***33.5***	**45.1**	***81.6***
	Cu %,%	0.26	*8.30*			0.16	*2.05*	0.13	*7.90*
	Zn %,%	2.29	*5.60*			0.15	*10.00*	0.11	*3.40*
	Au g/t,%	0.73	*14.3*					0.24	*39.1*
	Ag g/t,%	24.0	*15.7*					12.8	*35.2*
	Pb %,%	0.087	*14.1*			0.042	*8.70*	0.037	*45.8*

Underlined values are not on the same product

Introduction

Four test samples from the Grong Cu/Zn deposit in Norway were submitted for testing of the flotation response. The samples were submitted by Braddick Resources Ltd. with the technical supervision provided by Mr. Louis Bernard, an independent Mining Consultant.

The objective of the work was to determine the feasibility of separating the samples in three concentrates, copper, zinc and iron, and determine the metal distribution between these products.

The test program was executed in close communication with Mr. Louis Bernard.

Lakefield Research Limited

Hans Raabe
Senior Engineer

K.W. Sarbutt
Manager - Mineral Processing

Experimental testwork by: F. Vincent; G. Coppaway
Report preparation by: B.J. Scobie

Summary

1. Sample Description

Three crates of Grong ore were received on October 24, 1996 and were issued our receipt number LR9606664.

The crates contained a number of sample bags as identified in Table 2.

Table 2: Sample Information

Box Number	Number Bags	Sample I.D.	Weight kg	Name	Description
1	2	GO-1	17.6	Godejord	Sphalerite rich massive to semi-massive Ore
	1	GO-2	7.0	Godejord	As GO-1, semi-massive, increased Cu content
	1	SKM-2	10.3	Skiftesmyr	Massive Pyrite Ore
2	2	SKM-1	17.8	Skiftesmyr	Cu-rich disseminated, semi-massive ore
	1	SKM-2	12.2	Skiftesmyr	Massive Pyrite Ore
3	2	SKM-1	17.5	Skiftesmyr	Cu-rich disseminated, semi-massive ore
Total	**2**	**GO-1**	**17.6**	**Godejord**	**Sphalerite rich massive to semi-massive Ore**
	1	**GO-2**	**7.0**	**Godejord**	**As GO-1, semi-massive, increased Cu content**
	4	**SKM-1**	**35.3**	**Skiftesmyr**	**Cu-rich disseminated, semi-massive ore**
	2	**SKM-2**	**22.5**	**Skiftesmyr**	**Massive Pyrite Ore**

All samples with identical sample identification were combined, as displayed in the 'Total' section of the table. The combined samples were crushed to minus 3/4", on November 4, 1996, and a head-sample was riffled from each product for an ICP scan and mineralogical work. Only for samples GO-1 and SKM-1 were 12 test charges of 1 kg separated and crushed to minus 10 mesh. All products were stored in the freezer to minimize oxidation.

The full 25-element ICP scan of the four samples is appended and is summarized in Table 3.

Flotation test charges for sample SKM-2 were prepared on November 25, 1996 and for sample GO-2 on December 2, 1996.

Table 3 : Head Assays

		GO-1	GO-2	SKM-1	SKM-2
Ag	g/t	54.3	46.0	19.5	22.5
Au	g/t	1.82	0.76	1.39	0.38
Cu	%	1.00	3.20	1.60	0.97
Fe	%	17.9	14.3	30.9	33.7
Zn	%	15.7	19.3	0.35	1.90

2. Previous Testing

No previous testing was available on these ores.

3. Laboratory Testing Summary

Two reagent schemes were tested, based on previous testwork on other copper-zinc ores. The first uses SO_2 as iron sulphide depressant, the second uses sodium sulphite. In Tests F1 through F4, the sodium sulphite reagent scheme yielded better results on the GO-1 and SKM-1 samples. As a result, all further testing was performed using sodium sulphite.

The test results are summarized in Table 4 and described in detail in Section 4. The detail test conditions and assay results are appended.

The assays reported are those products with the best grade-recovery balance for each test. This is not always the final cleaning stage.

Table 4: Flotation Tests Summary

Test	Sample	Copper Conc		Zinc Conc		Iron Conc	
		% Cu	Cu-Recov	% Zn	Zn Recov	% Fe	Fe-Recov
F1	GO-1	7.40	77.0	53.5	46.8	45.8	22.1
F2	SKM-1	28.1	79.7	10.8	37.6	42.9	21.8
F3	GO-1	20.1	84.2	62.8	88.1	41.3	84.2
F4	SKM-1	23.4	87.4	0.85	76.6	48.3	33.5
F5	GO-1	19.5	83.6	64.2	89.0		
F6	SKM-2	25.2	80.5	32.5	62.7		
F7	SKM-2	22.4	83.8	56.4	76.4		
F8	GO-2	22.8	87.3	62.1	83.3		
F9	SKM-2	30.9	74.4	45.0	28.5		
F10	SKM-2	23.4	61.9	36.5	24.5		
F11	SKM-2	13.0	80.0	31.9	8.00	43.9	69.6
F12	SKM-2	27.4	48.9	45.9	76.0	45.1	81.6

The flotation of an iron concentrate was suspended after the first four tests, as too much iron was recovered in the copper and zinc flotation stages to yield meaningful data on iron concentrate grade and recovery. For sample SKM-2, two tests were specifically run for iron recovery in Tests F11 and F12

In summary, the best results obtained for each of the four samples are listed in Table 5.

Table 5: Best Test Results

Test Number		GO-1 F3		GO-2 F8		SKM-1 F4		SKM-2 F7,F12	
		Assay	*Distrib.*	Assay	*Distrib.*	Assay	*Distrib.*	Assay	*Distrib.*
Head	Cu %	1.00		3.20		1.60		0.97	
	Zn %	15.7		19.3		0.35		1.90	
	Au g/t	1.82		0.76		1.39		0.38	
	Ag g/t	54.3		46.0		19.5		22.5	
	Pb %	0.39		0.41		0.10		0.052	
Copper Concentrate	**Cu %,%**	**20.1**	***84.2***	**22.8**	***87.3***	**23.4**	***87.4***	**22.4**	***83.8***
	Zn %,%	13.4	*4.20*	10.1	*7.20*	0.31	*6.17*	4.89	*8.50*
	Au g/t,%	23.2	*58.5*	4.59	*73.9*	11.8	*49.1*	3.73	*35.4*
	Ag g/t,%	835	*70.5*	257	*68.4*	170	*50.4*	137	*21.9*
	Pb %,%	5.05	*60.1*	1.70	*53.4*	0.76	*46.9*	0.15	*10.00*
Zinc Concentrate	**Zn %,%**	**62.8**	***88.1***	62.1	*83.3*	0.85	*76.6*	56.4	***76.3***
	Cu %,%	0.16	*3.00*	0.33	*2.40*	0.24	*4.10*	1.45	*4.20*
	Au g/t,%	<u>*0.16*</u>	<u>*1.60*</u>	0.23	*7.00*			1.42	*10.4*
	Ag g/t,%	<u>*13.3*</u>	<u>*4.50*</u>	54.8	*27.4*			66.0	*8.20*
	Pb %,%	0.063	*3.40*	0.100	*5.70*	14.7	*14.7*	0.21	*11.0*
Iron Concentrate	**Fe %,%**	**41.3**	***84.2***			**48.3**	***33.5***	**45.1**	***81.6***
	Cu %,%	0.26	*8.30*			0.16	*2.05*	0.13	*7.90*
	Zn %,%	2.29	*5.60*			0.15	*10.00*	0.11	*3.40*
	Au g/t,%	0.73	*14.3*					0.24	*39.1*
	Ag g/t,%	24.0	*15.7*					12.8	*35.2*
	Pb %,%	0.087	*14.1*			0.042	*8.70*	0.037	*45.8*

Underlined values are not on the same product

Gold and silver assays were performed on selected concentrate samples only; in some cases on reconstituted, intermediate, concentrates. Table 6 lists the results of the gold and silver content in the copper concentrates.

Table 6: Gold and Silver Recovery in the Copper Concentrate

Test	Sample	Copper Concentrate					
		Cu (%)	Au (g/t)	Ag (g/t)	Cu-Distr	Au-Distr	Ag-Distr
F3	GO-1	20.1	23.2	835	84.2	58.5	70.5
F4	SKM-1	23.4	11.8	170	87.4	49.1	50.4
F6	SKM-2	25.2	5.4	93.0	80.5	45.4	13.3
F7	SKM-2	28.5	7.7	232	36.9	25.3	12.9
		22.4	3.7	137	83.8	35.4	21.9
F8	GO-2	21.9	11.1	346	25.3	53.7	27.7
		22.8	4.6	257	87.3	73.9	68.4
F9	SKM-2	30.9		266	74.4		29.2
F10	SKM-2	29.7	8.1	155	25.0	17.8	5.8
F11	SKM-2	13.0	4.3	85.7	80.0	68.6	23.4

The effect of the copper recovery and concentrate grade on the recovery of gold and silver in this product, for the SKM-2 sample, is demonstrated in Figures 1 and 2.

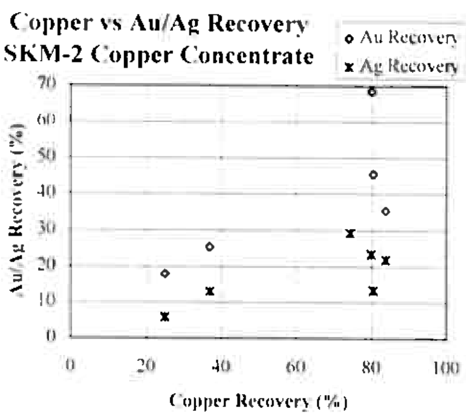

Figure 1

Gold and Silver Recovery as a Function of Copper Recovery

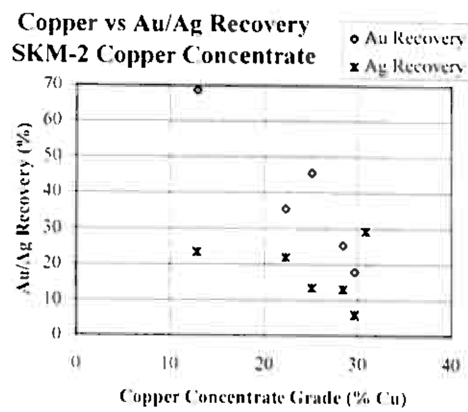

Figure 2

Gold and Silver Recovery as a Function of Copper Grade

Gold and silver recovery will benefit from additional testing with a variety of collectors and collector dosage rates.

Aeration of the pulp, following primary grinding, was used in Tests F1 through F6. It was dropped in Test F7, with no deleterious results on sample SKM-2; and not used in all subsequent tests. The value of aeration for each of the four samples is not established in this test program.

The majority of the test was performed on sample SKM-2, considered the most representative sample for the bulk of the deposit. Test F7, executed shortly after crushing the sample, was the most successful test on this ore sample, while overtime problems were encountered with oxidation being the most likely explanation. Despite storage of all samples in the freezer, oxidation could not be avoided. A simple test was executed to demonstrate oxidation. Equal weights of a -10 mesh sample and a 'fresh' sample were mixed with equal volumes of water and the pH was measured. The aged sample had a lower pH than the 'fresh' sample at 3.1 and 3.5 respectively, indicating an increase in oxidation[1]. The low pH of the 'fresh' sample indicates that the original sample is partially oxidized.

The changing sample chemistry exhibited itself by lower recoveries, despite identical test conditions. Reagent dosage rates, specifically collector and zinc activator, needed to be increased to compensate. Future testing on these ores should be performed on fresh sample only, prepared not more than a week prior to flotation testing.

Due to the limited number of tests in this program, no attempt was made to optimize conditions. Specifically, grinding was performed to a fine product size of K_{80} = 50-55 µm. It is likely that a more power efficient grinding scheme can be devised, using a coarser primary grind followed by regrinding of selected concentrates.

Similarly, reagent additions were high, specifically the depressant in the primary grind. Other depressants, or relocation of the depressant in the cleaning stages, may be more cost effective.

No mineralogy was performed to determine mineral composition or mineral liberation.

[1] A detailed procedure is appended

4. Laboratory Testing Detail

This section describes each test in detail. All detail test reports and mass balance calculations are attached.

Test F1

With no previous test information available, testing was started using a flotation scheme used for a Canadian Cu/Zn operation. The treatment method is based on an SO_2 depression of zinc and pyrite. The pH is controlled and the Eh is monitored throughout the test procedure. Flotation timing was based on previous test programs and modified, based on visual observations.

Table 7: Test Results - Test F1

F1 GO-1	Grind K_{80}	Assays Cu	 Zn	 Fe	 Pb	Distribution Cu	 Zn	 Fe	 Pb
Copper Concentrate		**7.41**	27.90	22.80	2.27	**77.0**	22.9	15.9	69.1
Zinc Concentrate		0.27	**53.49**	9.70	0.07	3.0	**46.8**	7.2	2.4
Pyrite Concentrate		0.20	1.79	**45.80**	0.09	1.4	1.0	**22.1**	1.9
Head	51	1.20	15.18	17.87	0.41	100	100	100	100

Copper concentrate grade was disappointing. Both zinc and pyrite were poorly depressed. Higher levels of depressants will be required in the grinding and aeration stages. Similarly, depressants will be needed in the cleaning stages. The majority of the lead is reporting to the copper concentrate.

Zinc concentrate grades were excellent at 66.5% Zn in the 3rd cleaner. In fact, an acceptable concentrate grade of 53.5% Zn was obtained in the first cleaner at a 10% higher zinc recovery. This recovery, at 46.8%, is low, mainly due to the loss of 44.5% of the zinc in the copper rougher.

Assuming that the pyrite concentrate contains no pyrrhotite, the concentrate is quite pure at 45.8% Fe[2]. Recovery, however, is quite poor at 22.1% Fe. During the test run, there was a concern that rougher flotation time was too short, and pyrite was lost in the tailings. A scavenger stage was used, which demonstrated that only 7.8% of the iron was lost in the rougher tails with only 2.0% recoverable after 2 minutes flotation and a doubling of the collector. The low iron recovery was mainly due to the iron losses in the previous flotation stages, in which 53.9% of the iron was removed. Another 16.3% of the iron was lost in the pyrite cleaners. The cleaners improved the iron content only marginally, but reduced the contaminants in the concentrate as follows: Cu: 0.39% to 0.20%, Zn: 2.01% to 1.79% and Pb: 0.19% to 0.09%.

Test F2

Laboratory Test F2 was performed on sample SKM-1, using a similar flotation scheme as was used in Test F1. Based on the observations in Test F1, some changes were made:

- SO_2 was added to the copper cleaning stages, which improved copper concentrate grade.
- Copper sulphate addition to the zinc flotation was reduced in view of the lower zinc content. During the test, the dosage was doubled (from 100 to 200 g/t) based on poor zinc showing. It appears that this activated the pyrite as well.
- A zinc scavenger was added. The objective was to determine if collector dosage in the zinc rougher was adequate and to minimize zinc recovery in the pyrite concentrate. Less than 1% of the zinc was recovered in this product, indicating that collector dosage was adequate. The highest grade iron product (52.5% Fe) was recovered in this product.
- The cleaner stages in the pyrite flotation were eliminated in preference to a staged rougher flotation. The objective was to determine reagent requirements and flotation kinetics.
- In view of the lower lead content of the ore, no lead assays were included.

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[2] Iron content in pyrite is 46.7% Fe. For pyrrhotite the iron content is 61.5% Fe.

The results are shown in Table 8.

Table 8: Test Results - Test F2

F2 SKM-1	Grind K_{80}	Assays Cu	Zn	Fe	Pb	Distribution Cu	Zn	Fe	Pb
Copper Concentrate		**28.10**	0.32	32.3		**79.7**	5.0	5.2	
Zinc Concentrate		2.02	**10.80**	44.9		1.3	**37.6**	1.6	
Pyrite Concentrate		0.32	0.11	**42.9**		2.9	5.4	**21.8**	
Head	55	1.61	0.29	28.2		100	100	100	

The SKM-1 sample, apart from the much lower zinc content and higher iron content, behaves differently in flotation. The pulp potential after grinding was negative 120 mV, compared to positive 40 mV for GO-1. Aeration increased the potential to a positive value. Froth colour in the copper flotation was black-grey. The reason for this has not been identified.

Copper concentrate cleaned well, using SO_2 as the cleaning depressant to 28.1% Cu. Fine-tuning the flotation conditions can be expected to increase the recovery above the 79.7% obtained in this test.

Zinc recovery in the zinc rougher was good; 90.6% of the zinc in the copper flotation tailings was recovered. Unfortunately, 66.3% of the iron in the copper tails was also recovered. Rejection of this iron in the zinc cleaners was good, but at a significant loss of the zinc and poor concentrate grade (10.8% Zn at 37.6% Zn-recovery). Lower copper-sulphate and collector addition rates may be able to improve zinc flotation.

Pyrite rougher flotation was difficult to assess in this test, as only 19% of the iron reported to the pyrite flotation. Of the iron in the zinc rougher tail, 35% was recovered in the zinc scavenger, prior to lowering the pH. The grade of this material (52.5% Fe) indicated a high proportion of pyrrhotite. Concentrate grades in the pyrite roughers were substantially lower (19.3% to 32.8% Fe). Figure 3 shows the pyrite flotation rougher kinetics. It should be borne in mind that only 19% of the iron was processed in the pyrite flotation and the kinetics show the 'tail' of the iron flotation.

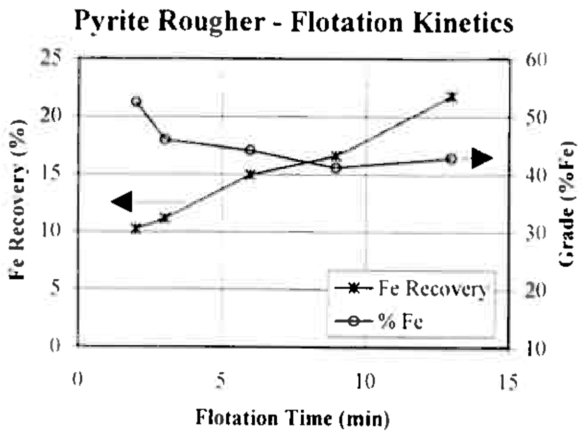


Figure 3: Iron Flotation Kinetics (F2)

Test F3

Sodium sulphite at elevated pH was used as depressant on the GO-1 sample. The results are listed in Table 9.

Table 9: Test Results - Test F3

F3	Grind	Assays				Distribution			
GO-1	K_{80}	Cu	Zn	Fe	Pb	Cu	Zn	Fe	Pb
Copper Concentrate		**20.1**	13.4	20.5		**84.2**	4.21	5.38	
Zinc Concentrate		0.16	**62.8**	1.60		3.00	**88.1**	1.88	
Pyrite Concentrate		0.26	2.29	**41.3**		8.32	5.60	**84.2**	
Head		1.10	14.6	17.5		100.0	100.0	100.0	

Copper flotation results were superior over those obtained in Test F1, using SO_2 depression. Flotation pH was 10.0 in the rougher and was increased from 10.5 to 11.0 in the cleaners. Soda ash was used in the third stage cleaning as a dispersant with little benefit. Rougher flotation time and collector addition will need to be tested to improve copper recovery (87.7% Cu recovery). Little, if any, iron sulphides were recovered in the copper concentrate. Sphalerite and gangue are the main diluants. Dextrin, as gangue depressant, can be tested to improve concentrate grade.

Copper sulphate and collector addition rates were lowered, relative to Test F1, based on the observation of a very tight froth. This resulted in a high zinc concentrate grade. In fact, cleaning beyond the rougher stage did not yield significant improvements. Longer rougher flotation times need to be tested to improve the zinc recovery in the rougher stage beyond the 88.1%.

As in the previous test, a zinc scavenger was included with the objective of removing contaminants, prior to pyrite flotation. Instead, 72.8% of the pyrite was floated. The key factor is the collector addition; all other factors were kept constant. Collector addition rates will be a critical factor in controlling zinc flotation grade and pyrite recovery in the zinc concentrate.

Given sufficient collector the pyrite floats readily and fast. Lowering the pH and addition of high collector rates did not yield significant improvements. Including the zinc scavenger concentrate, iron recovery in the combined concentrate was 96.7% of the iron in the zinc rougher tailings.

The zinc content in the pyrite concentrate, however, is high. This will be addressed with longer flotation times in zinc rougher flotation.

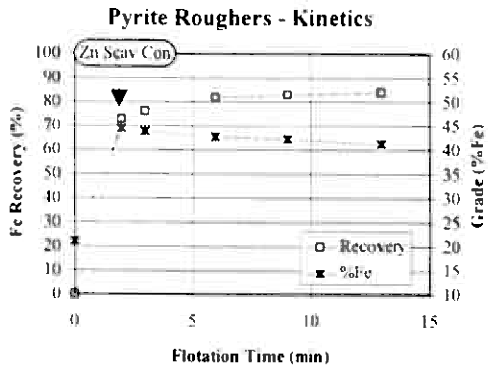


Figure 4: Pyrite Rougher Kinetics (F3)

Test F4

The test was performed on SKM-1 ore using a elevated pH and sodium sulphite depression. The test results are summarized in Table 10.

Table 10: Test Results - Test F4

F4 SKM-1	Grind K_{80}	Assays				Distribution			
		Cu	Zn	Fe	Pb	Cu	Zn	Fe	Pb
Copper Concentrate		**27.0**	0.32	31.9		**84.4**	5.28	5.46	
Zinc Concentrate		0.81	**1.54**	45.9		1.53	**15.4**	4.75	
Pyrite Concentrate		0.16	0.15	**48.3**		2.05	10.0	**33.5**	
Head		1.55	0.29	28.3		100.0	100.0	100.0	

Higher lime and depressant levels were added to the primary grind, to improve zinc and iron depression. In addition, collector rate in the copper rougher was reduced. Copper recovery in the rougher improved over Test F2, while zinc and iron recovery decreased marginally. The third concentrate grade was similar to Test F2, with a 4.5% higher recovery. Zinc and iron reporting to the copper concentrate was similar in the two tests.

Substantially lower collector was used in the zinc rougher (10 g/t relative to 50 g/t in Test F2). Zinc recovery suffered as a result; 86.1% of the zinc was recovered in the rougher, compared to 90.6% in Test F2). The lower collector did benefit iron rejection; 53.9% of the iron was recovered in the rougher concentrate compared to 66.3% in Test F2. In the cleaning stages, however, the zinc proved under-collected and little upgrading took place. Final concentrate grade was 1.54% Zn at a low 15.4% recovery. Longer rougher flotation times and collector addition to the cleaning stages may improve the results. Lower copper sulphate addition should also be tested to avoid activation of the pyrite/pyrrhotite.

As in Test F3, the addition of collector to the zinc scavenger activated the iron sulphides; 60% of the iron in the zinc rougher tailings was recovered in the zinc scavenger concentrate in five minutes flotation time. The flotation rate of iron sulphides is quite slow for the SKM-1 sample, compared to the GO-1 sample.

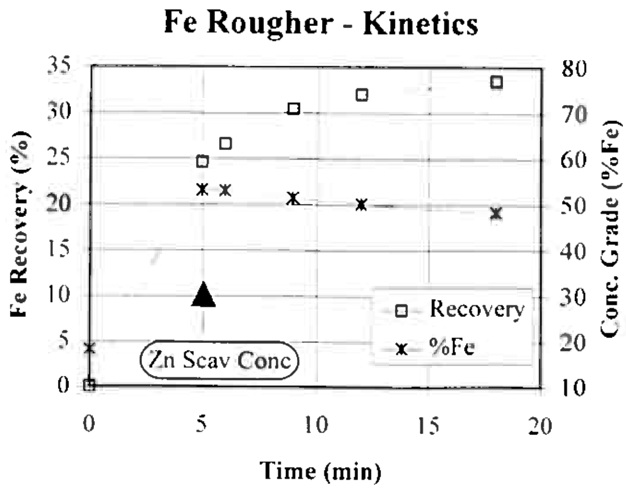


Figure 5: Iron Flotation Kinetics (F4)

Test F5

This test was performed on the GO-1 sample with the objective of improving the copper and zinc concentrates. The results were disappointing for the copper cleaning stages, while zinc recovery provided little problems. No pyrite flotation was performed in this test. The test results are summarized in Table 11.

Table 11: Test Results - Test F5

F5	Grind	Assays				Distribution			
GO-1	K_{80}	Cu	Zn	Fe	Pb	Cu	Zn	Fe	Pb
Copper Concentrate		**19.5**	9.5	19.9	6.1	**83.6**	2.9	5.1	70.9
Zinc Concentrate		0.28	**64.23**	2.07	0.07	5.59	**89.01**	2.42	3.74
Pyrite Concentrate									
Head									

Additional depressant was tested in the primary grind; both lime and sodium-sulphite were increased by 50% for additional iron sulphides depression.

Collector was stage added in the copper flotation and flotation time increased to 7 minutes from 5 minutes. While initial collector addition was halved, total collector addition was 50% higher than in Test F3. These changes resulted in a marginal reduction in the zinc recovery in the copper rougher concentrate, but a reduction in iron recovery of approximately 50%. However, copper was lost as well; rougher recover dropped as well, from 87.7% to 86.5%, albeit at a significantly higher grade of 16.3% Cu.

Copper losses were significant in all cleaning stages, except cleaner 1. The dextrin addition in the second cleaner proved too high and depressed all sulphides. Copper recovery dropped to 36.4% in this stage and was further reduced in the third cleaner by more dextrin. Finally, in the fourth cleaner, the $ZnSO_4$/NaCN mixture killed flotation and virtually nothing floated. Obviously depressant addition will need to be fine-tuned.

The objective in the zinc flotation was to increase rougher zinc recovery, by stage adding collector and longer flotation time in the zinc rougher. Flotation time was tripled, from 3 to 9 minutes, based on visual observation. The zinc recovery in the rougher stage was improved to 97.6% of the rougher feed, compared to 93.2% in Test F3. This also resulted in an increase in iron and copper recovery. Without additional depressants in the cleaning stages, these higher levels of contaminant carried through in the cleaning stages.

Compared to Test F3, the first cleaner produced a concentrate grade of 64.2% Zn at a 1.5% higher stage recovery. However, this was at a penalty of doubling copper and iron contamination to 0.28% Cu and 2.07% Fe.

Longer flotation times at minimal collector addition will be the key to optimizing zinc concentrate.

Test F6

Test F6 was run on the SKM-2 sample, which has a higher zinc value, but lower copper value than SKM-1. The test results are summarized in Table 12.

Table 12: Test Results - Test F6

F6 SKM-2	Grind K_{80}	Assays						Distribution					
		Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Copper Concentrate		**25.2**	11.6	27.0	0.2	5.4	93.0	**80.5**	18.9	2.6	11.9	45.4	13.3
Zinc Concentrate		0.27	**32.50**	24.10	0.08	0.62	22.80	1.02	**62.69**	2.77	5.59	6.24	3.88
Pyrite Concentrate													
Head		1.01	1.98	33.29	0.05	*0.38*	*22.50*	100.0	100.0	100.0	100.0	*100.0*	*100.0*

Grinding conditions for this test were not changed from those used in Test F4, on the SKM-1 sample.

The collector in the copper flotation was changed from a mix of A3418 and R208 to a single collector CA821. The objective was to determine the selectivity between copper and lead for this collector. Roughing was performed in two stages with stage addition of collector. Flotation time was increased by 40%. Rougher copper recovery was lower: 88.4% compared to 93% in Test F4. At the same time, lead recovery was also lower, 23.1% compared to 70% in Test F4. Zinc recovery was high in the copper rougher, at 22.7%, compared to 11% in Test F4, while iron recovery was low. These comparisons are limited, as the two tests were performed on different samples.

Copper concentrate upgrading yielded a 25.2% Cu grade, at a loss of 7.9% of the copper recovery. Zinc depression was poor; the concentrate still contained 11.6% Zn. Dextrin was tested as a depressant in cleaning stage 3, but the dosage target was too high and insufficient concentrate was collected to make assaying worthwhile.

Collector addition in the zinc rougher was reduced significantly from previous tests, with the objective of reducing iron recovery. This proved very effective; only 3.3% of the iron in the zinc rougher feed was recovered, compared to 54% in Test F4. Zinc rougher recovery suffered somewhat to 83.7%, compared to 86.1% in Test F4. The rougher grade was 30.6% Zn, which was upgraded only marginally to 32.5% after three cleaning stages. Iron depression was poor, despite sodium sulphite addition in cleaners 1 and 3.

The CA821 collector appears to be selective against lead, but copper recovery did suffer and zinc was activated. The Flex-31 collector may be too strong for a selective recovery of zinc relative to iron sulphides. Collectors A3418 and R208 should be tested in the zinc flotation.

Zinc depressants will be needed in the copper cleaning to lower the zinc content of the copper concentrate. Higher sodium-sulphite addition levels will need to be tested in zinc cleaning to depress the iron sulphides and improve zinc concentrate grade beyond 32.5% Zn. Microscopic examination will be required to determine if liberation is adequate in the two concentrates.

Test F7

Test F7 was performed on the SKM-2 sample with the objective of optimizing the flotation conditions. The test results are summarized in Table 13.

Relative to Test F6, on the same ore sample, grinding conditions were maintained the same, but copper aeration was eliminated.

Table 13: Test Results - Test F7

F7 SKM-2	Grind K_{80}	Assays						Distribution					
		Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Copper Concentrate		**22.4**	4.89	30.8	0.15	3.73	137	**83.8**	8.5	3.3	10.0	35.4	21.9
Zinc Concentrate		1.45	**56.4**	9.24	0.21	1.42	66.0	4.19	**76.3**	0.77	11.0	10.4	8.2
Pyrite Concentrate													
Head		0.97	2.07	*33.5*	0.053	*0.38*	*22.5*	100.0	100.0	100.0	100.0	100.0	100.0

In the copper flotation, the CA821 was replaced by the collector combination A3418/R208, while collector addition rate was lowered. The net effect of these changes was a slight drop in copper rougher recovery from 88.4% to 85.4%; while iron and especially zinc recovery dropped. Sodium sulphite was used in the first two stages to depress iron and copper concentrate grade increased to 24.7% Cu at a recovery of 82.3%. A mix of $ZnSO_4$ and NaCN was used in cleaners 3 and 4 to depress zinc. Despite the low addition rates of 10 and 20 g/t copper losses were substantial with marginal depression of zinc.

The elimination of the aeration is likely beneficial in the reduction of zinc floating in the copper circuit. The drop in collector created a high rougher grade (18.2% Cu) at a lower recovery.

In the zinc roughers, a longer flotation time, using higher dosage rates of a weaker collector (R208 vs Flex-31) and higher pH was treated. The stage recovery of zinc improved to 91.5%, from 83.7% in Test F6. In the cleaners, higher dosage rates of sodium sulphite with higher pH values were tested for iron depression. This resulted in a zinc concentrate of 56.4% Zn at a stage recovery of 86.8%.

Test F8

This is the only test performed on sample GO-2, which contains the highest values of copper and zinc of the four samples tested. The results are summarized in Table 14.

As in Test F7, the aeration following grinding was eliminated with no discernible effects. The reagents added to the grind were reduced, based on the high pH after grinding in Test F5.

Table 14: Test Results - Test F8

F8 GO-2	Grind K_{80}	Assays						Distribution					
		Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Copper Concentrate		**22.8**	10.1	21.5	1.70	4.59	257	**87.3**	7.20	20.6	53.4	73.9	68.4
Zinc Concentrate		0.33	**62.1**	2.91	0.096	.23	54.8	2.38	**83.3**	5.25	5.65	7.0	27.4
Pyrite Concentrate													
Head (Calc. *Assay*)		3.19	17.1	12.7	0.39	*0.76*	*46.0*	100.0	100.0	100.0	100.0	*100.0*	*100.0*

The copper rougher conditions were not changed from those used in Test F5, on the GO-1 sample. Rougher concentrate grade was a high 19.1% Cu at 91.3% recovery. Based on the high grade, rougher recovery can likely be improved.

In copper cleaning, pH was increased initially to reject iron, increasing the grade to 25% Cu at 77.7% copper recovery. Under the microscope large particles, believed to be sphalerite, were visible with copper inclusions. It was decided to regrind the second stage copper concentrate and to use the $ZnSO_4$/NaCN mix to depress zinc. As with previous tests, the zinc depressant is a strong copper depressant and copper recovery dropped substantially. The large particles did not grind as expected and upon further examination are now believed to be mica particles.

Overall, the copper is slow floating and flotation times will need to be increased to improve recovery. Zinc depression is difficult and different collectors may need to be tested, as well as lower addition rates.

The zinc flotation produced good results, producing a 60.5% Zn concentrate, after two cleaning stages, with a 96% stage recovery.

Test F9

This test was run as an optimization test on the SKM-2 sample. The objective was to increase recovery in the copper and zinc flotation. The results are summarized in Table 15.

Table 15: Test Results - Test F9

F9 SKM-2	Grind K_{80}	Assays						Distribution					
		Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Copper Concentrate		**30.9**	2.8	30.7	0.1	?	266	**74.4**	3.5	2.3	2.3	?	29.2
Zinc Concentrate		45.0	**12.1**	0.11	1.65			1.65	**28.48**	0.47	2.64		
Pyrite Concentrate													
Head		1.03	2.00	32.40	0.05	*0.38*	*22.5*	100.0	100.0	100.0	100.0	*100.0*	*100.0*

To increase copper recovery, the depressant levels in the primary grind were reduced. As both tests F9 and F10 were run concurrently, the sodium sulphite levels in the two tests were reduced from 750 g/t to 250 g/t and 500 g/t respectively. The objective was to maximize copper recovery in the copper roughers and depress the iron sulphides and zinc sulphides in the cleaning stages. However, once activated, the iron sulphides proved difficult to depress and the sodium sulphite proved to be a less effective depressant in the cleaner circuit than in the grinding mill.

Due to the lower depression of iron sulphides in the grinding mill, collector in the copper roughers was lowered to avoid too much recovery of the iron. Copper recovery in the rougher flotation did improve to 90.9%, relative to 85.4% in Test F7. However, the grade dropped from 18.2% Cu to 6.2% Cu with a proportionate increase in the iron and zinc recovery.

Iron depression in the copper cleaners using sodium sulphite was poor. Only after increasing the dosage to 900 g/t and increasing the pH to 11.5, in cleaner stage 2, did the grade improve to 16.0% Cu. In the third cleaner a new depressant, QHS, was tested. Despite the low addition rate of 30 g/t, iron as well as zinc depression was significant; although at a loss of copper. Grade increased to 30.9% Cu, but with a loss in recovery to 74.4%.

As a result of the higher recovery of the zinc in the copper roughers, the feed to the zinc flotation contained only 75.9% of the zinc in the feed (87.9% in Test F7). To maximize zinc recovery, additional $CuSO_4$ was used. Flotation time was increased while collector addition was maintained to the level of Test F7. Despite this, zinc rougher recovery was disappointing at 81.3%, relative to 91.5% in Test F7. An additional 10 g/t of collector in the zinc scavenger managed to recover only an additional 4.7% of the zinc.

Various depressants were used in the zinc cleaning; sodium sulphite, elevate lime to pH=12 and QHS. All reduced iron values in the concentrate, but also depressed the zinc. Overall concentrate at 45% Zn and 28.5% recovery is far worse than obtained in Test F7.

It is postulated that the lower depression levels require higher dosage rates of collector, as collector is consumed by the iron sulphides. The lower zinc recovery in the cleaners is then a result of poor collection relative to the minerals which are depressed.

Test F10

Test F10, again on the SKM-2 sample, was operated simultaneously with Test F9 and used a similar reagent scheme, based on observations in Test F9. The results are summarized in Table 16.

Table 16: Test Results - Test F10

F10	Grind	Assays						Distribution					
SKM-2	K_{80}	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Copper Concentrate		**23.4**	3.6	31.0	0.1	8.1	155	**61.9**	4.7	2.4	4.4	17.8	5.8
Zinc Concentrate		0.81	**36.50**	16.10	0.12			1.12	**24.5**	0.66	3.21		
Pyrite Concentrate													
Head		0.99	2.04	33.56	0.05	*0.38*	*22.5*	100.0	100.0	100.0	100.0	*100.0*	*100.0*

The sodium sulphite addition rate in the grind was increased to 500 g/t.

The higher level of iron depression allowed for an increase in collector addition, which improved copper rougher recovery by 1%, to 91.9%. Due to the relatively low effectiveness of sodium sulphite in depressing iron, in Test F9, QHS was tested as iron depressant. This depressant is very effective, but dosage rates are difficult to gauge visually and copper depression was higher than anticipated, dropping copper recovery in the first cleaner stage to 76.2%. Lower stage additions coupled with collector addition may be effective in depressing the iron, while maintaining copper recovery.

Raising the pH to 12 was very effective in dropping the free iron sulphides, however, at a loss of copper recovery. Future testing should address iron depression by pH only in the copper circuit.

Zinc rougher recovery was disappointing despite virtually identical conditions in Test F7. The major difference was in the 50% higher $CuSO_4$ addition. The higher weight recovery in both tests F9 and F10 indicate that gangue minerals and iron sulphides have been activated by the higher dosage of the zinc activator. Future testing should include lower $ZnSO_4$ addition rates.

Test F11

Test 11 was a repeat of Test F7, with the objective to produce an iron sulphide concentrate for gold distribution analysis. While conditions were kept virtually identical to Test F7, the results were very disappointing.

Table 17: Test Results - Test F11

F11 SKM-2	Grind K_{80}	Assays						Distribution					
		Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Copper Concentrate		**13.0**	5.13	36.4		4.3	85.7	**80.0**	15.5	6.80		68.6	23.4
Zinc Concentrate		11.8	**31.9**	17.2		2.32	57.20	6.10	**8.00**	0.30		3.10	1.30
Pyrite Concentrate		0.090	2.13	**43.9**		0.17	9.30	4.90	54.7	**69.6**		21.3	19.7
Head (Calc. *Assay*)		1.00	2.04	33.0		*0.38*	*22.5*	100.0	100.0	100.0		100.0	100.0

Grinding conditions were identical to Test F7.

Copper flotation conditions were identical to Test F7, with the exception of the rougher flotation time, which was increased by 50% in an attempt to improve copper recovery.

While no additional copper was recovered, a significantly higher weight, zinc and iron was present in the copper rougher concentrate. Whereas two stages of cleaning was sufficient in Test F7, the two cleaning stages in Test F11 increased the concentrate to only 13.0% Cu. The differences between the two tests is likely the result of different froth conditions (higher frother dosage in Test F11) and differences in froth pulling rate between the two technicians.

Zinc recovery in the zinc flotation stage was very poor, despite identical flotation conditions as in Test F7. Initially the poor results were blamed on errors in reagent mixing/addition and the test was repeated in Test F12. In Test F12, however, zinc recovery at 200 g/t of $CuSO_4$ was found to be poor, and addition had to be tripled before sufficient recovery was obtained. Thus, despite the storage of the test charges in the freezer, the chemistry of the ore had changed sufficiently to affect flotation behaviour.

An iron float was performed on the tailings of the zinc flotation to determine gold and silver recoveries in this product. A cleaning step was included, which proved to be of marginal benefit. Due to the low zinc recovery in the zinc circuit, the iron concentrate contained 54.7% of the zinc, and the assayed gold and silver values are of limited value. Iron recovery in the rougher stage was 87.8% of the iron flotation feed, with a grade of 43.9% Fe.

Test F12

This test was a repeat of Test F11. There was a concern that the poor results in Test F11 was a results in procedural errors by the second technician. Test F12 was performed by the same technician used in Test F7.

Table 18: Test Results - Test F12

F12 SKM-2	Grind K_{80}	Assays						Distribution					
		Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Copper Concentrate		**27.4**	4.64	29.6				**48.9**	4.10	1.60			
Zinc Concentrate		5.40	**45.9**	12.2				17.7	**76.0**	1.20			
Pyrite Concentrate		0.13	0.11	**45.1**	.037	.24	12.8	7.90	3.40	**81.6**	45.8	39.1	35.2
Head (Calc. *Assay*)		1.04	2.08	34.2	*.05*	*0.38*	*22.5*	100.0	100.0	100.0	100	100	100

Copper rougher recovery, in this Test, was poor at 66.7%. Similarly, copper losses were relatively high in the cleaning stages, compared to Test F7 (see also Figure 6). Insufficient collection is a likely explanation, possibly due to adsorption by the iron sulphides, as evidenced by the higher iron recovery in the rougher stage, compared to F7.

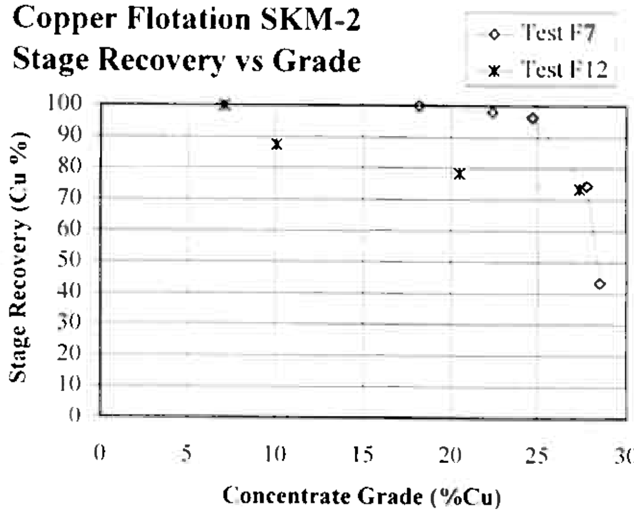


Figure 6: Copper Grade-Recovery (F11 compared to F7)

In the zinc flotation, the addition of $CuSO_4$ had to be tripled, compared to Test F7, to activate the zinc. Even at this higher rate, zinc floated slowly and the zinc scavenger concentrate was included with the rougher concentrate to ensure adequate zinc recovery; thus increasing flotation time by 50%. Even with these changes, rougher zinc recovery was low at 82.7%, relative to 87.9% in Test F7. Upgrading of the zinc was similar between the two tests. A concentrate with 45.9% Zn grade was obtained at 91.9% stage recovery.

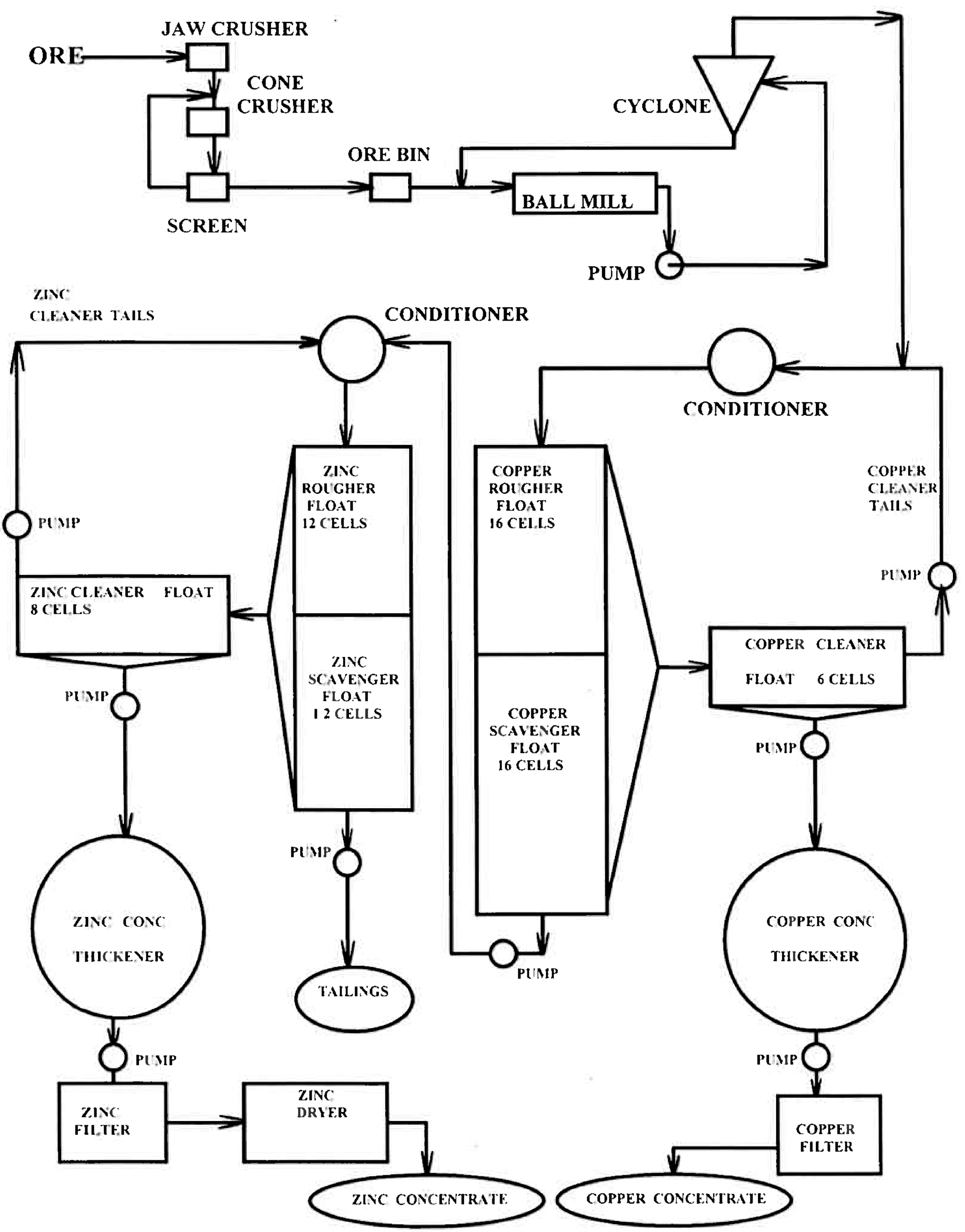
In the iron flotation, as with Test F11, the cleaning stage adds little value. The iron rougher recovered 92.4% of the iron in the zinc flotation tailings, at a grade of 45.1% Fe.

All indications, in Tests F11 and F12, suggest that some chemical change occurred to the test charges of the SKM-2 ore in the freezer. This does not show up in higher acidity values, as pH readings are similar as is lime consumption, between Test F12 and Test F7. The lower responsiveness of the copper and the higher dosage rates of $CuSO_4$, with slower zinc flotation kinetics, does point to an increase in reagent adsorption by the iron sulphides, likely as the result of surface oxidation.

Appendix 1

Laboratory Test Reports

FIGURE 4- GRONG MILLING FLOWSHEET



APPENDIX No.1

Ore Deposits In Nord-Trondelag, Norway by NGU
and
Geology Of The Skiftesmyr and Godejord Deposits
by Dr. Ing. Arne Reinsbakken

A.1

ORE DEPOSITS IN NORD-TRØNDELAG, NORWAY

PROPERTY	COMMODITY	LOCATION	OWNERSHIP	RESERVES (tonnes)	GRADE	COMMENTS
Joma	Copper Zinc	20 km east of Røyrvik, 150 km by road to the coastal town Namsos. Grong district.	Norsulfid AS	20 000 000 (before production)	1.50 % Cu 1.46 % Zn	Status: Mine. Discovered in 1911. Started production in 1972. Probably exhausted and closed by the end of 1996. Total production by the end of 1994; 9.6 mill tonn. Large proportion of original reserves consisted of pyritic ore. Regional helicopter-borne geophysics available.
Gjersvik	Copper Zinc	5 km west of Røyrvik. Grong district	Norsulfid AS	1 600 000 (before production)	1.60 % Cu 0.90 % Zn	Status: Mine. Discovered in 1909. Started production in 1993. Satelite-mine to Joma. Planned production about 0.5 mill.tonn. Regional helicopter-borne geophysics available.
Skorovatn	Copper Zinc Pyrite	5 km WSW of Tunnsjoen, 100 km by road to the coastal town Namsos. Grong district	Open	1 300 000	1.15 % Cu 1.80 % Zn	Status: Former producer. Discovered in 1873. Operated during the years 1952-1984 with total production of 5.6 mill. tonn, until 1972 mostly pyritic ore. Regional helicopter-borne geophysics available.
Visletten	Zinc Copper	8 km from the road. Grong district.	Open	800 000 (massive ore) 1 100 000 (incl. disseminated ore)	0.92 % Cu 3.86 % Zn	Status: Prospect. Intensive exploration, geology, geophysics and drilling (18 holes) in the 70'ies. Regional helicopter-borne geophysics available.
Skiftesmyr	Copper Zinc	15 km ENE of Grong, 65 km by road to the coastal town Namsos, Grong district	Nord-Trøndelag Fylkeskommune	3 500 000	1.16 % Cu 1.79 % Zn	Status: Prospect. Intensive exploration in the 70'ies and early 90'ies, geology, geophysics and drilling (76 holes (1973-92), totalling 13000 m). Regional helicopter-borne geophysics available.
Godejord	Zinc Copper Gold	12 km E of Grong, 65 km by road to the coastal town Namsos. Grong district.	Nord-Trøndelag Fylkeskommune	330.000	0.5 % Cu 3 % Zn 0.2 % Pb	Status: Under active exploration. During 1974-94, geology, gephysics, drilling (45 drill-holes). Richest ore zone; 100 000 t with 0.8 % Cu, 7 % Zn, 0.3 % Pb and 1 g/t Au. Gold-enriched VMS or epigenetic deposit. Regional helicopter-borne geophysics available.
Fimbu	Zinc Copper	24 km E of Grong, 80 km by road to the coastal town Namsos. Grong district	Open	~250.000	0.34 % Cu 3.15 % Zn	Status: Prospect. Discovered in 1914. Irregular exploration 1968-1985, geophysics, geology and drilling (27 holes). Regional helicopter-borne geophysics available.
Fremstfjell	Molybdenum Copper	32 km east of Grong. Grong district	Open	?	see comments	Status: Low-grade porphyry-type Cu-Mo deposit. Reserves unknown. Regional helicopter-borne geophysi available.
Skrattåsen	Zinc Lead Copper Silver	7 km NE of Steinkjer (harbour). 1.5 km from railway.	Falkhammar AS J. Råen	80 000	7 % Zn 2 % Pb 1 % Cu 70 g/t Ag	Status: Prospect. Discovered in 1880. Small-scale production at intervals between 1890 and 1927. Diamond drilling and geophysical surveys 1973-1980. Approximate grade and reserve estimates. Regional helicopter-borne geophysics available.

THE MØKLEVATNET PROJECT

Skiftesmyr and Godejord deposits

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MØKLEVATNET PROJECT:

The Skiftesmyr and Godejord Deposits

The Grong District covers a 3000 km^2 area in central Norway, bounded to the East by the Swedish border, to the west by the River Namsen, to the south by the Sandøla Valley and to the north by the large Lake Namsvattnet and the Børgefjell national park.

The Grong District is underlain dominantly by Lower Palaezoic metavolcanic, metasedimentary and intrusive rocks, of Mid. Ordovician age, that comprises the Gjersvik Nappe, part of the larger Køli Nappe of the Upper Allochthon tectanostratigraphy within this part of the central Scandinavian Caledonides. These nappe sheets contain thrust emplaced terrains that are far transported slices of volcanic, intrusive and sedimentary rocks of ocean floor, rifted-arc and back-arc marginal basin infill that have been thrust eastward onto the Baltoscandinavian basement (Baltic Shield).

The Gjersvik Nappe consists mainly of a magmatic complex that is dominated by a mafic volcanite and subvolcanic intrusive complex which are overlain by sediments that have been derived, for the most part, from erosion of the magmatic arc during a period of extensive uplift and erosion.

Mafic volcanites dominate the island arc - rifted arc complex with felsic volcanites forming only a minor component. The felsic volcanites occur at several stratigraphic levels, often associated with massive sulphide mineralizations that are generally overlain by thin layers of banded iron formation (BIF), which regionally can form extensive marker horizons throughout the district.

The whole sequence has undergone extensive folding and shearing deformation related to thrusting and Nappe emplacement. The rocks are generally moderate to strongly sheared (well foliated) and have undergone Upper Greenschist, grading into Lower Amphibolite facies metamorphism within the western part of the district.

Volcanic hosted massive sulphide (VHMS) mineralization is common in the volcanic complex, occurring at several stratigraphic levels. Although most of the deposits are small (< 1M metric tons = tonnes), several major deposits occur in the district (Joma = 20M, Skorovass = 10M and Skiftesmyr = 4M tonnes). Three deposits have been exploited or are currently under production. Skorovass was mined from 1952 to 1984, divided into 2 periods; 1952-76, 3.9M tonnes, grading 39.1% S and 1.1% Cu of pyrite fines mined for the production of sulphuric acid, and 1976-84, 1.7M tonnes grading 1.15% Cu and 2.71% Zn mined for production of Cu and Zn concentrates. When production started at Joma in 1972, the total reserves were calculated at c. 20M tonnes of massive and disseminated ore containing 32% S, 1.3% Cu, 1.7% Zn with only minor amounts of Pb and recoverable Ag and Au. Production at Joma is forecast to stop around mid summer next year (1997). At the end of 1994, 9.6M tonnes of ore has been mined at Joma, grading 1.50% Cu and 1.46% Zn. The Gjersvik deposit (c. 1.6M tonnes, grading 1.60% Cu and 1.0% Zn), which lies 25 km to the west of Joma, is currently being mined and the ore transported to Joma for processing. Mining starterd in 1993, based on c. 500 000 tonnes grading 2.15% Cu and 0.60% Zn.

The Møklevatnet Area

The Skiftesmyr and Godejord deposits occur in the Møklevatnet area at the SW corner of the Grong District, c. 20-30 km east of the Grong community centre. The Skiftesmyr deposit occurs at roughly the same stratigraphic level in the Gjersvik Gp. magmatic complex as the Skorovass and Gjersvik deposits to the north. These two deposits are overlain by a pronounced layer of felsic volcanites. The Skiftesmyr deposit occurs in a slightly different environment and is overlain by a thicker sequence of mixed felsic/mafic tuffs and/or volcaniclastics which grades upwards into more mafic dominated tuffs and massive to pillowed lava flows. These rocks are metamorphosed under Upper Greenschist to Lower amphibolite facies conditions. Regionally, the rocks show varying degrees of shearing and are for the most part moderately foliated, and in some areas volcanic structures and textures are preserved. On approaching the massive ore zone, however, the country rocks become extreamly sheared.

Skiftesmyr

The orebody at Skiftesmyr consists mainly of Zn-Cu rich massive pyritic ore that occurs as thin layers or as a continuous series of ore lenses forming a relatively thin, plate-like orebody. The thickness of the ore zone varies between 2-20m, with 4-6m being most common. The massive ore contains many fragments of country rock near its contact with the host rock, especially within the upper and easern parts of the orebody. These fragments appear to be remnants of fold hinges that have been ripped apart and now occur as loose fragments, floating within the strongly sheared orebody.

The orebody

The orebody at Skiftesmyr consists dominantly of massive sulphide layers and lenses enclosed within a quartz-sericite, albite and chlorite rich schistose country rocks that contain variable quantities of disseminated and veined sulphides, dominantly pyrite. Minor quantities of chalcopyrite (cpy) and sphalerite (sl) are also present within these altered and sheared rocks, cpy being mostly confined to the darker chlorite rich rocks and sl in the pale, quartz-sericite and albite rich rocks. The massive ore is dominantly pyritic with varying subordinate quantities of cpy and sl and minor amounts of pyrrhotite (po). The Ag and Au mineralogy at Skiftesmyr has to date not been studied and the distribution of these precious metals within the orebody is little known because of the sparse amount of analytical data presently available. The main gangue minerals are quartz, chlorite and calcite.

Copper and Zinc are antipathetically related to each other and show a clear zonal distribution within the massive pyrite orebody, which is typical for most VHMS deposits. The Cu rich ore dominates within the eastern and upper levels of the orebody and Zn rich ore is consentrated in the western part and at depth in the orebody. The deposit is open at depth and towards the west, where the orebody also becomes distinctly thinner.

The massive pyritic ore at Skiftesmyr is a compact, homogeneous ore type of medium-grained size, generally in the range of 1-5mm. Although no detailed mineralogical study has been done here, the individual pyrite grains appear to be granular in nature haveing well developed grain boundaries and are relatively clean with few sulphide inclusions.

Chalcopyrite and sphalerite usually form grains at the boundaries between the larger pyrite grains. Pyrrhotite occurs only as a minor constituent within the massive ore and is found mainly along shear planes and late fractures that cut across the pyrite grain boundaries. Thus, mineral separation of the Skiftesmyr ore should give relatively clean products and should not cause great problems, as did for i.e., the extremely fine-grained, complex pyritic ores from Skorovass and Joma.

Host rocks

The massive orebody is enclosed in intensely altered rocks that adjacent to the orebody are strongly sheared and schistose. On the south side of the orebody, the FW rocks are dominated by pale coloured, quartz-sericite and albite rich rock carrying large quantities of disseminated pyrite and quartz-pyrite veins. These grade into darker, chlorite rich rocks that contain minor quantities of pyrrhotite. The altered rocks are arranged in a zonal pattern around the orebody. Quartz-sericite rich rocks occur adjacent to the orebody and grade outwards away from the massive orebody into albite- and chlorite-rick rocks. Further into the FW, the chlorite-rich rocks grade into more normal chloritic greenstones rich in epidote and carbonate and with minor disseminations of po and py. Volcanic structures such as pillows are present in these rocks.

Near the surface, within the immediate HW to the orebody, the country rocks are visibly paler in colour, harder and are richer in quartz-albite with less quartz-sericite and chlorite. Deeper within the orebody, along the HW, the rocks are more schistose and become richer in quartz-sericite and chlorite.

On surface, to the NE of the main showing along the western edges of the Storedalen valley, pale coloured quartz-sericite and albite rich rocks occur that contain zones rich in pyrite disseminations and quartz-pyrite veining surrounded by darker chlorite rich rocks with only minor pyrite. These intensly altered and strongly sulphide impregnated rocks are thought to represent the feeder zone to the massive sulphide ores at Skiftesmyr that lie to the W and SW.

This N-NE trending zone of strong pyrite disseminated rocks corresponds with a strong EM anomaly found in the overburden covered lowland area to the N of the main showing. This anomali can be traced for several km to the NE.

North of the surface expression to the massive ore horizons, the HW rocks to the orebody consists of a of sequence of variably layered, massive felsic volcanites and/or intrusive sills (?) that are interlayered with fine-laminated felsic to mafic tuffs or tuffites? Some of these felsic layers show clearly turbiditic/ volcaniclastic textures (fine- to coarse-grained beds) and soft sediment slumping folds are observed. The quartz and albite rich felsic layers are generally variably magnetic in nature. Some extremely magnetic layers have been observed in drill core. The high magnetite contents in certain quartz rich layers may be derived from reworking of earlier magnetite bearing felsic volcanic rocks. This layered felsic tuff/ volcaniclastic sequense grades upwards into mafic dominated tuffs with minor felsic layers, which in turn grades stratigraphically upwards into a mafic massive and pillowed flow sequence.

The layered felsic-mafic tuffite/volcaniclastic and overlying mafic lava unit that lies to the north of the orezone is interpreted to be younger than the massive sulphide mineralization and associated altered HW and FW mafic volcanic rocks that host the massive orebody at Skiftesmyr. On surface, near the main showing, massive sulphide ore is in contact with irregular lenses (overlain by), bands and fragments of magnetite bearing quartzites (recrystallized chert), which are interpreted as silica rich exhalites. Magnetite-bearing chert, with minor amounts of po and py, occur as layers and lenses of varying thickness and extent, at the contact between the two main rock units at Skiftesmyr. In simplest terms, the altered mafic volcanic complex forms the stratigraphic FW and the mixed felsic/mafic tuff-volcaniclastic complex forms the HW to the massive orebody.

Late, feldspar-phyric felsic and pale green gabbroid dykes are found cutting the ore zone and the overlying tuff/volcaniclastic complex.

Structures

The orebody occurs partially within a major shear zone. The massive pyritic ore, containing numerous fragments of folded country rocks, occurs as parallel ore layers and lenses within what appears to be a major shear zone along the HW side, throughout the whole length of the orebody. This is well demonstrated in most vertical sections. The shear zone appears to be an early structure and the ore zone plunges steeply to the NW within this structure. The distribution of Cu- and Zn-rich zones within the orebody and the ore thickness also appears to plunge in a NW direction, suggesting that they also may be related to later folding and shearing deformation.

The surface geological map over Skiftesmyr shows that the area has been folded into a major open flexure. The rocks to the NE and E of the main showing trend to the north and dip steeply to the west and rocks within the ore zone and to the west, trend roughly E-W and dip steeply (60-65°) to the north. This is a late crenulation type fold having NE steeply plunging fold axes.

Ore reserves and production plans

The earliest ore reserve calculation quoted for the Skiftesmyr deposit was carried out in 1977 by Grong Gruber A/S and gave a geological ore reserve of 3.5M tonnes grading 1.16% Cu and 1.79% Zn. Later drilling has not changed to any degree this figure from 1977, as much of the later drilling (1980-92) was confined to filling in details within the upper levels of the orebody.

In 1992, Norsulfid A/S presented an ore reserve calculations for the Skiftesmyr deposit which included all drilling done on the deposit up to 1992 (Norsulfid A/S company report to the Mining Commission, BV 2882). Plans for both underground and an open pit mining was also presented in 1992 (Norsulfid A/S company report to the Mining Commission, BV 2883):

1) cut-off 1%Cu equivalent: total 2 746 470 tonnes grading 1.23% Cu, 1.86% Zn, 11.37 ppm Ag, 0.35 ppm Ag and 37.52% S. The calculated ore zone has a strike length of 400m and a vertical length of 400m. The thickness of the ore zone varies between 2-21m.

2) cut-off 2% Cu equivalent: total 1 759 417 tonnes grading 1.38% Cu, 2.13% Zn, 12.99 ppm Ag and 0.37 ppm Au.

3) According to underground mining plans reported by Norsulfid A/S (report BV 2883), a total of 2 684 000 tonnes of ore was planned to be taken out (cut off 1% Cu equivalents), grading 1.08% Cu, 1.63% Zn, 8.65 ppm Ag, 0.31 ppm Au and 34.6% S. The reduced tonnage and grades quoted here results from ore being tied up in pillars and from waste rock dilution.

Godejord

The Godejord deposit lies c. 3-4 km SSW of Skiftesmyr, in a slightly different geological environment. The rocks at Godejord consist of a complex dominated by mafic volcanites and minor tuffite/ sediments that have been strongly deformed and metamorphosed under Lower Amphibolite facies conditions. The whole sequence appears to be inverted at Godejord. Mafic volcanites dominate. The lower part of the sequence is dominated mostly by thick layers of massive flows, dykes and subvolcanic high level doleritic intrusions or sills and the upper part by pillowed flows. These two units are separated by a very persistent BIF/tuffite horizon that forms a prominant marker horizon throughout the district. At Godejord, this unit forms the HW to the main (East) orebody.

This sequence of mixed mafic volcanites/ tuffitic rocks is distinctly different from those found at Skiftesmyr. Trace element characters of the volcanic rocks are distinctly different from those found at Skiftesmyr, and of those in the Gjersvik Gp. in general. It has been suggested (Grenne and Erichsen, 1996) that the Godejord volcanites may in fact be older than the Gjersvik Gp. rocks, and possibly of late Proterozoic (Cambrian?) age. The Godejord volcanites may be related to a belt of amphibolitic greenstones that host a major Fe deposit (BIF) found to the west of the Grong District.

Only minor intrusive rocks are found at Godejord. Thin feldspar-phyric felsic dykes are present near the ore zone and pale coloured gabbroic bodies are found to the north.

The Godejord ore zone lies at a level in the thick volcanic sequence that is dominated by calc. rich tuffites with iron formations and cherts intercalated with mafic volcanites and minor felsic unites. The total strike length of the mineralized zone is in excess of 2km. However, the most interesting mineralization is confined to a c. 500 m long zone centered around the main Godejord showing. The ore zone has roughly a E-W trend and dips steeply (60-70°) to the north. The thickest part of the orebody appears to plunge steeply to the NE, which is in agreement with interpretations made by Outokumpu OY in 1992 for down-hole geophysics on the whole eastern ore zone.

The East orebody at Godejord lies adjacent to a prominant magnetite-bearing quartzite (recrystallized chert). This silica exhalite unit that forms the HW to the ore, is folded into a tight isocline just west of Godejord, the northern limb of this fold continues for many km to the east. The rocks immediately surrounding this quartzite horizon is strongly sheared and the quartzite is often found as lenses along the strongly sheared extended limbs. Rocks that are in contact with the quartzite are also strongly altered into pale albite-epidote- carbonate rich assemblages that often show zonal arrangements, grading from quartz-sericite-pyrite to quartz-albite through to chlorite-epidote-carbonate (siderite-ankerite-dolomite?) rich rocks trending away from the most intensely altered, central parts of the mineralized zone. Several parallel zone of altered rocks have been noted surrounding the Godejord mineralization.

Host rocks

Interpretation of the host rocks is difficult as many different geologists have logged the drill core over the years. Because of the sheared nature of these rocks, it is difficult to compare the lithological data from drill holes over the whole deposit to the surface geology. The drill holes are dominated by rocks that are strongly banded to laminated, often on a cm-dm scale. These laminated rocks were originally interpreted as tuffites, showing variable contents in felsic to mafic type laminae and layers rich in carbonate, and all posible gradations of these. However, much of these layered units are stongly sheared and some are mylonitic in nature. Much of the carbonate 'tuffites' up in the HW to the ore zone (surface and down to 110-150m above HW of the ore zone) are rich in calcite, ankerite and siderite (dolomite?) porphyroblasts, and some zones rich in large hornblend sheaths are also common. These rocks can also be interpreted as resulting from alteration during a period of intense hydrothermal activity related to the formation of ore mineralization at Godejord.

Below the ore zone, tuffite sequence with felsic layers is more common and an up to 10m thick quartz keratophyre unit, possibly intrusives/dykes are also common.

Greenstones, present as relatively homogeneous fine to medium grained metabasalts, are more common away from the ore zone, both above and below the ore zone. A 35-50m thick sequence occurs 20m below the ore zone. Pillowed and dykes like structure are observed locally within the drill core and are both also observed on surface to the north of the drill sites.

Banded Iron Formation (BIF) horizons occur at several places in the stratigraphy, as thin, partly fine-laminated bands. Pure BIF is most common at levels 190-230m (called the New Godejord zone) and 35-40m above the main ore horizon. Within these zones, the BIF can occur as pure Fe-sulphide, magnetite or pink Mn-rich garnet rich bands. Band thicknesses from mm to several dm are common, often intercalated with layers of pure quartzite (recrystallized chert). Gradations between the 3 types (sulphide-, oxide- and garnet-chert) are common and gradations between pure BIF and various tuffitic rocks is also common, such as felsic to mafic tuffites with varying contents of magnetite, pink garnet and Fe-sulphides and quartz-rich tuffites.

Godejord orebody

The Godejord deposit is a strongly tectonized Zn-Cu-(PB-Ag-Au) mineralization with variable contents of pyrite. The deposit occurs in an area that is strongly covered by overburden and the mineralized zone is uncovered in several small workings over a distance of 1100 m along its E-NE strike direction. Mineralization is concordant with the enclosing strongly foliated rocks that dip steeply (50-75°) to the N-NW. Most of the zone contains relatively weak sulphide mineralization.

Semi-massive to massive mineralization is only found in the eastern part of the zone, around the main showing at Godejord. Here, the ore zone is closely associated to a magnetite-bearing quartzite lens (recrystallized chert), that for the most part forms the HW to the deposit and locally can reach thicknesses up to 10m. The mineralized zone at Godejord is confined to the FW of a prominant quartzite (chert) horizon that trends roughly E-W and dips 60-70° to the north. The orebody is thickest and richest around the main showings (called the John Godejord skjerp) and the most interesting mineralization plunges steeply to the NE.

Ore mineralization ocurs within a zone containing a variety of host rocks ranging from quartz, carbonate, quartz-sericite and actinolite-tremolite rich layers. Pyrite dissemination is most common and quite variable and interlayered with bands of semi-massive to massive pale, honey yellow coloured sphalerite and chalcopyrite rich disseminations in dark hornblend-actinolite rich layers. The individual layers vary from cm to dm in thickness (i.e., 10-30cm thick layers of massive sphalerite). Au mineralization may be found associated with quartzite lenses and layers within the mineralized zone, as is the case to the west of here. Au may have been derived through remobilization from chert/BIF layers that have been tectonically reworked and hydrothermally altered during the period of sulphide deposition.

The richest parts of the deposit occurs within a zone up to 60m long and 15m thick, where grades can reach up to several % Cu, 0.7%Pb, 80 ppm Ag, 5 ppm Au and 25% Zn. This ore type, with relatively high values of Zn, Pb, Au and Ag is somewhat atypical for the Godejord sulphide ore zone. Another anomalous trait of the Godejord deposit is its large quantities of extremely Fe-poor, pale honey-coloured sphalerite.

At depth, below the surface extent of the main showing at Godejord, the mineralized zone becomes more tectonized and strongly sheared, with quartzite lenses and remobilized quartz fragments occurring throughout the ore zone. The mineralized zone appears to form several en echelon ore lenses that are cut by several steeply dipping shear zones.

Deeper within the ore zone, at 150-200m depth in DDH 121,126 and 127, the ore zone is much thinner and more tectonized and irregular in nature. Here, the ore zone consists mainly of weak disseminations of mainly sphalerite and pyrite occurring as irregular slivers and sheared lenses within a breccia-like to irregular bands of quartz-sericite matrix. Bands of pure quartzite are found and bands rich in more or less pure pyite are also common. Sulphide disseminated tuffites interlayed with BIF bands (often garnet bearing) are found in the immediate HW to the mineralized zone. At the FW contact to the ore zone, the mineralization is generally in sharp contact with layered tuffites.

Rich ore, found near the surface around the main showing at Godejord, does not continue down to depth. At c. 250m depth the whole mineralized zone is less than 1m thick and strongly tectonic in nature.

From the East orebody (main showing area), the mineralized zone at Godejord continues to the west for c. 1.5km and for about c. 1km to the east. The thin mineralized zone corresponds to a prominant IP anomally in the strongly overburden cover terrain. To the west, the mineralized zone has a max. thickness of 4-5m consisting of stongly altered quartz-sericite-albite rich rock carrying variable quantities of pyrite and minor sphalerite that occur as dissemination and veins. This sulphide disseminated alteration zone appears to cut through a more or less homogeneous, massive dolerite complex. The orebody beneath the quartzite horizon at Godejord is interpreted as being a calc-silicate skarn mineralization, occurring as actinolite-tremolite, quartz, carbonate and quartz-sericite rich rocks that contain variable disseminations to semi-massive to massive mineralization rich in pyrite-sphalerite-chalcopyrite±galena.

Ore Reserves

An ore reserve calculation was done by Norsulfid A/S (Norsulfid A/S report to Mining Commission; BV 2882) in 1992 for a feasibilitty study for underground drift at Godejord (report BV 2884 for underground mining plans). The reserves were based on underground production taken at levels between the surface (307m) down to the 240m level (300, 280, 260 and 240m levels). Ore reserves for the planned production was calculated at 76 221 tonnes grading 0.76% Cu, 7.76% Zn, 24.47 ppm Ag and 0.83 ppm Au. A 16.5% waste rock dilution factor is included in these figures.

NGU did an ore reserve calculation for Godejord for the North Trøndelag Fylkeskommune in 1996, based on all drill holes (31 DDH) from the deposit. This includes 2-3 new deep hole that intersect the ore zones at a depth of 250m below the main showing. The mineralized zone of interest is called the 'East Orebody' and lies beneath the main showing at Godejord, over a strike length of 500m and down to a depth of 250m (NGU report 96.024). The NGU reserve is quoted in two figures base on a cut-off of ; 1) 1% Cu equivalent, and 2) 2% Cu equivalent:

1) cut-off = 1% Cu equivalent; deposit size, 250 000-300 000 tonnes (150m strike length plus 200m plunge length to depth) grading 0.6% Cu, 4.2% Zn, 0.1% Pb, 15 ppm Ag and 0.4 ppm Au.

2) cut-off = 2% Cu equivalent; deposit size 100 000 tonnes (100m strike length and 100-120m plunge length at depth) grading 0.8% Cu, 6.9% Zn, 0.2% Pb, 20 ppm Ag and 0.8 ppm Au.

References:

Grenne, T. and Erichsen, E., 1996: 3-D modellering, tonnasje- og gehaltberegning av Godejordforekomsten, Grong, Nord-Trøndelag. NGU Rapport nr. 96.024. 41p.

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Stromeyerite and Mckinstryite from the Godejord Polymetallic Sulphide Deposit, Central Norwegian Caledonides

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Trondheim, Norway

The Cu—Ag—S minerals, stromeyerite and mckinstryite, have been found for the first time in a stratabound polymetallic pyritic deposit in the Caledonides of central Norway. The surface specimens examined contained approximately 0.5% Ag, 1.8% Cu, 15.0% Zn and over 10 g/t Au and showed the mineral association pyrite, sphalerite, chalcopyrite, galena, tennantite, bornite, Cu—Ag sulphides, covelline, native Au, a Cu—Sn sulphide, and a new mineral of composition Ag_5CuTeS_2. The Cu—Ag sulphides appear to be replacing preexisting sulphides, with the exception of pyrite and sphalerite. The nature of this replacement is discussed. Analyses, by microprobe, of the Cu—Ag—S phases are reported and compared with published data. The stromeyerite shows an average composition $Cu_{1.01}AgS$, the mckinstryite $Cu_{0.77}Ag_{1.19}S$. Values are reported of the reflectance at 542 nm for both minerals. The data indicate that stromeyerite is optically positive with Rg: 30.7%, Rm: 27.3%, Rp: 25.8% while mckinstryite is negative with Rg: 32.5%, Rm: 31.9%, Rp: 27.6%.

Introduction

Investigations of the mineralogy of a small stratabound polymetallic sulphide deposit occurring in rocks of the lower Palaeozoic (Caledonian) geosynclinal-orogenic belt of central Norway have shown it to be unusually rich in a number of minerals which are normally present, if at all, in only minor amounts in similar ores. These minerals include Cu—Ag-sulphides, an unknown phase of the composition Ag_5CuTeS_2, a Cu—Sn-sulphide, bornite, idaiite, covelline, and native gold.

The present communication is an account of the mode of occurrence, compositions, optical properties and possible origins of the two Ag—Cu—S phases present, mckinstryite and stromeyerite.

The Deposit of Godejord's Prospect

The little showing known as John Godejord's prospect (Oftedahl 1958) is situated on the north side of the valley of the Sanddöla river, a tributary to the larger Namsen river which flows into the Norwegian Sea at Namsos, some 130 km northeast of Trondheim (see map Fig. 1). Fig. 1 shows Godejord's situation in the metamorphic Paleozoic rocks of the Grong district, just to the north of the east-west belt Precambrian gneisses known as the Grong culmination. This belt of gneisses, generally transverse to the axial direction of the Caledonides, separates the Grong district's metavolcanics and metasediments from similar rocks of the Trondheim district. Both the Grong and Trondheim districts are important subprovinces for the occurrence of stratabound polymetallic sulphide ores in the general Caledonian metallogenetic belt. In the Grong area occur the large deposits at Skorovass (ca. 7 mill. tons) and Joma (15—20 mill. tons), both of which are in production. The deposit at Stekenjokk in Sweden (20 mill. tons) occurs in a continuation of Grong-type lithologies across the international border.

The presence of large intrusive masses of both gabbro, trondhjemite and granite of apparently Caledonian age in the Palaeozoic rocks north of the Grong culmination should be noted. In the present context it is of interest

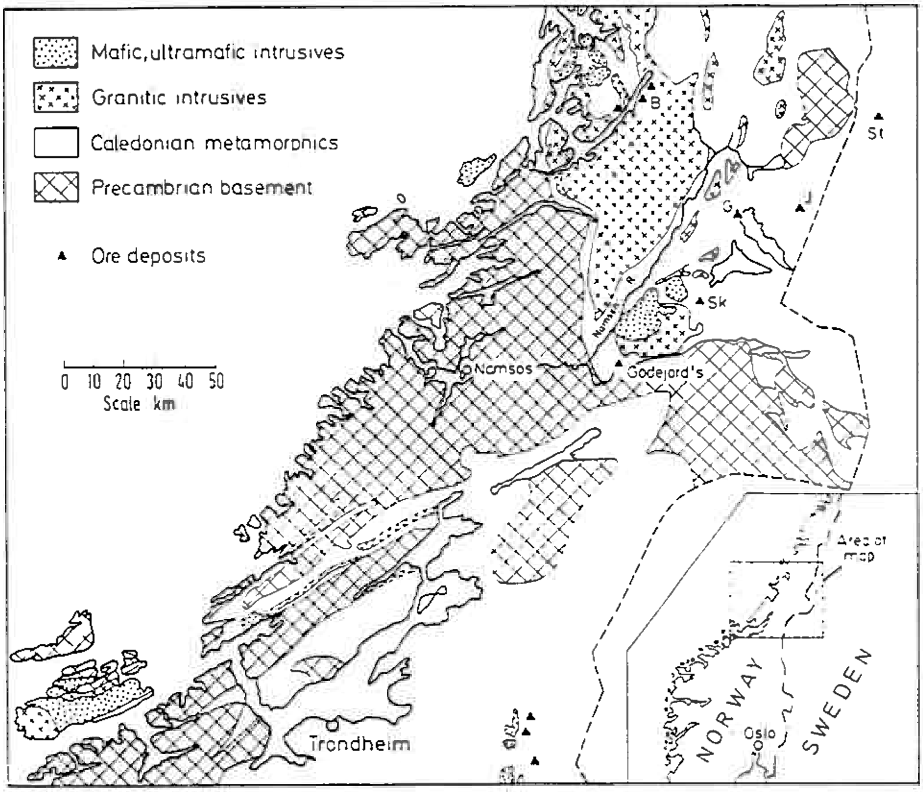

Fig. 1. Simplified geological map of the central Norwegian Caledonides showing the location of Godejord's prospect. After HOLTEDAHL and DONS, (1960). B — Auriferous epigenetic veins in the Bindalen area; G — Gjersvik massive sulphide deposit; J — Joma massive sulphide deposit (in production); Sk — Skorovas massive sulphide deposit (in production); St — Stekenjokk massive sulphide deposit (under development)

that gold and silver-bearing veins of apparently epigenetic-hydrothermal character are spatially related to the largest of the granitic masses in Bindalen (see Fig. 1) and in Svenningdalen, further north.

The area around the prospect is heavily covered and the mineralization is exposed only in series of small, shallow trenches across its strike, which seems to be less than 200 m in length. Widths up to 3 m are exposed in these trenches. The mineralization appears to be concordant with the enclosing metamorphic rocks.

Mineralogy of the Deposit

The sulphide mineralization exposed in the trenches along the outcrop at Godejord's prospect shows a distinct banding parallel to the ore walls, a banding which is defined mainly by variable sulphide-gangue ratios and to a less extent by variations in the relative proportions of the sulphide minerals themselves. Microscopical examination of selected hand-specimens (not necessarily representative of the exposed ore) show the following ore mineral assemblage: pyrite, sphalerite, chalcopyrite, galena, tennantite, bornite, idaite, covelline, Cu—Ag—S minerals, an Ag—Cu—Te sulphide and an Sn—Cu sulphide. The gangue minerals are predominantly quartz, clacite, tremolite and mica. Modal analyses by point counting in a limited number of polished sections showed the following compositional variations:

pyrite	6.0 to 40.0 vol. percent
sphalerite	3.0 to 30.0 vol. percent
chalcopyrite	1.0 to 14.0 vol. percent

galena	0.0 to 3.0 vol. percent	
bornite	0.0 to 0.3 vol. percent	
tennantite	0.0 to 0.2 vol. percent	
Cu–Ag sulphides	0.0 to 4.0 vol. percent	
covelline	0.0 to 0.3 vol. percent	
native Au	present	

A composite sample, composed of half of every hand specimen collected from the outcrop trenches and weighing several kilograms, gave the following analysis.

Cu	1.84%	
Zn	15.34%	
Pb	0.20%	Analyst I. Römme,
Ag	0.44%	Geologisk Institutt, NTH.

Three parallel splits of this sample were analysed for *Au* at the Geological Survey of Norway and gave, respectively 120, 13 and 14 g/t.

The ore shows a typical metamorphic recrystallised fabric, with subhedral to anhedral grains of pyrite, of an average grain size of 0.5 mm, between which occur the softer "matrix" sulphides and the gangue minerals in varying proportions.

The copper-silver sulphide minerals occur in the matrix to the pyrite grains as irregular anhedral areas, either as a filling between these grains, or in a polygonal mosaic along with the other matrix minerals (Fig. 2).

Fig. 2. Godejord ore, showing euhedral pyrite (white, py), subhedral sphalerite (medium grey, sl) anhedral chalcopyrite (off-white, cp) and gangue minerals (dark). Cu–Ag sulphides (light grey) fill in between the other minerals, showing especially sharp, straight boundaries against the pyrite and sphalerite. Plane polarised reflected light, air, 36 ×

Microscopical examination of polished sections of the specimens showed three different phases within the areas of copper-silver sulphides. The most abundant of these phases showed optical properties which corresponded closely with those of stromeyerite, ideal formula, $CuAgS$. This identification was confirmed by X-ray diffraction powder pattern. The second most abundant phase was tentatively identified as mckinstryite from phase-relation considerations and optical properties. Its identity was confirmed by microprobe analysis (see below).

The third phase occurred in relatively small quantities and then only as small, rounded, apparently exsolved, drop-like bodies in the Cu–Ag sulphides. Microprobe analysis showed this phase to be an apparently unknown copper-silver-tellurium sulphide having the compositional formula $CuAg_5TeS_2$. Work is in progress to determine the mineralogical nature of this phase and the results will be reported in a separate publication.

Textures Shown by Stromeyerite and Mckinstryite

The two minerals occur generally intimately associated in irregular patches as part of the matrix to the more euhedral minerals occurring in the Godejord ore (mainly pyrite and, less frequently, sphalerite). The internal textures of the patches are dominantly anhedral, irregular to polygonal, grains of very variable grain size. The relative proportions of the two Cu–Ag–S minerals vary considerably from patch to patch. In some cases only one of them may be present, in others they may be present in more or less equal proportions.

The relative positions of the two minerals in many cases appear to be random, but in a significant number of instances it can be seen that the mckinstryite occupies a position at the borders of the patches, intervening between the stromeyerite and adjacent minerals in a manner strongly suggesting it is replacing the stromeyerite from the grain boundaries inwards (Fig. 3). In many cases long, lancelike protrusions of mckinstryite may penetrate the stromeyerite, indicating that the replacement is partly controlled by the stromeyerite cleavage.

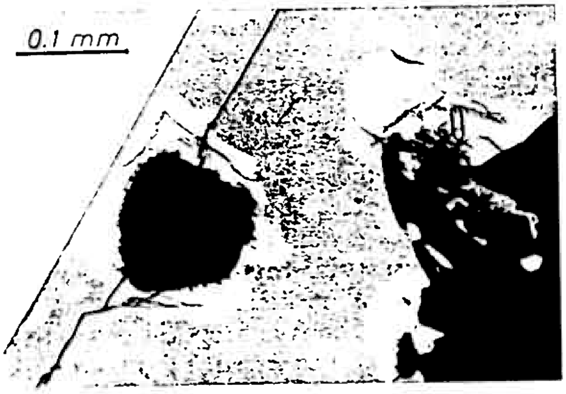

Fig. 3. Mckinstryite (light grey) intervening between stromeyerite (dark grey) and gangue minerals (black) in a manner suggesting it is replacing the stromeyerite from the gangue-stromeyerite contacts. Note the straight unaffected crystal edge of pyrite (white), upper left. Irregular veins of covellite (grey) appear in the mckinstryite but not in the stromeyerite. Plane polarised reflected light, air, 130×

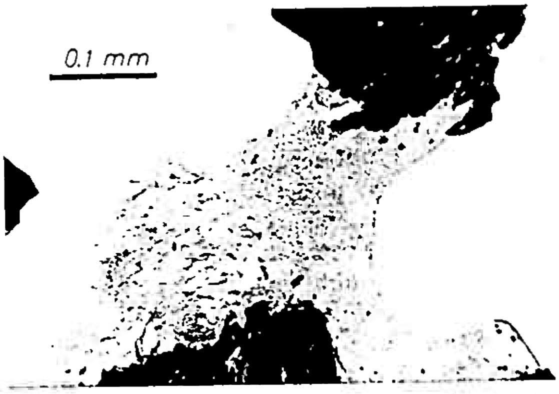

Fig. 4. Cu—Ag sulphides (grey) replacing chalcopyrite to left, producing a ragged, myrmekitic texture. Boundary against galena to right smoother, but also probably replacive. Plane polarised reflected light, air, 130×

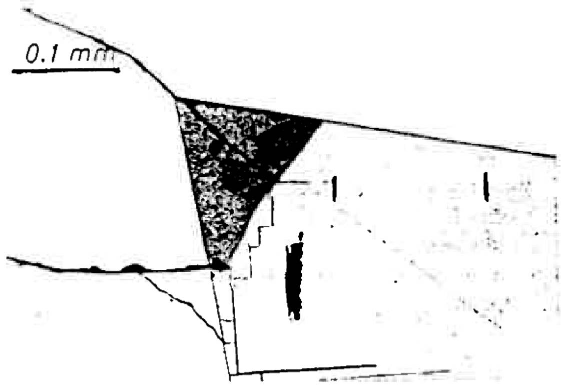

Fig. 5. Mckinstryite (medium grey) with remnants of stromeyerite (dark grey) replacing galena in angle between euhedral pyrite crystals. Reflected polarised light, air, 130×

Another noticeable feature is that the mckinstryite is very often seamed and intersected by short veinlets of covelline (see Fig. 3), a feature which is never seen in the stromeyerite. The possible significance of this is discussed below (p. 11).

Apart from their mutual textural features, the Cu—Ag—S minerals also show distinctive textures relative to the other minerals in the ore. On one hand they exhibit sharp, rectilinear boundaries against the euhedral pyrite and sphalerite; on the other, they show irregular to extremely ragged boundaries towards the other sulphide minerals. This difference in textural appearance would indicate an absence of reactivity of the Cu—Ag sulphides towards pyrite and sphalerite as against considerable reactivity towards minerals such as chalcopyrite, galena and tennantite. Some of the most striking examples of this reactivity can be seen in the case of chalcopyrite, which often shows a ragged, myrmekite-like, texture with the Cu—Ag sulphides along grain boundaries or in isolated grains surrounded by these minerals (Fig. 4). The other affected sulphides show less spectacular mutual relations with the stromeyerite or mckinstryite; in nearly all cases the boundaries are smoothly irregular, intersecting the textural features of the sulphides (see Fig. 5). The above observations indicate that the Cu—Ag sulphides are replacing all the other sulphide minerals in the deposit, with the exception of the pyrite and sphalerite. A discussion of the nature of this replacement is given later in this paper.

Data on Stromeyerite and Mckinstryite

Chemistry

Because of the intimate intergrowths shown by the Cu—Ag—S minerals at Godejord, it was not practically possible to separate out pure fractions for chemical analysis. Thus, quantitative analyses for Cu, Ag, and S were carried out in polished sections of the ore on the

Table 1

STROMEYERITE

Wt %	Area 1	2	3	4	5	6	Average wt %	at pptns
Cu	31.2	31.2	32.0	31.2	32.0	30.9	31.4	1.01
Ag	52.7	52.3	53.2	53.6	53.6	51.5	52.8	1.00
S	15.5	15.6	16.2	15.3	15.8	16.1	15.7	1.00
Total	99.4	99.1	101.4	100.1	101.4	98.5	99.9	

MCKINSTRYITE

Wt %	Area 1	2	3	4	5	6	Average wt %	at pptns
Cu	23.3	23.7	22.9	24.0	23.2	23.8	23.5	0.77
Ag	59.8	61.2	62.2	61.4	62.6	61.9	61.5	1.19
S	15.7	15.2	15.4	15.4	15.1	15.5	15.4	1.00
Total	98.8	100.1	100.5	100.8	100.9	101.2	100.4	

Electron microprobe analysis of Stromeyerite and Mckinstryite from Godejord, Grong area, Norway. Anal. S. Bergstöl.

ARL-EMX 09 electron microprobe housed in the X-Ray Laboratory of the Department of Physics, NTH, Trondheim.

As standards for the analyses were used metallic silver, metallic copper and a standard chalcopyrite. Later a synthetic mckinstryite with composition $Cu_{26.67}Ag_{40.0}S_{33.33}$ (atom percent), synthesized at 250 °C, was placed at our disposal by Dr. B. J. Skinner. The use of this synthetic standard greatly facilitated the microprobe analyses of the natural minerals in our samples.

At first, attempts to obtain reproducible analytical results on the stromeyerite and mckinstryite were unsuccessful, due to their instability under the electron beam. This instability is probably due to loss of sulphur from the carboncoated surface of the sulphides. Mckinstryite seems to be more unstable under the electron beam than does stromeyerite. However, by dint of reducing the voltage to 12 KV, the beam current to 6 NA and by using a defocussed electron beam, good, consistent results were obtained on several grains of both minerals.

Corrections for atomic number factor, absorption and fluorescence were calculated from a correction program, mainly based on that of Springer (1967) modified for a UNIVAC computer by T. Slind. By using Skinner's synthetic standard, mckinstryite could be analysed without corrections. Several points in different areas of the two minerals in six polished sections were analysed.

When analysing mckinstryite, care had to be exercised as it contains inclusions or remnants of stromeyerite which are difficult to distinguish under the microscope of the microprobe

Other elements than those analysed were not detected in either of the minerals.

Table 1 shows the analytical results from six areas each of stromeyerite and mckinstryite from Godejord, and a calculated average composition of each mineral. In Table 2a the Godejord average analysis and calculated atomic composition for stromeyerite are compared with some previously published data.

The calculated atomic composition for the Godejord stromeyerite lies very close to the theoretical 1:1:1 Cu:Ag:S ratio, with only

Table 2a. *Analyses of stromeyerite*

	Locality	1	2	3	4	5	6
Wt per cent	Cu	31.4	31.0	31.5	31.2	32.1	31.24
	Ag	52.8	53.3	52.5	52.3	52.1	53.01
	S	15.7	16.0	15.4	15.8	15.3	15.75
	Total	99.9	100.3	99.4	99.3	100.3	100.00
Atom ppms	Cu	1.01	0.98	1.01	1.00	1.06	1
	Ag	1.00	0.99	0.99	0.98	1.01	1
	S	1.00	1.00	1.00	1.00	1.00	1

References: 1. Stromeyerite, Godejord average. This study.
2. Stromeyerite, Foster mine, Cobalt, Ont. PALACHE, BERMAN and FRONDEL (1944, p. 190)
3. Stromeyerite, Foster mine, Cobalt, Ont. PETRUK *et al.* (1971, p. 207)
4. Stromeyerite, Morrison mine, Cobalt, Ont. PETRUK *et al.* (1971, p. 207)
5. Stromeyerite, Guarismey, Mexico. PALACHE BERMAN and FRONDEL (1944, p. 190)
6. Ag Cu S

Table 2b. *Analyses of mckinstryite*

	Locality	1	2	3	4	5	6
Wt per cent	Cu	23.5	24.9	23.9	24.63	24.59	21.6
	Ag	61.5	60.0	61.0	60.32	59.95	65.1
	S	15.4	15.1	15.1	15.01	14.86	13.7
	Total	100.4	100.0	100.0	99.96	99.40	100.4
Atom ppms	Cu	0.77	0.82	0.80	0.82	0.83	0.80
	Ag	1.19	1.18	1.20	1.19	1.20	1.41
	S	1.00	1.00	1.00	1.00	1.00	1.00

References: 1. Mckinstryite, Godejord, average. This study.
2. Mckinstryite, Foster Mine, Cobalt, Ont. SKINNER *et al.* (1966)
3. Mckinstryite, synthetic. SKINNER, (1966)
4. Mckinstryite, (av.), Chañarcillo, Chile. CLARK and ROJKOVIČ (1971)
5. Mckinstryite, (av.) Echo Bay, NWT. CLARK and ROJKOVIČ (1971)
6. Mckinstryite, (av.) Echo Bay, NWT. ROBINSON and MORTON (1971)

a very slight excess of Cu. The Godejord average composition also compares well with those quoted from the literature, so that there seems no doubt as to the chemical identity of the mineral.

The average analysis and calculated atomic composition for the Godejord mckinstryite is compared in Table 2b with previously published data for this mineral. It can be seen that the Godejord figures lie close to the type mckinstryite from Foster Mine, Cobalt (SKINNER *et al.* 1966) and to SKINNER's (1966) synthetic material. However, compared to these and to the others quoted, the Godejord mckinstryite shows a deficiency in metals, especially in the case of copper. SKINNER *et al's.* material was not analysed by microprobe. The latest available microprobe analyses of mckinstryite appear to be those by ROBINSON and MORTON (1971) and CLARK and ROJKOVIČ

Table 3. *Reflectance values for stromeyerite and mckinstryite*

Mineral and Investigators		λ nm	Maximum R % Range	Maximum R % Average	Minimum R % Range	Minimum R % Average	
Stromeyerite[1]).	This study	542	27.3–27.9	27.75	25.8–26.7	26.3	(8 mmnts.)
Stromeyerite[1]).	This study	542	29.0–30.7	30.1	26.0–26.9	26.45	(16 mmnts.)
Stromeyerite[1]).	This study	542	27.3–30.7	29.3	25.8–26.9	26.4	(24 mmnts.)
Stromeyerite	CAMERON (1963)	549	31 –33	—	28 –29	—	(18 mmnts.)
Stromeyerite	UYTENBOGAARDT and BURKE (1971)	546		29.6		25.8	Range of values.
Stromeyerite[2]).	CLARK and ROJKOVIČ (1971)	546		28.9		25.3	Range of values.
Mckinstryite[1]).	This study	542	30.9–32.5	31.8	28.4–30.6	29.1	(18 mmnts.)
Mckinstryite[2]).	ROBINSON and MORTON (1971)	546	30.3–35.7	32.8	27.6–31.2	29.0	
Mckinstryite[2]).	CLARK and ROJKOVIČ (1971)	546		34.6		31.5	Range of values.

[1]) Godejord material.
[2]) Echo Bay material.

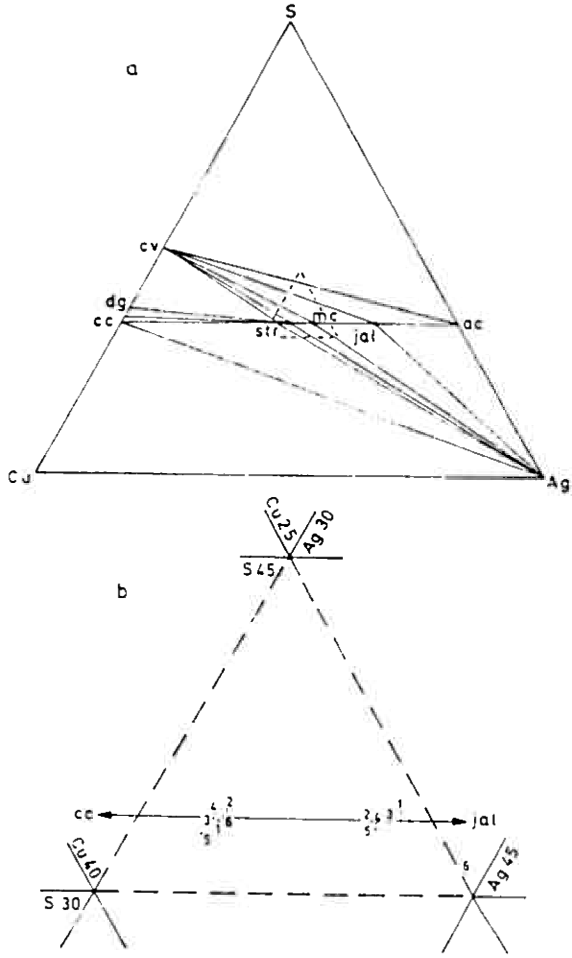

Fig. 6a) The minerals of the Cu—Ag—S system after Skinner (1966). b) Partial Cu—Ag—S triangular plot showing the compositions of six stromeyerites (to left) and six mckinstryites (to right), Numbers correspond to those in Table 2a and 2b, respectively

(1971) on material from Echo Bay, N.W.T., Canada and Chañarcillo, Chile. Table 2b shows that ROBINSON's and MORTON's Echo Bay average analysis has a marked excess of silver compared with all the other analyses in Table 2b. These authors suggest that the excess is due to a "type of Schottky defect whereby some of the anion sites in the lattice are vacant" (1971, p. 347). CLARKE's and ROJKOVIČ's analysis figures show that the Echo Bay material they analysed is much less Ag-rich than ROBINSON's and MORTON's material, with a composition close to the others reported in Table 2B.

In Fig. 6b the atomic compositions of the minerals listed in Tables 2a and 2b have been plotted on part of the Cu—Ag—S triangular diagram. The unusual composition of ROBINSON's and MORTON's Echo Bay material is clearly shown on this plot.

Optics

In polished section the two Cu—Ag-sulphides show similar properties, a feature which initially caused some difficulties in distinguishing between them during observations of their textural relations. Both minerals, being very soft, were difficult to polish, especially where they occurred interstitially to pyrite grains. They are light-sensitive and etch readily.

The stromeyerite has a grey colour with a slight violet tint in ordinary light and exhibits a distinct reflection pleochroism in lighter and darker shades. Anisotropy is strong with distinctive colours; bluish purple to tan. The mckinstryite has a slightly lighter grey colour than the stromeyerite, distinct pleochroism and less strong, but still very distinct, anisotropy with pale greyish blue and light brown colours.

Reflectance values for both minerals have been determined at 542 nm using the Leitz MVP microphotometer in the ore microscopy laboratory of Geologisk Institutt, N.T.H. The standard used was NPL calibrated Black Sika standard No. 82.

The results are shown in Table 3. The few available data from the literature have been added for comparison, even though they were not measured at one and the same wavelength. The Godejord figures must be considered tentative; the error due to wavelength discrepancy is probably the least of those affecting the measurements. However, the data obtained appear to show reasonable agreement with the few that have been published.

In the case of the stromeyerite, there appear to be two groupings of the data for the maximum value of the reflectance. Eight of the twenty four values for R_{max} show a spread from 29.0 to 30.7 %. In the case of R_{min} all twenty-four readings fall into the range 25.8—26.9 %. The data of the Godejord stromeyerite as a whole compare well with the range of values pub-

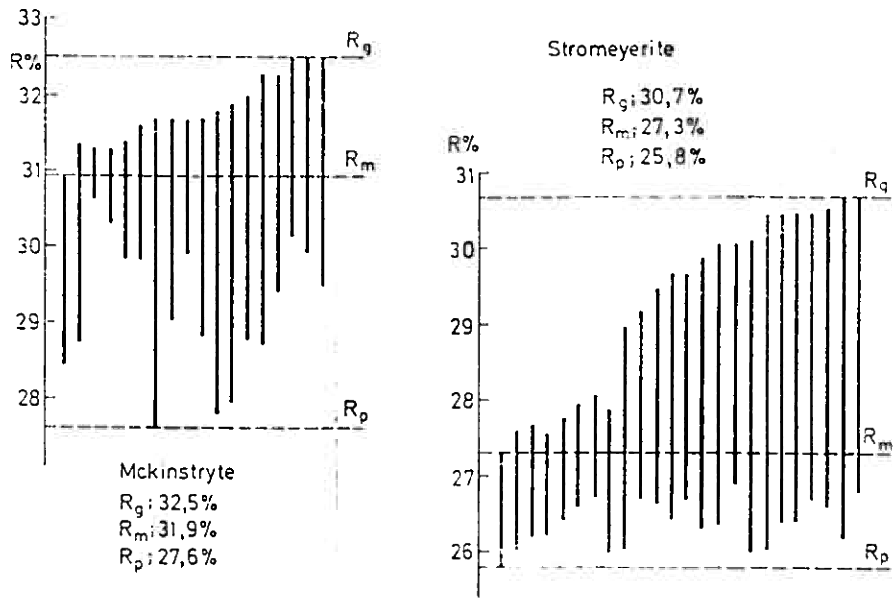

Fig. 7. Diagrammatic representation (after Cameron, 1963) of the reflectivity data for the two Cu—Ag—S minerals at Godejord's prospect. (Measured at 542 nm)

lished by Uytenbogaardt and Burke (1971) as well as those reported by Clark and Rojkovič (1971) for Echo Bay stromeyerite. However they are considerably lower than the values obtained by Cameron (1963).

It will be seen from Table 3 that for mckinstryite the spreads of values for R_{max} and R_{min} are less for the Godejord material than those published by Robinson and Morton (1971) for material from Echo Bay, N.W.T. While the figures for average R_{max} are comparable, the average R_{min} for the Godejord material is about 1% lower. As already shown, Robinson's and Morton's Echo Bay mckinstryite is considerably richer in silver than that from Godejord and this seems to account for the higher reflectance values measured by these workers. They themselves, found that within the Echo Bay material, "the reflectance values were often. . . . higher in areas of proven higher silver content" (1971, p. 343).

Clark and Rojkovič (1971) reported a range of reflectance values for mckinstryite from Echo Bay which shows rather higher maximum and minimum values than those obtained in the present study or by Robinson and Morton (1971).

Fig. 7 is a graphical representation of the reflectance data for the two Godejord Cu—Ag-sulphides. Cameron (1963) has used such a graphical approach to determine optical symmetry of opaque minerals from reflectance measurements. While perhaps not enough data have been obtained, it may be said that the figures give at least an indication of the values of R_p, R_m and R_g for the minerals as well as of their optical signs.

Following the convention employed by Cameron (1963) it can be seen that, at the wavelength employed, 542 nm, the stromeyerite is optically positive, in agreement with what Cameron found for this mineral (1963, Fig. 1). The mckinstryite on the other hand appears from the data in Fig. 7 to be conventionally optically negative at 542 nm (R_m closer to R_g than to R_p). This is the opposite sign from that found for the Echo Bay mckinstryite at 546 nm by Robinson and Morton (1971, p. 343).

Discussion

The Association Stromeyerite-Mckinstryite

The geological significance of the mutual occurrence of these two natural phases in the Cu—Ag—S system has been only sporadically discussed in the literature — mainly due to the relatively recent recognition of mckinstryite

Table 4. *Recorded occurrences of mckinstryite-bearing associations*

Authors	Locality	Cu—Ag—S minerals				
SKINNER, JAMBOR, ROSS 1966	Foster Mine, Cobalt	mck	strom	Ag		
GRYBECK and FINNEY 1968	Silver Plume, Colorado	mck	jalp		cov	
ROBINSON and MORTON 1971	Echo Bay, N.W.T.	mck		Ag		acanthite
CLARK and ROJKOVIČ 1971	Echo Bay, N.W.T.	mck	strom		cov	cc
CLARK and ROJKOVIČ 1971	Chañarcillo, Chile	mck	strom	(Ag,Cu)	cov	
BERGSTÖL and VOKES (this study)	Godejord, Grong	mck	strom		cov	
JOHAN 1967	Bohutin, Pribram	mck	jalp			acanthite
IMAI and LEE 1973	Tada mine, Japan	mck or jalp	strom	Ag		

as a naturally-occurring mineral. The natural occurrence of a phase corresponding to the low-temperature β-phase, $Cu_{0.8}Ag_{1.2}S$, of DJURLE (1958) was predicted some years ago by SKINNER (1966) after his laboratory studies of the Cu—Ag—S system. SKINNER *et al.* (1966) subsequently identified the compound as a mineral in a specimen collected many years previously from the Foster Mine, Cobalt, Ontario, and gave it the name mckinstryite in honour of Hugh Exton Mckinstry. Since then mckinstryite has been recognised in a number of different localities in association with one or other additional member of the Cu—Ag sulphide group. Table 4 lists the occurrences which have been recorded in the literature up to the time of writing. Fig. 6a (after SKINNER 1966) shows the phase relations in the Cu—Ag—S system at 25° and indicates the possible associations of the naturally occurring phases. The association stromeyerite-mckinstryite-covellite is one of these, and, it would appear from Table 4, one that is frequent in the few natural occurrences described.

Most of the investigators who have described natural occurrences of mckinstryite and stromeyerite have discussed the association, based upon SKINNER's (1966) investigations of the Ag—Cu—S system.

SKINNER *et al.* (1966) discuss the temperature relations indicated by the pure mass of coarse-grained crystals of mckinstryite at the Foster Mine, Cobalt, and conclude strongly in favour of a crystallisation temperature below 94.4 °C, the temperature above which the mineral breaks down to a two phase assemblage of jalpaite and a cation-disordered compound of composition $(Cu_{0.96}Ag_{1.04})S$. In the specimen which was examined by SKINNER *et al.* (1966) mckinstryite coexists with minor amounts of native silver and stromeyerite — "an assemblage in complete agreement with the phase diagram below 94.4 °C" (SKINNER *et al.* 1966, p. 1389). (Such an argument does not necessarily hold for complex intergrowths of stromeyerite and mckinstryite such as those observed at Godejord. These could have been formed by cooling of higher temperature (> 94 4 °C) assemblages (B. SKINNER, pers. comm. 1974)).

In the case of the Echo Bay, N.W.T., occurrence there are diverging opinions on the significance of the mckinstryite-bearing assemblages which have been investigated. ROBINSON and MORTON (1971, p. 346), while considering that mckinstryite may be one of the last minerals to crystallize, at a temperature "presumably less than 94.4 °C", cite S-isotope evidence as supporting an hypogene origin for the mineral. In particular they consider that the sulphur in the mineral was derived from the ore-forming fluid and not from the replacement of other sulphides.

CLARK and ROJKOVIČ (1971, p. 8—9) also found that the mckinstryite at Echo Bay is "probably restricted to the near surface exposures of the bornite-chalcopyrite-tetrahedrite-stromeyerite assemblage, which represents the final episode in the hypogene development of the deposit". However these authors concluded, from textural considerations, that the mckinstryite formed by replacement of the stromeyerite under *supergene* conditions. (see below).

Imai and Lee (1973) have noted the association native silver, stromeyerite, mckinstryite (or jalpaite) and an undetermined Cu—Ag mineral at the Tada mine, a xenothermal deposit in the Western Kinki metallogenetic province of Japan. Again, this association is considered to have been deposited at "the latest stage of mineralization" (presumably hypogene?).

At Chañarcillo, Chile, mckinstryite "has definitely replaced stromeyerite, which itself is a product of the supergene sulphide enrichment of silver sulfosalts". (Clark and Rojković 1971, p. 9). Grybeck and Finney (1968, p. 1540) consider that the jalpaite (with which mckinstryite is associated) at the Payrock Mine, Silver Plume, Colorado, is also of supergene origin. It is worth noting, as do Clark and Rojković (1971), that at Echo Bay, Chañarcillo and Silver Plume, the association of mckinstryite with covellite, is apparently a consistent feature. According to Clark and Rojković (1971, p. 8) the proportions of the two minerals approximate those which would result from the breakdown of stromeyerite with minor loss of copper.

As has already been described, the association mckinstryite-covellite is a feature of the Godejord sulphides; no covellite has been observed other than in association with mckinstryite. Clark's and Rojković's explanation may be applicable in this case too, although it must be said that the mckinstryite and covellite at Godejord do not show typical breakdown textures (e.g. myrmekite). However the evidence from Godejord would seem to support Clark and Rojković in their prediction that "mckinstryite is likely to be widely recognised as an alteration product of stromeyerite" (1971, p. 10).

The question whether the association mckinstryite-stromeyerite represents the results of hypogene or supergene deposition, or of a combination of the two processes still appears to be an open one.

Origin of the Godejord Cu—Ag-*Sulphides*

A solution to the question of the origin of the stromeyerite and mckinstryite at Godejord's prospect is not only desirable from a scientific point of view. From an economic point of view it would be of greatest significance if it could be deduced that the minerals were a) supergene and therefore confined to the surface-near parts of the deposit or b) hypogene and therefore presumably not so confined.

At the outset it must be said that minerals of this nature are not usually regarded as being typical primary minerals of the stratabound massive sulphide deposits, certainly not in the amounts met with in the Godejord specimens. Such general considerations do make a primary hypogene origin for the Cu—Ag sulphides seem unlikely.

Textural investigations on the Godejord specimens also show that the Cu—Ag sulphides are not part of the fabric shown by the mineralization as a whole. They have undoubtedly been deposited in their present positions by a replacement process affecting already existing sulphides in the ore. They have been shown to replace chalcopyrite and, less spectacularly, galena, and tennantite. The pyrite and sphalerite appear to be totally unaffected by this replacement.

A point that may be mentioned here, is that many pyrite and some chalcopyrite grains show a large number of small inclusions. These inclusions are rounded or irregular in shape and mostly consist of sphalerite and galena, but chalcopyrite, bornite and Cu—Ag—S minerals are also found. Some of the inclusions are multicomponent types where the minerals are: Cu—Ag—S minerals, probably both stromeyerite and mckinstryite, Cu—Ag—Te sulphide, chalcopyrite, bornite, galena, sphalerite, native gold, tennantite and possibly Cu—Sn sulphide. These inclusions favour a hypogene origin for the stromeyerite, mckinstryite and other minerals in them. The rounded inclusions strongly suggest that they might have originated as liquid droplets, possibly formed by the selective melting of components in the ore during the metamorphism which it obviously has undergone. The possible formation of such melts at metamorphic temperatures has already been discussed by one of us (Vokes 1971) and the matter will not be pursued further here, except to point out that the compositions of the inclusions are such that they might be expected to form low-temperature melts. Whether the bulk of the stromeyerite and mckinstryite, occurring interstitially to the pyrite and sphalerite grains, could also be the result of metamorphic melting must remain an open question for the present.

Postulation of a supergene origin for the Cu—Ag—S and associated minerals at Godejord also raises a number of problems. It has already been mentioned that the Cu—Ag sulphides replace preferentially chalcopyrite, galena and tennantite — in decreasing order of preference. Sphalerite and pyrite are completely unaffected and these are just those minerals that the Ag-rich supergene sulphides should cement on first, according to the Schürmann series of chalcophile affinities. (see, for example, PARK and MACDIARMID 1964, pp. 436—440). In addition, microscopic examination of a large number of polished sections fails to reveal any other typical supergene minerals, which could be expected in the cementation zone of such a deposit.

The only mineral of recognised supergene origin to be found is covelline, but as has been shown already, this is solely associated with mckinstryite, which appears to be replacing the stromeyerite.

A supergene cementation origin for the Cu—Ag sulphides also raises the question of the mineralogical nature of the primary silver-bearer(s) in the ore. From general considerations, both galena and tennantite seem to be likely silver bearers. Microprobe investigations (S. B.) have shown that the Godejord galena is low in Ag (under the detection limit of about 500 ppm). However the few grains of tennantite that have been analysed by this technique show an average of 0.34% Ag. One may thus say that a possible source for the supergene enriched silver does exist in the ore in the form of silver-bearing tennantite. Such a mode of occurrence, for the stromeyerite at least, would accord with the opinion of RAMDOHR (1969, p. 480) that it is in most cases "a very typical mineral of the secondary supergene sulphide enrichment zone" and that, as such, it forms "especially from silver-rich fahlore..."

It is planned to investigate further the distribution of the various minerals in the Godejord ore with a view to determining their modes of origin. The persistence or otherwise in depth of the Cu—Ag—S minerals, for instance, will solve the question of whether these are of supergene or hypogene origin and it is hoped that drill core specimens will be made available in the near future for this purpose.

Acknowledgements

The authors are grateful to Dr. B. J. SKINNER, Yale University, for the loan of a synthetic standard mckinstryite, and to Mr. IAN CRIDDLE of the British Museum, (Natural History), London, for some excellent polished sections of the mckinstryite-stromeyerite bearing material. The text has benefited from a critical appraisal by Dr. G. KULLERUD, Purdue University, Indiana.

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APPENDIX No. 2

Outokumpu 1992 Study of Skiftesmyr and Godejord

SKIFTESMYR FOREKOMST UNDERJORDISK DRIFT.

Skivpallbryting:

LEVELS Nivåavstand 30 m
Barrierpilar mot dagen
Horisontale pilarer mellom brytningsrommene 15 m
ORE IN SITU Råmalm mengde uten (without) vertikale pilarer (cut off 1% Cu-ekv.)

	2 648 000	tonn
	1,08	Cu%
	1,63	Zn%
	0,31	Au ppm
	8,65	Ag ppm
	34,6	S%

Investeringer:

MAIN ADIT	1 - Grunnstoll 650 m	5,8 mill.kr
ACCESS RAMP	2 - Hovedramp 3152 m	25,6 mill.kr
MAIN VENT.	3 - Hovedventilasjon	10,2 mill.kr
OTHER	4 - Andre faste anlegg	5,1 mill.kr
	SUM	46,7 mill.kr
EQUIPMENT	5 - Maskinpark (ny) new	43,7 mill.kr
Coarse Crusher	6 - Grovknuser anlegg (nytt) new	7,2 mill.kr
	SUM	50,9 mill.kr
	SUM TOTAL	97,5 mill.kr
USED	Brukte maskiner 5 og 6 kostnad	10 - 15 mill.kr.

Driftskostnader:

Tunneling	Ortdrift	8000 kr/m
mining	Gruveavdeling	250 kr/m^3

23. april 1992/eml

GODEJORD FOREKOMST

UNDERJORDISK DRIFT.

Skivpallbryting	nivåavstand	20 m
Råmalm hvis bryting til		260 m.o.h.
	76 221 tonn	
	0,76 Cu%	
	7,76 Zn%	
	0,83 Au ppm	
	24,47 Ag ppm	
	16,5% gr.tilblanding (DILUTION)	
Gråberg:	Fra skjæring	6329 tonn
	Fra skråvei	22540 tonn
	Fra vent.sjakt	550 tonn
	Sum	29419 tonn
Ortdrift:	I malm	200 m
	I gråberg	392,5 m

Joma, 27. april 1992/EL

APPENDIX No.2a

Metallurgical Investigations of the Skiftesmyr Ore by NGU and Outokumpu:1977

Report on orientation flotation tests on ore from the Skiftesmyr Deposit, Grong Gruber A/S

By Odd Eidsmo
NTH, Feb., 1977.

Translated by Arne Reinsbakken
Trondheim, 1st Feb., 1997.

SUMMARY:
The intention with these tests was primarily to investigate how the silver content in the crushed raw-ore feed is distributed in the different flotation concentrates. The listing of assays, Appendix 1, shows that the Cu-concentrate will contain 50-60 g/t Ag while the Zn-concentrate contains 40-50 g/t.
From the silver content in the feed, 30% is found in the Cu-concentrate while 20% is found in the Zn - concentrate. Hence, about half of the silver from the zinc flotation will be lost in the tailings.

Otherwise, these tests indicate that the Skiftesmyr ore is easy to mill and that it will be possible to make a high class concentrate with good recovery for both Copper and pyrite.

Providing that the sphalerite does not contain too much iron, it should also be possible to make a high class Zn - concentrate.

The minerals show a satisfactory liberation on grinding down to 50-60% -325 mesh which for this type of ore would require 10-12 kW/t for grinding in a conventional mill.

Feed.
The tests were performed on drill cores selected by Arve Haugen. Haugen's sample descriptions are found in Appendix 2.

The samples recieved had a weight of 2.2 kg and was marked: Ddh 1-2-3-4, sample 1.

The ability of the ore mineral liberation has been investigated by Terje Malvik and presented in a report dated January 1977.

Malvik's investigation has been made on two samples.

Sample 1: This represents a section down the dip of the ore zone (surface exposure?)and includes drill holes 4-1-2-3. This sample is virtually identical with the above sample used for the flotation tests.

Sample 2: This represents a section that more or less follows the strike of the ore zone, at approximately the same depth, and includes drill holes 5-16-1-10.

From Malvik's report, one can see that sample 2 obtained mineral liberation in a coarser fraction than that in sample 1.

Malvik's report is found in Appendix no. 3.

....

Flotation tests and Analyses
Tests are run in batch scale with 0.5 kg charges.

screen (mesh) analyses for ground goods:

Mesh	65	150	200	270	325	-325
weght %	0.1	6.0	12.8	13.0	10.1	58.0

Copper flotation is run in four steps with 10 g/t KAX in each step -pH 11.5.

Zinc flotation is run in three steps + schawenger. Copper sulphate addition (tilsats) 400 g/t and 20 g/t KAX in each step - pH 12.0.

In order to get an orientation of the Cu- and Zn- contents in a pyrite concentrate, ca. 50 g/t KAX was added after the zincschawenger and the amount that was 'willing' to come over with the collectors (samlertilsatsen) was floated out.

Results from the best tests are shown in appendix 1.

The elements in the different products are analysed by atomic absorption at Grong Gruber A/S.

There is a very good aggreement in Cu, Zn and Fe between the analysed feed (pågang) and the feed calculated out from the analysed values and the weight -% in the products. Silver, on the other hand, shows much less aggreement, 10 ppm in the analysed feed and only 7 ppm in the sum of the products (calculated pg.).

Silver is also controll analysed at NGU using atomic absorption.

Appendix 1 shows that NGU analysed the raw ore feed to 13 ppm and the calculated feed was 12 ppm, thus a very good aggreement between the analysed and calculated feed.

It is interesting to notice that for Cu-concentrate 1, which has 25.44 %(?) Cu, Grong Gruber A/S and NGU have recieved exactly the same value.

For the pyrite concentrate and pyrite mp./tailings, NGU got 2 and 6 times higher values, respectively, than Grong Gruber.

....

Appendix 1. **Skiftesmyr**

Results from flotation tests on ore samples (rågods) marked BhHull 1-2-3-4, sample 1.

1 Ag - analyses done by Grong Gruber A/S.

2 Ag - analyses done by NGU (Dept. Eng. Knut Solem).

Utv. = extraction, gain

Vekt % = weight %

Rå-kons. = raw concentrate

pyritmp./Avgang = pyrite ? /tailings

Pågang beregnet = ore feed calculated

Pågang analysert = ore feed analysed

Appendix 2. 2 pages

Grong Grubere A/S
Exploration div.

to: O. Eidsmo
From: A. Haugen
Date: 1 Nov., 1976.

Samples for Microscopic Investigations - T. Malvik.

1. The ore zone at Skiftesmyr (appendix) is to be investigated microscopically for mineral dressing parameters, for eg., liberation grinding. In order to obtain representative samples based on diamond drill hole - samples, two vertical sections were set out through the deposit, at roughly right angles to each other. Two samples were taken from these two sections:

Sample 1: represents a section along the plunge of the orebody and contains drill holes 4-1-2-3.

Sample 2: represents a section more or less along the strike of the orebody, in a somewhat even depth and contains drill holes 5-16-1-10.

These two samples represent therefore 7 of the 12 drill hole that we have in the field.

2. The drill core was split at Joma. Part of the sample is stored in the core boxes. The other part was coarse-crushed with a relative fine adjusted jaw crusher. The coarse-crushed material was split and half of this was stored at Joma. The other half was fine crushed and further split and sent to analyses.

3. Selection of samples was as follows: The orebody's thickness in each drill-holes is calculated. Pure silicate host-rock fragments and bands that occur in the orebody are not included in the samples (these contain in order 0.05% Cu and 0.1% Zn). A new sample is split out from the stored sample of each analysis sample from each drill hole. Each split has a weight that corresponds to it's core length, m, and it's specific weight (density):

m x sp. wt. x 10 = weight of split.

This procedure was followed for each core length from the ore zone in the drill hole that was sent to analyses. All the samples for each drill hole was then added together to make one sample. After this, all the samples from the drill holes in the chosen vertical section were then combined to form a composite.

4. Sample 1 weighs ca. 1100 gram and sample 2 weighs ca. 600 gram. Most of the material has a grain size in the order of 1-6 mm and very little will be over 10 mm. Sample 2, which contains drill hole 16, contains, for the most + 10 mm, because this drill-hole was crushed using a plate crushers and a coarser jaw-crusher.

5. Parallel samples to the two samples refered to above were sent from Joma to O. Eidsmo on Fri., 29 Oct., 1976.

<u>Appendix 3:</u> An evaluation of the grinding liberation factors for chalcopyrite and sphalerite from the Skiftesmyr Deposit.

- by Terje Malvik, Geological Institute- NTH., Jan., 1977.

<u>1. Summary.</u>

<u>1.1 Sample Material.</u> was composed of two drill-core samples. The sample material is described in more detail in the Appendix.

<u>1.2 Method of investigation.</u> An empirical model, described in Teknisk Rapport (Technical Report) 28/2 (BVLI), is used to evaluate the grinding free liberation properties.

<u>1.3 Conclusion.</u> Chalcopyrite and sphalerite are relatively coarse-grained in the ore from Skiftesmyr. Much coarser-grained than at Skorovass, and also more coarse-grained than the same minerals in ore from the Tverrfjellet deposit.

Fine grinding to give ca. 55% liberation at - 45 microns for chalcopyrite and sphalerite is regarded as a reasonable starting point, and it is assumed that this corresponds to a total fine grind liberation in the area 40-45% -45 microns.

2. Description of Investigation.

2.1 Fine Grinding. Preliminary grinding tests were carried out in order to choose the necessay grinding time to obtain a reference fine grind 50% -45 microns for chalcopyrite and sphalerite. The fractions + 45 and - 45 microns were analysed for Cu and Zn at Grong Gruber A/S. The results of these analyses are found in the appendix.

Appendix 1 shows the fine grind (partical size) as a function of grinding time for the two ore samples.

The curves show that a grinding time of 21 min. for sample 1 and 13 min. for sample 2 gives the reference fine grind liberation (?) 50% -45 microns for chalcopyrite and sphalerite.

The fraction -125+90 microns of the grind product was taken out for preparation of polished sections and a grain analyses was conducted on these sections.

2.2 Grain Analyses. Table, appendix 2, shows results of grain analyses.

This investigation is concerned only with the study of the grinding liberation factors for chalcopyrite and sphalerite and, therefore, only grains with these minerals are registered in the traverses across the polished sections. The distance between the traverse lines is 0.5 mm.

There is a distinction made here between whole grains (100% of mineral), half grain (ca. 20-80% of mineral), contaminated grain 5-20% (minority mineral represents ca. 20% of grain), contaminated grain < 5% (minority mineral represents < 5% of grain) and multiphase grain which is a grain with three minerals in approx. equal amounts.

The following terms are used in the Table:

b = (bergart) = rock
sl = sinkblende = sphalerite
cp = kobberkis = chalcopyrite
py = svovelkis = pyrite
po = magnetkis = pyrrhotite
gn = blyglans = galena
fahl = fahlerts = tetrahedrite - tennantite solid solution

In regards to the ore microscopic work, it was noted that mixed grains were generally of a simple type, and that chalcopyrite and sphalerite also occur as inclusions mostly in pyrite. Such grains are divided out as separate types.

Fahlerts (possible silver bearing) is registered in a half grain associated with chalcopyrite in sample no. 2.

On the basis of this grain analyses, the degree of liberation in the fraction -125+90 microns is calculated to:

	chalcopyrite	sphalerite
sample no. 1	73.7 ± 2.3	72.6± 2.7
sample no. 2	78.9 ± 2.1	81.4 ± 2.5

The degree of liberation is given ± standard deviation.

2.3 Liberation curves. When one constructs the developement of the total degree of liberation as a function of the grinding down of a mineral, it is important to keep in mind that a mineral distribution curve for the grinding product of the ore will have an influence on the development of the curve for total liberation.

In Appendix 3 and 4 the developement of the total degree of liberation for chalcopyrite and sphalerite is constructed on the basis of the model described in T.R. 28/2, and with the mineral distribution curve nearly parallels the curve for chalcopyrite and sphalerite in the feed to the mineral dressing plant at Tverrfjellet (Folldal A/S).

The curve for Tverrfjellet is chosen because the ore types here, to a certain degree, can be compared with the ore types at Skiftesmyr.

The total degree of liberation is marked by a field with hatched lines, because the degree of liberation in the fraction ± standard deviation is used in the model.

The corresponding curves for chalcopyrite and sphalerite in the feed to the flotation plant at the Tverrfjellet and Skorovas mines are also shown in order to illustrate the relative liberation relationships between the three deposits.

Chalcopyrite and sphalerite is a slightly coarser grained in sample 2 than in sample 1 (the curve for sample 2 is a little higher in the diagram).

Within the individual samples, both chalcopyrite and sphalerite show nearly identical liberation relations.

2.4 Evaluation of the actual grade of liberation. The actual grade of liberation will be given by the wished grade of liberation for the minerals.

An investigation of the flotation feed (23-26/03-76), for the Skorovas mine, gives a grade of liberation for chalcopyrite to be 78%.

If one chooses this value of the liberation grade as a starting point, this corresponds to the grinding down of chalcopyrite and sphalerite in sample 1 to ca. 45 - 48% -45 micron, and in sample 2, ca. 40 - 44 % -45 micron.

In all likelihood, a total liberation grade in the order of ca. 85 % would be better for chalcopyrite and sphalerite in a relatively coarse-grained ore type as Skiftesmyr.

This can be obtained for sample 1 by grinding chalcopyrite and sphalerite to the order of 54 - 58 % - 45 micron, and 50 - 54 % - 45 micron for sample 2.

A reasonable middle grade of grinding could therefore be suggested to be ca. 55 % - 45 micron for chalcopyrite and sphalerite.

What this means in the total grinding of the sample will depend on to which degree chalcopyrite and sphalerite grind down selectively.

In the laboratory grinding done for this investigation, chalcopyrite and sphalerite is to a large degree ground down selectively.

On comparing the grinding curves on Appendix 1 with similar curves for raw flotation feed from Tverrfjellet (T.R. 28/02 p. 245), the selective grinding appears to be comparable, and one can say that with grinding the Skiftesmyr ore in a large scale drift one will get approximately the same relationship as that found for Tverrfjellet.

This will mean that at a total grinding of the ore in the order of 40 - 45 % - 45 micron will give ca. 55 % - 45 micron for chalcopyrite and sphalerite.

Appendix 1: Graph which shows axes with 'grinding time in min.' versus 'mineral particle size as a function of grinding time % -45 micron'.

Appendix 2: Table with 'Grain analyses of fraction -125+90 micron of two grind products from the Skiftesmyr ore'.

Antall registrerte korn med kobberkis og sinkblende = no. of grains registered with chalcopyrite and sphalerite

helkorn = whole grain
halvkorn = half grain
smittede korn 5-20% = contaminated grain 5-20%
smitted korn 5% = contaminated grain 5%
multifase korn = multi-phase grain

Beregnete frimalingsgrader = calculated grade of liberation.
prøve = sample

Appendix 3:
Graph 1 - most likely developement of the total grade of liberation for chalcopyrite in sample no. 1 from Skiftesmyr on the basis of mineral distribution curve approximately parallel to the curves for chalcopyrite and sphalerite in flotation feed from the Tverrfjellet mine.

Graph 2 - most likely developement of the total grade of liberation for sphalerite in sample no. 1 from Skiftesmyr on the basis of mineral distribution curve approximately parallel to the curves for chalcopyrite and sphalerite in flotation feed from the Tverrfjellet mine.

Appendix 4:

Graph 1 - most likely developement of the total grade of liberation for chalcopyrite in sample no. 2 from Skiftesmyr on the basis of mineral distribution curve approximately parallel to the curves for chalcopyrite and sphalerite in flotation feed from the Tverrfjellet mine.

Graph 2 - most likely developement of the total grade of liberation for sphalerite in sample no. 2 from Skiftesmyr on the basis of mineral distribution curve approximately parallel to the curves for chalcopyrite and sphalerite in flotation feed from the Tverrfjellet mine.

Bilag 3-8 sider

En vurdering av frimalingsegenskapene til kobberkis og sinkblende i Skiftesmyrforekomsten.

Utført januar 1977
ved Geologisk Institutt, NTH

av

Terje Malvik

En vurdering av frimalingsegenskapene til kobberkis og sinkblende i Skiftesmyrforekomsten.

1. Sammendrag.

1.1 Prøvemateriale. Dette bestod av to prøver av borkjerner. Vedlegg beskriver prøvematerialet nærmere.

1.2 Undersøkelsesmetodikk. En empirisk modell, beskrevet i Teknisk Rapport 28/2 (BVLI), er benyttet til å vurdere frimalingsegenskapene.

1.3 Konklusjon. Kobberkis og sinkblende opptrer relativt grovkornig i malmen. Vesentlig mer grovkornig enn i Skorovas-malmen, og også mer grovkornig enn tilsvarende mineraler i Tverrfjellet.

En nedmaling til ca. 55 % ÷45 mikron for kobberkis og sinkblende er antydet som et fornuftig utgangspunkt, og det er antatt at dette tilsvarer en total nedmaling i området 40-45 % ÷45 mikron.

2. Nærmere beskrivelse av undersøkelsen.

2.1 Nedmaling. For å bestemme nødvendig maletid for å oppnå referansenedmalingen 50 % -45 mikron for kobberkis og sinkblende ble det utført innledende maleforsøk. Fraksjonene +45 og ÷45 mikron ble analysert på Cu og Zn ved Grong Gruber. Analyseresultater i vedlegg.

Bilag 1 viser nedmaling som funksjon av maletid for de to malmprøvene.

Av kurvene fremgår at maletid 21 min. for prøve 1 og 13 min. for prøve 2 gir referansenedmalingen 50 % ÷45 mikron for kobberkis og sinkblende.

Av dette maleproduktet ble fraksjonen ÷125+90 mikron siktet ut, planslip fremstilt og kornanalyse utført.

2.2 Kornanalyse. Tabell, bilag 2, viser resultatet av kornanalysen.

Denne undersøkelsen tar kun sikte på å vurdere frimalingsegenskapene til kobberkis og sinkblende, -derfor er bare korn med disse mineralene registrert ved traversering over slipene med linjeavstand 0,5 mm.

Det er skilt mellom helkorn (100 % av mineralet), halvkorn (ca. 20 til 80 % av mineralene), smittede korn 5-20 % (minoritetsmineralet utgjør ca. 5-20 % av kornet), smittede korn < 5 %, (minoritetsminerale utgjør under ca. 5 % av kornet) og multifase korn som er korn med tre mineraler i ca. like store mengder.

Følgende betegnelser er anvendt i tabellen :

b	=	bergart
sl	=	sinkblende
cp	=	kobberkis
py	=	svovelkis
po	=	magnetkis
gn	=	blyglans
fahl	=	fahlerts

Det kan bemerkes til mikroskoperingen at blandingskornene generelt er av enkle typer, men kobberkis og sinkblende opptrer også som inneslutninger i første rekke i svovelkis. Slike korn er skilt ut som egne typer.

Fahlerts (mulig sølvbærer) er registrert i et halvkorn med kobberkis i prøve nr. 2.

Med grunnlag i kornanalysen er frimalingsgradene i fraksjonen ÷125+90 mikron beregnet til :

	kobberkis	sinkblende
Prøve nr. 1	73,7 ± 2,3	72,6 ± 2,7
Prøve nr. 2	78,9 ± 2,1	81,4 ± 2,5

Frimalingsgradene er angitt ± standardavviket.

2.3 Frimalingskurver. Når man skal konstruere utvikling av total frimalingsgrad som funksjon av nedmaling for et mineral, er det viktig å være klar over at mineralfordelingskurvene i maleprodukt av malmen vil innvirke på kurveforløpet.for total frimaling.

På bilag 3 og 4 er utviklingen av total frimalingsgrad for kobberkis og sinkblende konstruert med basis i modell beskrevet i T.R.28/2, og med mineralfordelingskurver tilnærmet parallelle kurvene for kobberkis og sinkblende i pågang flotasjon ved Tverrfjellet.

Kurvene fra Tverrfjellet er valgt fordi malmtypen til en viss grad er sammenlignbar med Skiftesmyr-malmen.

Total frimalingsgrad er markert med et skravert felt som skyldes at frimalingsgrad i fraksjon plus og minus standardavviket er anvendt i modellen.

Inntegnet er også tilsvarende kurver for kobberkis og sinkblende i pågang flotasjon ved Tverrfjellet og Skorovas Gruber for å anskueliggjøre relative frimalingsforhold mellom de tre forekomstene.

Kobberkis og sinkblende opptrer litt mer grovkornig i prøve 2 enn i prøve 1. (Kurver for prøve 2 er litt høyere i diagrammet).

Innen den enkelte prøve viser kobberkis og sinkblende nært sammenfallende frimalingsforhold.

2.4 Vurdering av aktuell nedmalingsgrad. Aktuell nedmalingsgrad vil være gitt av ønsket frimalingsgrad for mineralene.

Ved undersøkelse av pågang flotasjon (23-26/3-76), Skorovas Gruber, er frimalingsgrad for kobberkis bestemt til 78 %.

Velger man denne verdien for frimalingsgrad som utgangspunkt tilsvarer det en nedmaling av kobberkis og sinkblende i prøve 1 til ca. 45 - 48 % ÷45 mikron, og i prøve 2 ca. 40 - 44 % ÷45 mikron.

Sannsynligvis vil en total frimalingsgrad i størrelsesorden ca. 85 % være mer gunstig for kobberkis og sinkblende i en relativt grovkornig malmtype som Skiftesmyr.

Dette oppnås for prøve 1 ved en nedmaling av kobberkis og sinkblende i området 54 - 58 % ÷45 mikron, og 50 - 54 % ÷45 mikron i prøve 2.

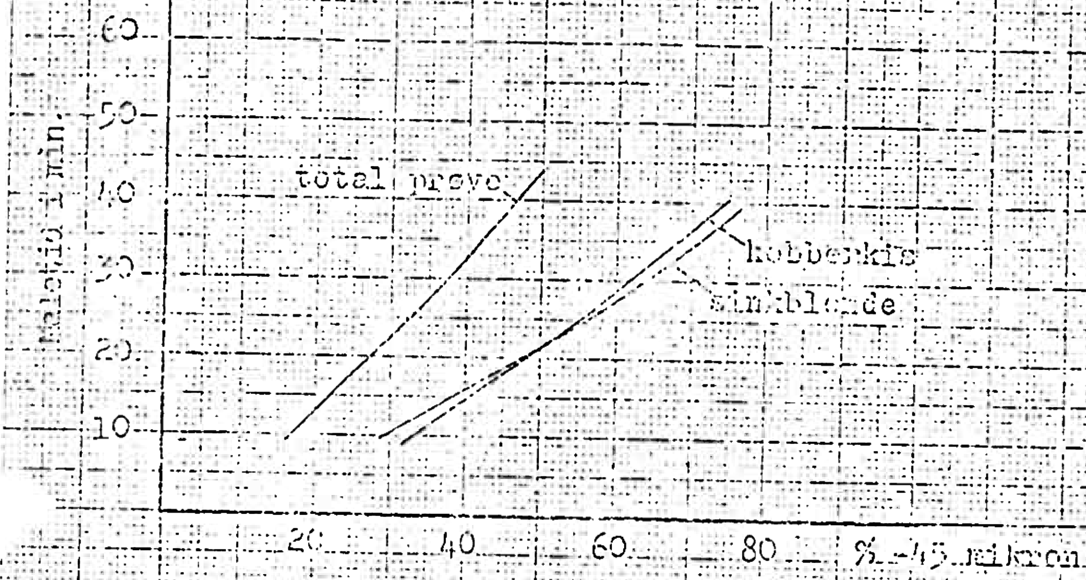
En fornuftig midlere nedmalingsgrad vil derfor kunne antydes å være ca. 55 % ÷45 mikron for kobberkis og sinkblende.

Hva dette betyr i total nedmaling av prøven vil være avhengig av til hvilken grad kobberkis og sinkblende males ned selektivt.

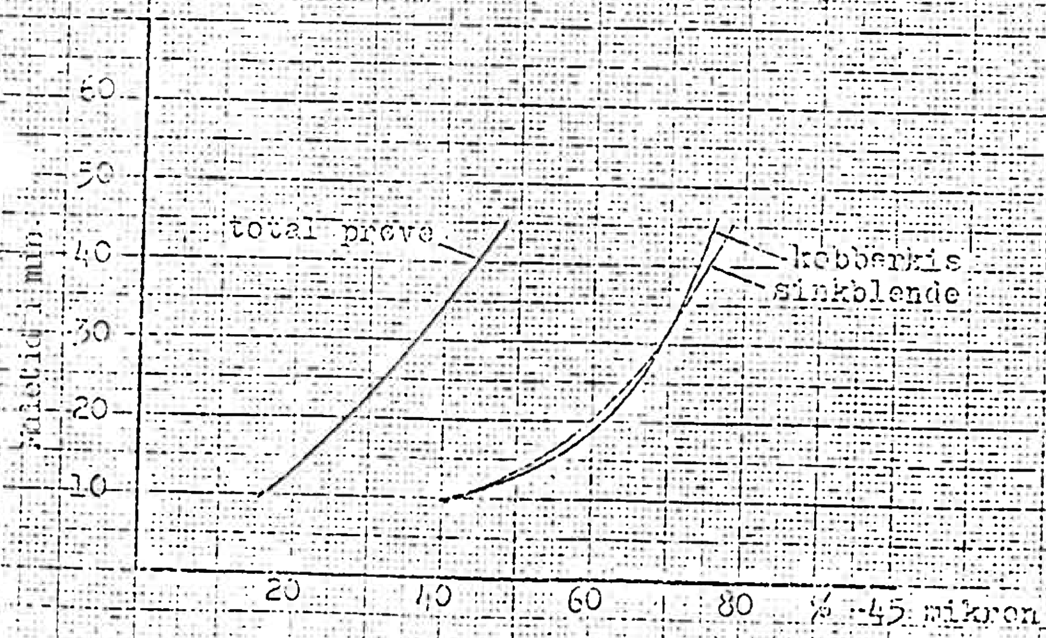
Ved laboratorienedmalingen utført ved denne undersøkelsen er kobberkis og sinkblende i stor grad nedmalt selektivt.

Sammenlignes nedmalingskurvene på bilag 1 med tilsvarende kurver for rågods fra Tverrfjellet (T.R.28/2 s.245), kan den selektive nedmalingen sies å være sammenlignbar, og man kan antyde at man ved maling i drift av Skiftesmyrmalmen vil kunne få tilnærmet samme forhold som ved Tverrfjellet.

Dette vil da tilsi at en total nedmaling av malmen i området 40 - 45 % ÷45 mikron vil gi ca. 55 % ÷45 mikron for kobberkis og sinkblende.



Nedmaling som funksjon av maletid, prøve nr. 1.



Nedmaling som funksjon av maletid, prøve nr. 2.

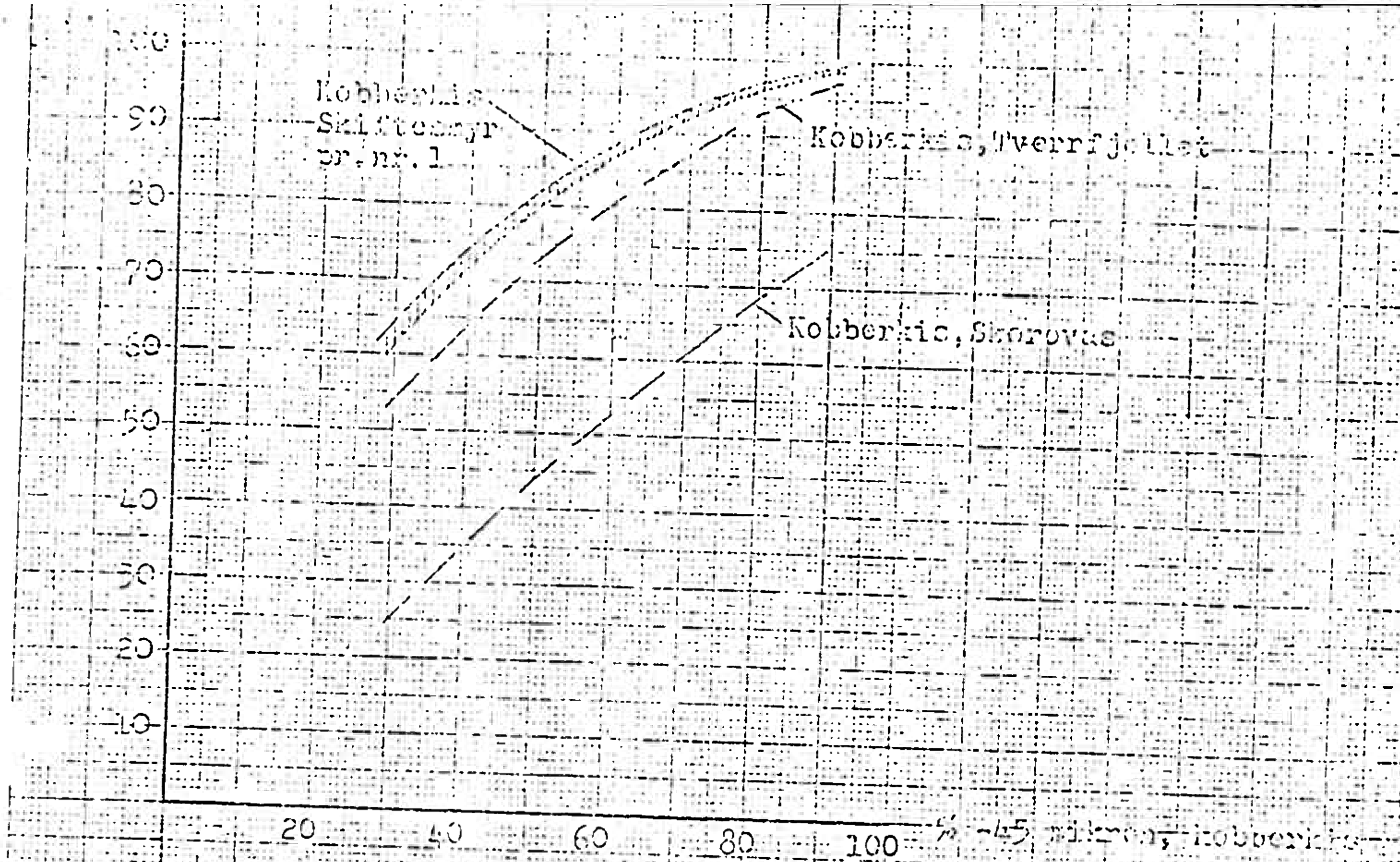
Kornanalyse av fraksjonen -125+90 mikron av to maleprodukt av Skiftesmyr-malmen.

Antall registrerte korn med kobberkis og sinkblende

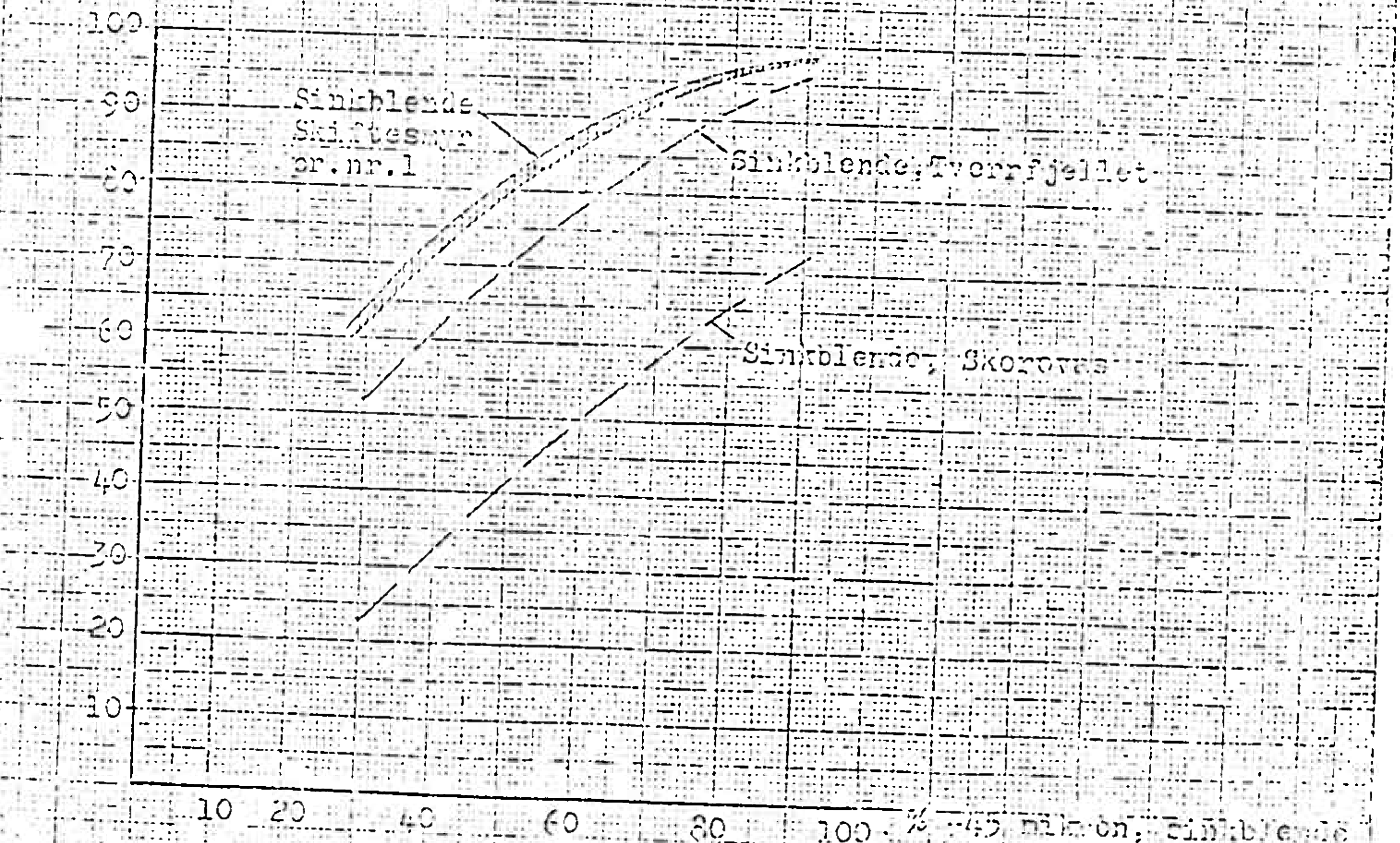
		prøve nr. 1	Prøve nr. 2
Helkorn	cp	126	169
	sl	121	148
Halvkorn	cp/sl	12	11
	cp/py	26	27
	cp/b	4	2
	cp/fahl	-	1
	cp/po	2	3
	sl/py	22	18
	sl/po	1	-
	sl/b	-	1
Smittede korn 5-20 %	cp/py	4	-
	cp/sl	3	6
	cp/po	-	1
	cp/b	-	5
	cp/gn	-	1
	py/cp	90	71
	py/cp	6	6
	sl/cp	10	6
	po/cp	-	2
	b/cp	4	2
	sl/py	1	1
	sl/b	1	-
	py/sl	49	34
	py/sl	6	-
	b/sl	3	2
Smittede korn 5 %	py/cp	46	29
	py/cp	12	9
	po/cp	1	-
	sl/cp	7	6
	b/cp	8	10
	sl/gn	1	-
	py/sl	22	16
	py/sl	11	1
Multifase korn	cp/sl/py	4	-

Beregnete frimalingsgrader :

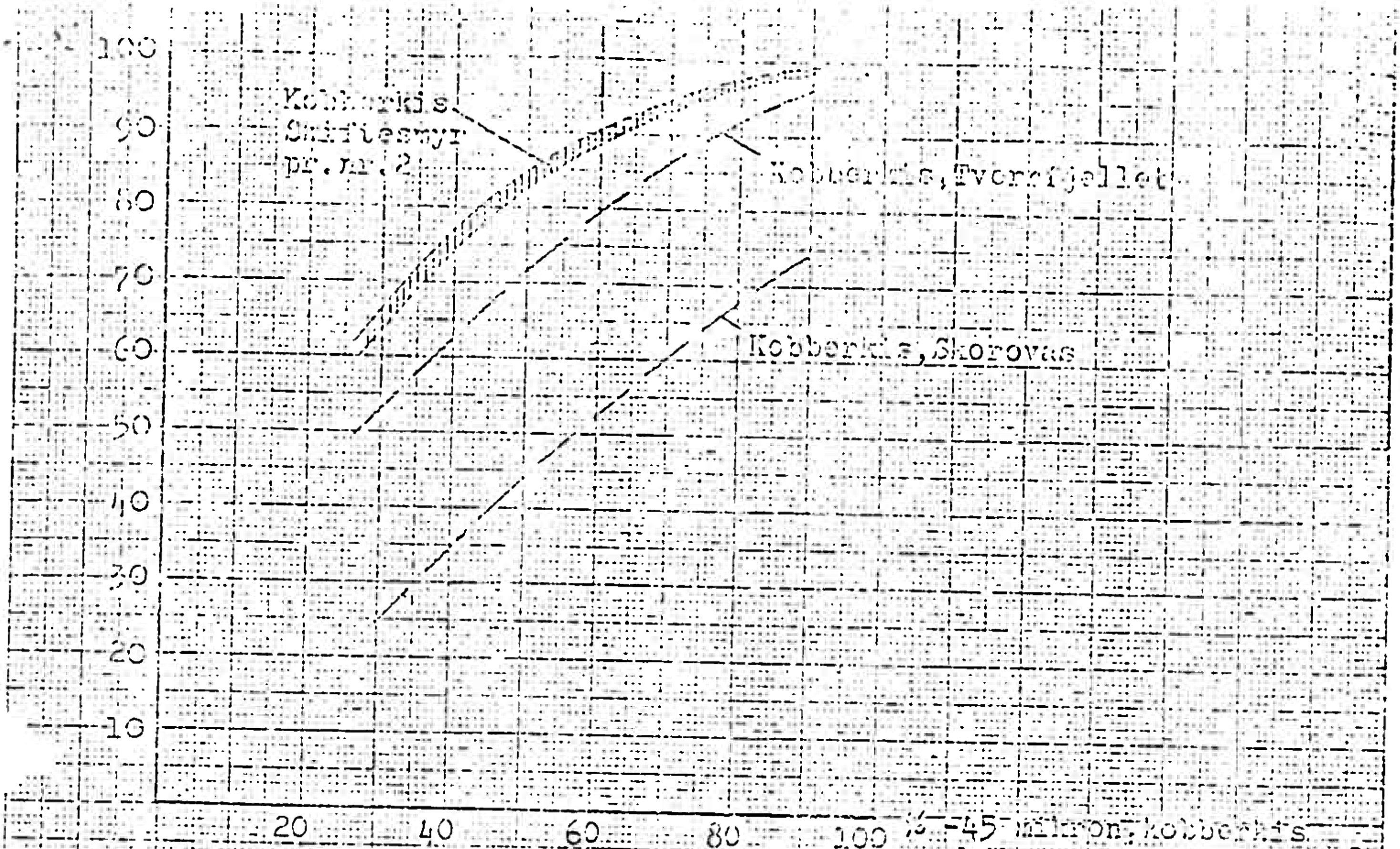
	cp	sl
prøve nr.1	73,7±2,3	72,6±2,7
prøve nr.2	78,9±2,1	81,4±2,5



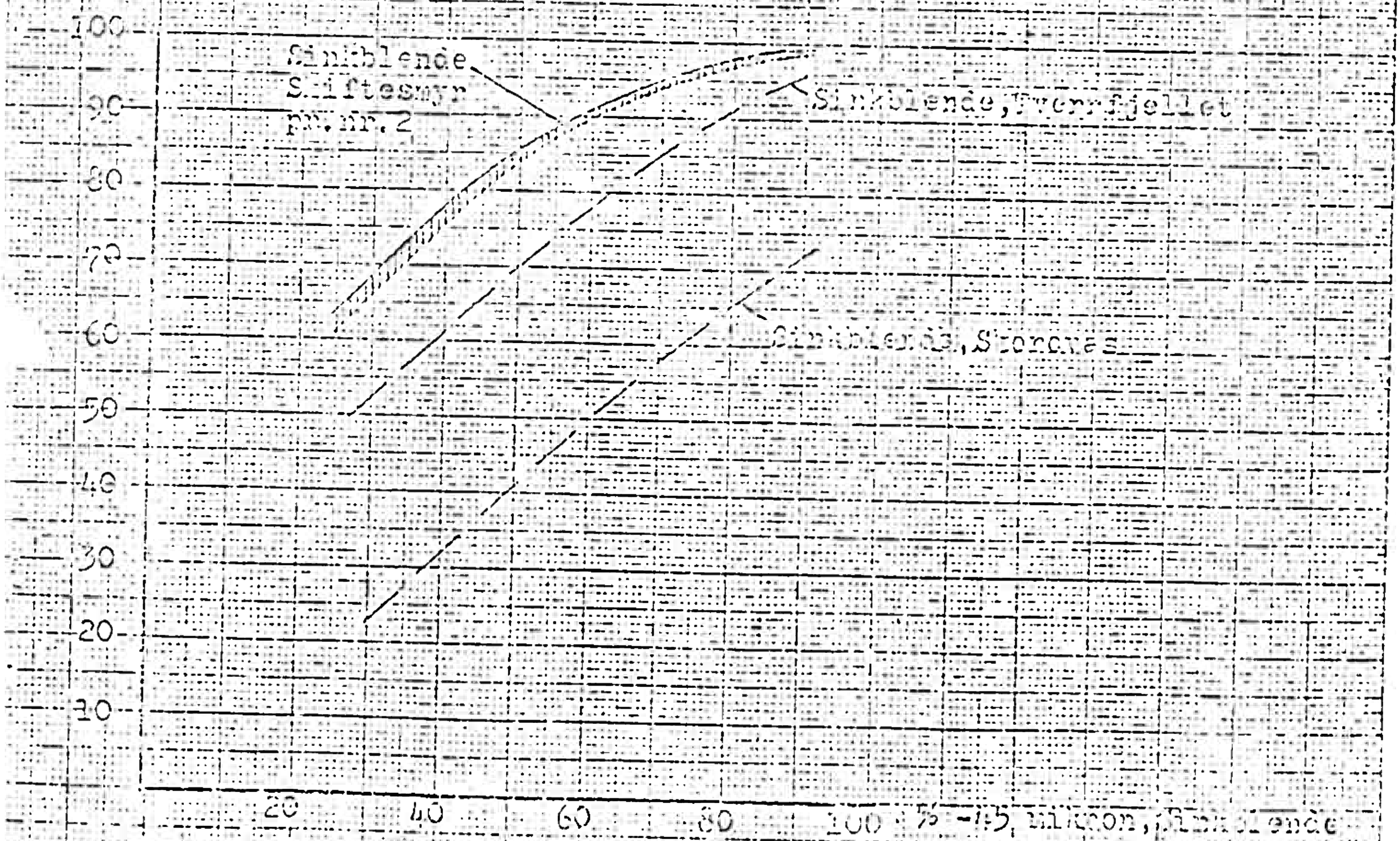
Sannsynlig utvikling av total frimalingsgrad for kobberkis i prøve nr.1 fra Skiftesmyr med grunnlag i mineralfordelingskurver tilnærmet parallelle kurvene for kobberkis og sinkblende i pågang flotasjon, Tverrfjellet Grube.



Sannsynlig utvikling av total frimalingsgrad for sinkblende i prøve nr.1 fra Skiftesmyr med grunnlag i mineralfordelingskurver i et malprodukt [illegible] parallelle kurvene for kobberkis og sinkblende i pågang flotasjon, Tverrfjellet Grube.



Sannsynlig utvikling av total frimalingsgrad for kobberkis i prøve nr. 2 fra Skiftesmyr med grunnlag i mineralfordelingskurver i et maleprodukt tilnærmet parallelle kurvene for kobberkis og sinkblende i pågang flotasjon, Tverrfjellet Grube.



Sannsynlig utvikling av total frimalingsgrad for sinkblende i prøve nr. 2 fra Skiftesmyr med grunnlag i mineralfordelingskurver i et maleprodukt tilnærmet parallelle kurvene for kobberkis og sinkblende i pågang flotasjon, Tverrfjellet Grube.

- Grubedivisjonen

RAPPORT

over orienterende flotasjonsforsøk med malm fra Skiftesmyr

for

Grong Gruber A/S

NTH, febr. 1977

O. Eidsmo

Odd Eidsmo

Skift flo.80

SUMMARY:

The intention with these tests was primarily to decide how the silver content in the feed appear in the different floatation concentrates. The listing of assays, Attachment 1, shows that the Cu-concentrate will contain 50-60 g/t Ag while the Zn-concentrate contains 40-50 g/t.
From the silver content in the feed 30% is found in the Cu-concentrate while 20% is found in the Zn-concentrate. Hence around half of the silver will be lost in the tailing from the zinc flotation.

Otherwise these tests indicate that the Skiftesmyr ore is easy to mill and it will be possible to make high class concentrate with a good recovery both for copper and pyrite.

Provided that the sphalerite does not contain too much iron, it should also be possible to make high class Zn-concentrate.

The minerals show a satisfactory liberation by grinding down to 50-60% -325 mesh which for this type of ore would require 10-12 kWh/t for grinding in a conventional mill.

Feed.

The tests are made on drill cores selected by Arve Haugen. Haugen's description of the sampling is given in Attachment 2.

The received sample had a weight of 2.2 kg and was marked: DDH 1-2-3-4, Sample 1.

The ore minerals liberation ability has been investigated by Terje Malvik and presented in a report dated January 1977.

Malviks's investigation has been made on two samples.

Sample 1: This represents a section down dip of the ore zone and includes the drill holes 4-1-2-3. This sample is virtually identical with the above sample used for the flotation tests.

Sample 2: This represents a profile that more or less follows the strike of the ore zone, at approximately same depth, and includes the drill holes 5-16-1-10.

From Malviks report, one can read that sample 2 obtain mineral liberation in a coarser fraction than sample 1.

Malviks report is enclosure no.3

Hensikten med disse forsøkene var først og fremst å få klarlagt hvordan råmalmens sølvinnhold fordeler seg i respektive flotasjons-produkter. Sammenstilling av analyser på bilag 1 viser at Cu-konsentratet vil holde 50 til 60 g/t sølv. Zn-konsentratet 40 til 50 g/t. Av råmalmens sølvinnhold vil ca. 30% gå i Cu-konsentratet og ca. 20% i Zn-konsentratet. Det vil si at ca. halvparten av sølvet vil være igjen i avgangen fra sinkflotasjonen.

Forøvrig indikerer disse forsøkene at Skiftesmyrmalmen er forholdsvis lett å opprede, så det vil være mulig å lage høyverdige konsentrater med bra utvinning både for kobber og pyrit.

Forutsatt at sinkblenden ikke inneholder for mye jern, er det også gode muligheter for å lage et høyverdig Zn-konsentrat.

Malmen frimales tilfredsstillende ved nedmaling til 50-60% -325 mesh som for denne malmtype tilsvarer 10-12 kWh/t for nedmaling i konvensjonelle møller.

Rågods.

Forsøkene er kjørt på borkjerner uttatt av Arve Haugen. Haugens beskrivelse av prøvetakningen fremgår av bilag 2.

Tilsendte prøve på 2,2 kg var merket: BhHull 1-2-3-4, Prøve 1.

Malmens frimalingsegenskaper er undersøkt av Terje Malvik og presentert i rapport dat. januar 1977.

Malviks undersøkelser er utført på to prøver.

Prøve 1: Representerer et profil langs malmens fall og inneholder borhullene 4-1-2-3. Denne prøven er praktisk talt identisk med ovennevnte prøve som flotasjonsforsøkene er kjørt på.

Prøve 2: Representerer et profil mer eller mindre langs forekomstens strøk, i et noenlunde jevnt dyp og inneholder borhullene 5-16-1-10.

Det fremgår av Malviks rapport at prøve 2 frimales på et noe grovere stadium enn prøve 1.

Malviks rapport vedlegges som bilag 3.

Flotasjonsforsøk og analyser.

Forsøkene er kjørt i batchskala med 0,5 kg charger.

Sikteanalyse for nedmalt gods:

Mesh	65	150	200	270	325	-325
Vekt%	0,1	6,P	12,8	13,0	10,1	58,0

Kobberflotasjonen er kjørt i fire trinn med 10 g/t KAX i hvert trinn - pH 11,5.

Sinkflotasjonen er kjørt i tre trinn + schawenger. Kobbersulfat-tilsats 400 g/t og 20 g/t KAX i hvert trinn - pH 12,0.

For å få en orientering om Cu- og Zn-innholdet i et pyritkonsentrat, ble det etter sinkschawenger tilsatt ca. 50 g/t KAX og flotert ut den mengden som var "villig" til å komme over med denne samler-tilsatsen.

Resultater fra det gunstigste forsøket er vist på bilag 1.

Elementene i de forskjellige produktene er bestemt ved atom-absorpsjon av Grong Gruber.

For Cu, Zn og Fe er det meget god overensstemmelse mellom analysert pågang og pågang beregnet på grunnlag av gehalt og vekt-% i produktene. For sølvet derimot er overensstemmelsen mindre bra, 10 ppm i analysert pågang og bare 7 ppm i sum produkter (beregnet pg.)

Det er utført kontrollanalyser for sølvet ved NGU som også har kjørt på atomabsorpsjon.

Av bilag 1 fremgår at NGU har analysert rågodset til 13 ppm og beregnet pågang viser her 12 ppm, altså meget god overensstemmelse mellom analysert og beregnet pågang.

Det er interessant å ~~iaktta~~ merke at for Cu-kons. 1 med 25,44 Cu har Grong og NGU funnet nøyaktig samme gehalt.

For pyritkons. og pyritmp./avg. har NGU ~~h.h.v.~~ henholdsvis 2 og 6 x høyere gehalt enn Grong.

Bilag 1.

Skiftesmyr.

Resultater fra flotasjonsforsøk med rågods merket BhHull 1-2-3-4, Prøve 1.

1 Ag-bestemmelser utført av Grong Gruber

2 - " - " - " - " NGU v/avd.ing. Knut Solem

PRODUKT	Vekt%	Cu		Zn		Fe		Ag [1]		Ag [2]	
		%	Utv.	%	Utv.	%	Utv.	ppm	Utv.	ppm	Utv.
Cu-kons. 1	1,1	25,44	23,5	1,89	1,2	31,4	0,9	54,0	8,5	54	5,2
- " - 2	1,9	21,10	33,7	2,30	2,5	32,3	1,6	50,0	13,5	53	8,8
- " - 3	1,1	18,68	17,3	4,87	3,1	32,8	1,0	60,3	9,5	69	6,6
- " - 4	1,1	11,98	11,1	7,83	4,9	33,8	1,0	70,4	11,1	75	7,2
- " - 1-4= Rå-kons.	5,2	19,6	85,6	3,9	11,7	32,6	4,5	57,3	42,6	61	27,8
Zn-kons. 1	1,9	1,82	2,9	45,38	49,0	14,1	0,7	20,0	5,4	26	4,3
- " - 2	1,0	2,56	2,1	37,53	21,3	18,5	0,5	35,0	5,0	40	3,5
- " - 3	1,8	2,84	4,3	15,84	16,2	32,6	1,6	62,0	15,9	61	9,5
- " - 1-3= Rå-kons.	4,7	2,3	9,3	31,2	86,5	21,2	2,8	37,6	26,3	42	17,3
Zn-schawenger	1,6	0,95	1,3	1,34	1,3	42,1	1,8	46,0	10,5	47	6,6
Pyritkons.	22,1	0,03	0,5	0,01	0,1	47,1	27,8	3,5	11,1	7	13.5
Pyritmp./Avgang	66,4	0,06	3,3	<0,01	<0,4	35,5	63,1	1,0	9,5	6	34,8
Pågang beregnet	100,0	1,19	100,0	1,78	100,0	37,4	100,0	7,0	100,0	12	100,0
- " - analysert		1,18		1,84		37,7		10,0		13	

B1129 A - 2 sider

GRONG GRUBER A/S
PROSPEKTERINGSAVDELING.

Til: O.Eidsmo
Fra: A. Haugen
Dato: 01.11.76.

PRØVE TIL MIKROSKOPUNDERSØKELSE - T. MALVIK.

1. Malmsonen i Skiftesmyr (bilag) skal undersøkes mikroskopisk med henblikk på oppredningsparametre, f.eks. frimaling. For å få representative prøver basert på diamantborhulls- kjerner, ble lagt to profiler gjennom forekomsten, mest mulig på tvers av hverandre og ut fra dette, to prøver uttatt:

Prøve 1: Representerer et profil langs malmens fall og inneholder borhullene 4 - 1 - 2 - 3.

Prøve 2: Representerer et profil mer eller mindre langs forekomstens strøk, i et noenlunde jevnt dyp og inneholder borhullene 5 - 16 - 1 - 10.

Disse to prøvene representerer dermed 7 av de 12 aktuelle borhull vi ahr i feltet.

2. Borkjerner blir i Joma kløyvd. En ene del arkiveres i kjernekassen. Den andre del går til grovknusing med en relativt fint stilt kjefteknuser. Av det grovknuste blir utsplittet en halv-del for oppbevaring. Den andre finknuses, splittes ned og går til analyse.

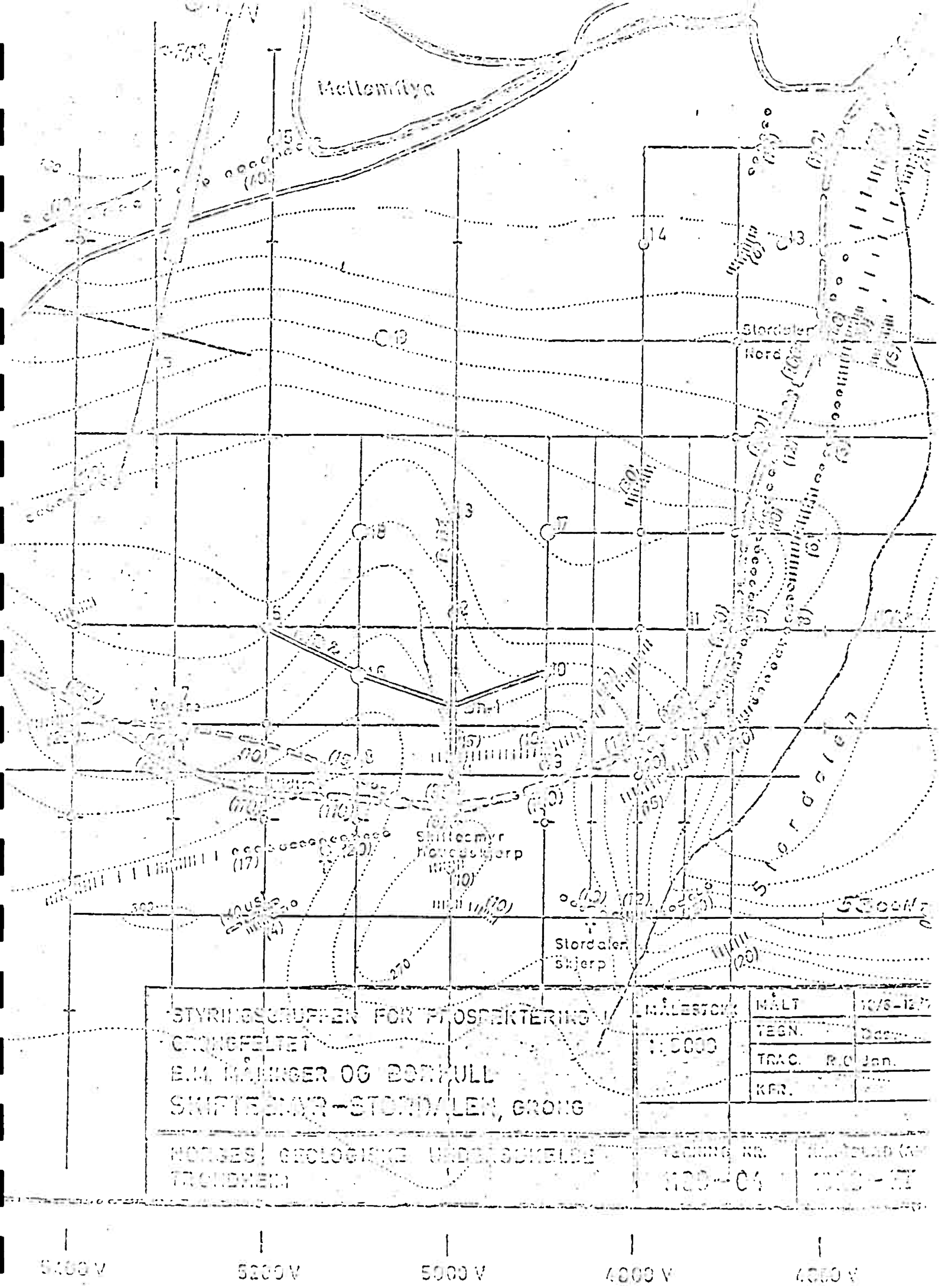
3. Uttaket av prøvene er gjort som følger: Malmsonens mektighet i hullet er bestemt. Rene gångfjellsfliser som opptrer inne i malmen er ikke medtatt i prøvene (disse inneholder i størrelsesorden 0,05% Cu og 0,1% Zn). Fra den arkiverte utsplitt av grovknust materiale for hvert analyseuttak i hvert borhull, så splittet ut en prøve. Denne utsplitten har en vekt ifølge kjernelengden, m,og dens spesifikke vekt - altså

m x sp.v. x 10 = utsplittens vekt.

Det samme ble gjort for alle kjernlengdene som har gått til analyse i borhullets malmsone. Alle prøvene for hvert borhull blir slått sammen. Deretter ble alle prøver fra borhull i de utvalgte profiler slått sammen.

4. Prøve 1 har vekt ca. 1100 gram, prøve 2 ca. 600 gram. Det vesentligste av materialet har en kornstørrelse i området 1-6 mm. Svært lite vil være over 10 mm. Prøve 2 som inneholder bh 16 har mest + 10 mm fordi dette hull ble knust med med tallerkenknuser og grovere kjefteknuser.

5. Det foreligger i Joma en paralell til de to prøver som nå er uttatt. Prøvene ble sendt fra Joma til O.Eidsmo fredag 29.10.76.



Mellommyra
Stordalen
Nord
Skiftesmyr
Hovedskjerp
Stordalen
Skjerp
Stordalen
STYRINGSGRUPPEN FOR PROSPEKTERING I
GRONGFELTET
E.M. MÅLINGER OG BORHULL
SKIFTESMYR-STORDALEN, GRONG
MÅLESTOKK
1:5000
MÅLT
TEGN.
TRAC.
KFR.
NORGES GEOLOGISKE UNDERSØKELSE
TRONDHEIM
5400 V
5200 V
5000 V
4800 V
4600 V

APPENDIX No. 3

Metallurgical Testing of Skiftesmyr/Godesjord Ores by Lakefield Research

Appendix 1

Laboratory Test Reports

Sample Information					
Box Number	Number Bags	Sample I.D.	Weight kg	Name	Description
As Received					
1	2	GO-1	17.6	Godejord	Sphalerite rich massive to semi-massive Ore
	1	GO-2	7.0	Godejord	As GO-1, semi-massive, increased Cu content
	1	SKM-2	10.3	Skiftesmyr	Massive Pyrite Ore
2	2	SKM-1	17.8	Skiftesmyr	Cu-rich disseminated, semi-massive ore
	1	SKM-2	12.2	Skiftesmyr	Massive Pyrite Ore
3	2	SKM-1	17.5	Skiftesmyr	Cu-rich disseminated, semi-massive ore
Composites	**2**	**GO-1**	**17.6**	**Godejord**	**Sphalerite rich massive to semi-massive Ore**
	1	**GO-2**	**7.0**	**Godejord**	**As GO-1, semi-massive, increased Cu content**
	4	**SKM-1**	**35.3**	**Skiftesmyr**	**Cu-rich disseminated, semi-massive ore**
	2	**SKM-2**	**22.5**	**Skiftesmyr**	**Massive Pyrite Ore**

Head Assays

ICP Scan

		GO-1	GO-2	SKM-1	SKM-2		GO-1	GO-2	SKM-1	SKM-2
Ag	g/t	54.3	46.0	19.5	22.5					
Al	g/t	2200	3000	33000	20000	%	0.22	0.30	3.30	2.00
As	g/t	100.0	< 30	265	< 30	%	0.010		0.027	
Au	g/t	1.82	0.76	1.39	0.38					
Ba	g/t	290	20.0	430	190	%	0.03	0.00	0.04	0.02
Be	g/t	< 1.0	< 1.0	< 1.0	< 1.0	%				
Ca	g/t	50000	11000	4600	1500	%	5.00	1.10	0.46	0.15
Cd	g/t	530	650	49.0	89.0	%	0.05	0.07	0.00	0.01
Co	g/t	12.0	8.40	39.0	37.0	%	0.00	0.00	0.00	0.00
Cr	g/t	26.0	32.0	57.0	62.0	%	0.00	0.00	0.01	0.01
Cu	**g/t**	**10000**	**32000**	**16000**	**9700**	**%**	**1.00**	**3.20**	**1.60**	**0.97**
Fe	**g/t**	**179000**	**143000**	**309000**	**337000**	**%**	**17.90**	**14.30**	**30.90**	**33.70**
K	g/t	420	180	10000	8000	%	0.04	0.02	1.00	0.80
La	g/t	< 50	< 50	< 50	< 50	%				
Mg	g/t	11000	14000	8100	3200	%	1.10	1.40	0.81	0.32
Mn	g/t	470	220	92.0	61.0	%	0.05	0.02	0.01	0.01
Mo	g/t	51.0	120	< 10	< 10	%	0.01	0.01		
Na	g/t	530	200	1200	1800	%	0.05	0.02	0.12	0.18
Ni	g/t	38.0	33.0	95.0	20.0	%	0.00	0.00	0.01	0.00
P	g/t	600	730	700	99.0	%	0.06	0.07	0.07	0.01
Pb	g/t	3900	4100	1000	520	%	0.39	0.41	0.10	0.05
Sb	g/t	< 30	< 30	< 30	< 30	%				
Se	g/t	< 50	< 50	< 50	< 50	%				
Sn	g/t	< 20	< 20	< 20	< 20	%				
Te	g/t	< 10	< 10	< 10	< 10	%				
Y	g/t	< 5.0	< 5.0	14.0	< 5.0	%			0.0014	
Zn	**g/t**	**157000**	**193000**	**3500**	**19000**	**%**	**15.70**	**19.30**	**0.35**	**1.90**

Test No: F1 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: 11/12/1996**

Purpose: Cu/Zn/Pyrite Float

Procedure:

Feed: 1000 grams of GO-1 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill (NB).

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at start of stage	
	Lime	NaCN	SO2	A3418	R208	Flex-31	CuSO4	MIBC	Grind	Cond.	Froth	pH	Eh
Grind	1000	10	-	-	-	-	-	-	45			9.8	+40
Cu Aeration	200	-	450	-	-	-	-	-		20		8.4	+90
Cu Rougher	-	-	-	10	20	-	-	50		1	5	8.2	+110
Cu 1st Clnr	-	-	-	-	-	-	-	15		1	3	8.0	+100
Cu 2nd Clnr	-	-	-	-	-	-	-	10		1	3	7.8	+100
Cu 3rd Clnr	-	-	-	-	-	-	-	5		1	3	7.8	+140
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	2710	-	-	-	-	-	1000	-		4		10.5	+30
Zn Rougher	1000	-	-	-	-	50	-	50		1	5	10.5	+80
Zn 1st Clnr	100	-	-	-	-	-	-	25		1	3	11.0	+100
Zn 2nd Clnr	100	-	-	-	-	-	-	10		1	2	11.0	+70
Zn 3rd Clnr	50	-	-	-	-	-	-	5		1	2	11.0	+90
Pyrite Flotation on Zn Rougher Tails													
	H2SO4											7.0	+160
Pyrite Rougher	1000	-	-	-	-	100	-			1	5	7.6	+145
Pyrite Scavenger						100					2	7.8	+120
Pyrite 1st Clnr	-	-	-	-	-	-	-	5		1	2	7.8	+140
Purite 2nd Clnr	-	-	-	-	-	-	-			1	2	7.8	+145

Stage	Cu, Zn, Pyrite Roughers	Cu, Zn, Pyrite Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F1 Project: Braddick-Grong, 5031 Operator: F.V.

Date: 11/12/1996

Metallurgical Balance

Product	Weight		Assays, %, g/t				% Distribution			
	grams	%	Cu	Zn	Fe	Pb	Cu	Zn	Fe	Pb
Cu 3rd Clnr Conc	123.2	12.4	7.41	27.9	22.8	2.27	77.0	22.9	15.9	69.1
Cu 3rd Clnr Tail	10.2	1.03	2.15	41.1	15.0	0.68	1.9	2.8	0.9	1.7
Cu 2nd Clnr Tail	24.1	2.43	1.28	39.6	17.3	0.52	2.6	6.3	2.4	3.1
Cu 1sr Clnr tail	45.4	4.58	0.91	41.5	15.5	0.28	3.5	12.5	4.0	3.1
Zn 3rd Clnr Conc	82.5	8.32	0.15	66.5	1.39	0.02	1.0	36.5	0.6	0.3
Zn 3rd Clnr Tail	12.5	1.26	0.44	51.9	9.53	0.11	0.5	4.3	0.7	0.3
Zn 2nd Clnr Tail	36.6	3.69	0.47	24.7	28.5	0.19	1.5	6.0	5.9	1.7
Zn 1st Clnr Tail	105.3	10.6	0.45	7.39	39.7	0.18	4.0	5.2	23.6	4.7
Pyrite 2nd Clnr Conc	85.3	8.61	0.20	1.79	45.8	0.09	1.4	1.0	22.1	1.9
Pyrite 2nd Clnr Tail	12.5	1.26	0.71	2.63	42.2	0.32	0.7	0.2	3.0	1.0
Pyrite 1st Clnr Tail	58.8	5.93	0.61	2.20	40.2	0.31	3.0	0.9	13.3	4.5
Pyrite Scav. Conc.	11.6	1.17	0.70	2.48	30.0	0.55	0.7	0.2	2.0	1.6
Pyrite Scav. Tail	383.0	38.6	0.07	0.48	2.67	0.07	2.2	1.2	5.8	6.8
Head (Calc)	991.0	100.0	1.20	15.2	17.9	0.41	100.0	100.0	100.0	100.0
Head (direct)			**1.00**	**15.7**	**17.9**	**0.39**				

Calculated Grades and Recoveries

Cu 3rd Clnr Conc	12.4	7.41	27.9	22.8	2.27	77.0	22.9	15.9	69.1
Cu 2nd Clnr Conc	13.5	7.01	28.9	22.2	2.15	78.9	25.6	16.7	70.9
Cu 1st Clnr Conc	15.9	6.13	30.5	21.5	1.90	81.5	32.0	19.1	74.0
Cu Rougher Conc	20.5	4.96	33.0	20.1	1.54	85.0	44.5	23.1	77.1
Zn 3rd Clnr Conc	8.3	0.15	66.5	1.39	0.02	1.0	36.5	0.6	0.3
Zn 2nd Clnr Conc	9.6	0.19	64.6	2.46	0.03	1.5	40.8	1.3	0.7
Zn 1st Clnr Conc	13.3	0.27	53.5	9.70	0.07	3.0	46.8	7.2	2.4
Zn Rougher Conc	23.9	0.35	33.0	23.0	0.12	7.0	52.0	30.8	7.1
Zn Rougher Feed	79.5	0.23	10.6	17.3	0.12	15.0	55.5	76.9	22.9
Pyrite 2nd Clnr Conc	8.6	0.20	1.79	45.8	0.09	1.4	1.0	22.1	1.9
Pyrite 1st Clnr Conc	9.9	0.27	1.90	45.3	0.12	2.2	1.2	25.0	2.9
Pyrite Rougher Conc	15.8	0.39	2.01	43.4	0.19	5.2	2.1	38.4	7.4
Pyrite Rougher Feed	55.6	0.17	0.96	14.8	0.12	8.1	3.5	46.1	15.8

LR-5031
Braddick-Grong

Test No: F1

Eh Profile

Time (min)	Condition	Eh	pH
0	Out of Grinding Mill	+40	9.8
	After Lime Addition	+10	10.6
	After SO2	+60	9.0
1	Air on	+70	9.0
2		+70	9.0
3		+70	9.0
5		+70	8.9
5.5	After air off for 30 sec	+60	8.8
10		+70	8.7
10.5	After air off for 30 sec	+70	8.7
15		+85	8.5
15.5	After air off for 30 sec	+85	8.5
20		+95	8.4
20.5	After air off for 30 sec	+90	8.4

Test No: F1
Project: Braddick-Grong, 5031
Operator: F.V.
Date: 11/12/1996

1000 grams of GO-1 Sample, crushed to 10 Mesh

Objective:

Initial flotation test, based on Kidd Creek Reagent Scheme
Produce three concentrates: Copper, Zinc and Pyrite.

Observations:

Copper Flotation:

grey colored froth in roughers, especially at end of flotation time.
nice copper color in cleaners, stable froth, lots of material in cleaner conc.

Zinc Flotation:

Pyrite coloration in rougher, especially at end.
Roughers cut short due to expected high pyrite content
Additional collector tested visibly, no increase in zinc coloration; i.e. zinc not undercollected.
This product was not pulled into zinc rougher concentrate, but became part of pyrite concentrate.
Scavenger should be tested to eliminate zinc from pyrite concentrate
froth in cleaners very thick and not very stable, big bubbles bursting at the rim, frother addition did not make it better

Pyrite flotation:

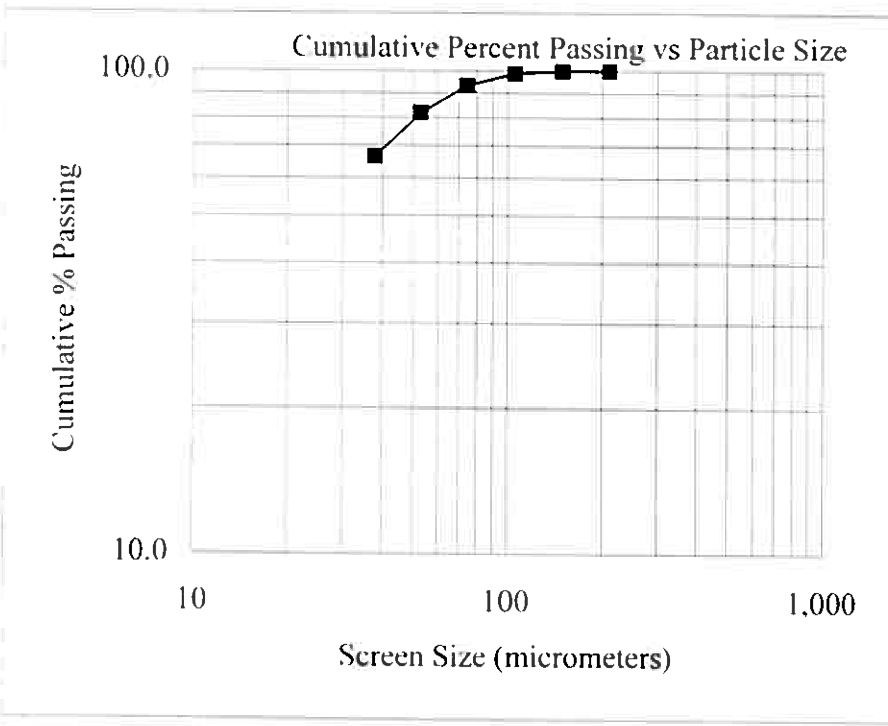
Zinc coloration at start of pyrite rougher.
Slow floating, still pulling at end of 3 minutes targeted.
Added additional collector and pulled another 2 minutes.
Galena hue visible in frothe after second collector addition.
black-grey color in cleaners, stable froth, lots of material in cleaners

Company **Lakefield Research** **LR-5031**

Size Distribution Analysis

Sample: Pyrite Scav. Tail **Test No.:** F1

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulativ	Cumulative
65	212	0.1	0.1	0.1	99.9
100	150	0.3	0.2	0.2	99.8
150	106	2.2	1.2	1.5	98.5
200	75	9.5	5.4	6.9	93.1
270	53	20.2	11.5	18.3	81.7
400	38	27.2	15.4	33.7	66.3
Pan	-38	116.9	66.3	100.0	0.0
Total	-	**176.4**	**100.0**	-	-
K80	**51**				



Test No: F2 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: 11/13/1996**

Purpose: Cu/Zn/Pyrite Float

Procedure:

Feed: 1000 grams of SKM-1 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill (NB).

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at start of stage	
	Lime	NaCN	SO2	A3418	R208	Flex-31	CuSO4	MIBC	Grind	Cond.	Froth	pH	Eh
Grind	1000	10	-	-	-	-	-	-	45			8.8	-170
Cu Aeration	200	-	450	-	-	-	-	-		20		8.1	+75
Cu Rougher	-	-	-	10	20	-	-	25		1	5	7.8	+80
Cu 1st Clnr	-	-	-	-	-	-	-			1	3	7.5	+100
Cu 2nd Clnr	-	-	100	-	-	-	-			1	3	7.2	+100
Cu 3rd Clnr	-	-	200	-	-	-	-	10		1	2	4.6	+180
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	740	-	-	-	-	-	200	-		4		10.5	+20
Zn Rougher	260	-	-	-	-	50	-			1	3	10.5	+0
Zn Scavenger	120	-	-	-	-	50	-			1	2	10.5	-10
Zinc cleaning on rougher concentrate													
Zn 1st Clnr	1740	-	-	-	-	-	-	10		1	3	11.0	-20
Zn 2nd Clnr	750	-	-	-	-	-	-	10		1	2	11.0	-45
Zn 3rd Clnr	750	-	-	-	-	-	-	5		1	2	11.0	+80
Pyrite Flotation on Zn Rougher Tails													
	H2SO4												
Pyrite Rougher 1	1000	-	-	-	-	100	-	10		1	1	7.0	+70
Pyrite Rougher 2	-	-	-	-	-	-	-	25		1	3	7.3	+70
Pyrite Rougher 3	-	-	-	-	-	50	-	10		1	3	7.4	+65
Pyrite Rougher 4	-	-	-	-	-	100	-	30		1	4	7.6	+50

Stage	Cu, Zn, Pyrite Roughers	Cu, Zn, Pyrite Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F2 Project: Braddick-Grong, 5031 Operator: F.V.

Metallurgical Balance

Product	Weight grams	Weight %	Assays, %, g/t Cu	Zn	Fe	Pb	% Distribution Cu	Zn	Fe	Pb
Cu 3rd Clnr Conc	45.2	4.6	28.10	0.32	32.3	1.06	79.7	5.0	5.2	51.6
Cu 3rd Clnr Tail	12.1	1.2	6.0	0.29	35.4	0.72	4.5	1.2	1.5	9.4
Cu 2nd Clnr Tail	21.6	2.2	2.2	0.27	34.1	0.33	2.9	2.0	2.6	7.7
Cu 1sr Clnr tail	35.8	3.6	2.0	0.36	31.2	0.17	4.5	4.5	4.0	6.6
Zn 3rd Clnr Conc	10.0	1.0	2.0	10.80	44.9	0.048	1.3	37.6	1.6	0.5
Zn 3rd Clnr Tail	22.1	2.2	0.5	1.22	49.3	0.042	0.7	9.4	3.9	1.0
Zn 2nd Clnr Tail	20.2	2.0	0.5	0.44	47.1	0.056	0.7	3.1	3.4	1.2
Zn 1st Clnr Tail	259.6	26.3	0.1	0.32	51.9	0.032	1.5	28.9	48.4	8.9
Zn Scav Conc	54.1	5.5	0.3	0.10	52.5	0.035	0.8	1.9	10.2	2.0
Pyrite Ro Conc 1	13.3	1.3	0.3	0.08	19.3	0.042	0.2	0.4	0.9	0.6
Pyrite Ro Conc 2	26.8	2.7	0.5	0.15	39.5	0.051	0.8	1.4	3.8	1.5
Pyrite Ro Conc 3	18.1	1.8	0.4	0.10	25.8	0.045	0.4	0.6	1.7	0.9
Pyrite Ro Conc 4	29.4	3.0	0.3	0.11	49.4	0.043	0.6	1.1	5.2	1.4
Pyrite Ro Tail	419.1	42.4	0.1	0.02	4.9	0.015	1.4	2.8	7.4	6.8
Head (Calc)	987.4	100.0	1.61	0.29	28.2	0.09	100.0	100.0	100.0	100.0
Head (direct)			**1.60**	**0.35**	**30.9**	**0.10**				

Calculated Grades and Recoveries

Cu 3rd Clnr Conc	4.6	28.1	0.3	32.3	1.06	79.7	5.0	5.2	51.6
Cu 2nd Clnr Conc	5.8	23.4	0.3	33.0	0.99	84.2	6.3	6.8	61.0
Cu 1st Clnr Conc	8.0	17.6	0.3	33.3	0.81	87.1	8.3	9.4	68.6
Cu Rougher Conc	11.6	12.7	0.3	32.6	0.61	91.6	12.8	13.4	75.2
Zn 3rd Clnr Conc	1.0	2.0	10.8	44.9	0.048	1.3	37.6	1.6	0.5
Zn 2nd Clnr Conc	3.3	1.0	4.2	47.9	0.044	1.9	47.0	5.5	1.5
Zn 1st Clnr Conc	5.3	0.8	2.8	47.6	0.049	2.6	50.1	8.9	2.7
Zn Rougher Conc	31.6	0.2	0.7	51.2	0.035	4.1	79.0	57.4	11.7
Zn Rougher Feed	88.4	0.2	0.3	27.6	0.026	8.4	87.2	86.6	24.8
Zn Scav Conc.	5.5	0.3	0.1	52.5	0.035	0.8	1.9	10.2	2.0
Pyrite Ro Conc 1	1.3	0.3	0.1	19.3	0.042	0.2	0.4	0.9	0.6
Pyrite Ro Conc 1-2	4.1	0.4	0.1	32.8	0.048	1.0	1.8	4.7	2.1
Pyrite Ro Conc 1-3	5.9	0.4	0.1	30.6	0.047	1.4	2.4	6.4	3.0
Pyrite Ro Conc 1-4	8.9	0.4	0.1	36.9	0.046	2.0	3.5	11.6	4.3
Py Ro Conc 1-4 & Zn Scav Con	14.4	0.3	0.1	42.9	0.042	2.9	5.4	21.8	6.4
Pyrite Rougher Feed	51.3	0.1	0.0	10.4	0.020	3.4	6.3	19.0	11.1

LR-5031
Braddick-Grong

Test No: F2

Eh Profile

Time (min)	Condition	Eh	pH
0	Out of Grinding Mill	-120	8.8
	After Lime Addition	-170	10.1
	After SO2	-170	9.0
1	Air on	+20	8.9
2		+40	8.8
3		+50	8.7
5		+60	8.6
5.5	After air off for 30 sec	+40	8.6
10		+70	8.5
10.5	After air off for 30 sec	+50	8.4
15		+75	8.3
15.5	After air off for 30 sec	+70	8.3
20		+80	8.2
20.5	After air off for 30 sec	+75	8.1

Test No: F2
Project: Braddick-Grong, 5031
Operator: F.V.
Date: 11/12/1996

Objective:

Initial flotation test, based on Kidd Creek Reagent Scheme
Produce three concentrates: Copper, Zinc and Pyrite.

Observations:

Copper Flotation:

black-grey colored froth in roughers, especially at end of flotation time, very small bubbles
Mineralization only at the edge of the froth and does not last very long
A lot of shiny particles under the microscope in the 1st Cl tail
Cu mineralzation appears after 15-30 sec of flotation (black froth ar first) and dissapears after 2 mins of frothing
Nice stable froth

Zinc Flotation:

Pyrite coloration after conditioning, very bright color after collector is added in rougher
Scavenger was tested to eliminate zinc from pyrite concentrate
very shiny colored froth in cleaners that floats for about 2 mins then becomes dark
Under the microscope, again a lot of shiny particles left in the 1st Cl tail
During filtering, a little bit of zinc (green material) was visible, some copper (red) and a lot of pyrite (brown and black)

Pyrite flotation:

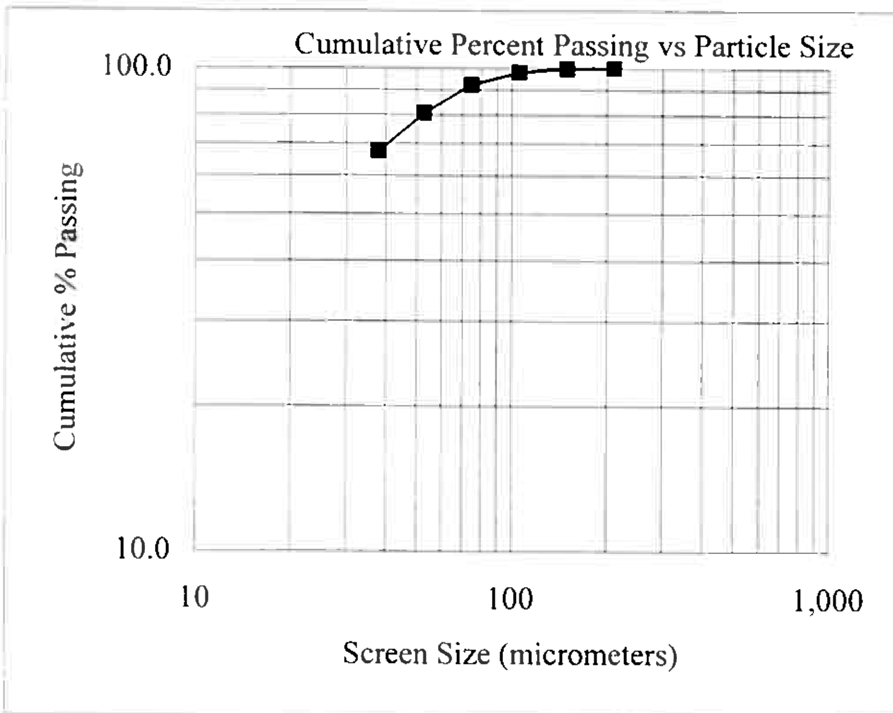
Very dark froth with slow floating, still pulling at end of 3 minutes targeted.
Froth not very stable despite additional collector and froth in subsequent roughers.
Galena hue visible in froth after second collector addition.
black-grey color in cleaners, stable froth, not a lot of of material in cleaners
A big cake left in the rougher tail

Company | **Lakefield Research** | **LR-5031**

Size Distribution Analysis

Sample: Pyrite Ro Tail **Test No.:** F2

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulativ	Cumulative
65	212	0.1	0.1	0.1	99.9
100	150	0.7	0.4	0.5	99.5
150	106	2.9	1.8	2.3	97.7
200	75	8.9	5.6	7.9	92.1
270	53	18.2	11.4	19.4	80.6
400	38	21.5	13.5	32.9	67.1
Pan	-38	106.7	67.1	100.0	0.0
Total	-	**159.0**	**100.0**	-	-
K80	**55**				



Test No: F3 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: 11/14/1996**

Purpose: Cu/Zn/Pyrite Float using Na2SO3

Procedure:

Feed: 1000 grams of GO-1 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill (NB).

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at start of stage	
	Lime	Na_2CO_3	Na_2SO_3	A3418	R208	Flex-31	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH	Eh
Grind	1000	-	500	-	-	-	-	-	45			9.4	+30
Cu Aeration	-	-	-	-	-	-	-	-		20		8.9	+85
Cu Rougher	210	-	-	10	20	-	-	35		1	5	10.0	+80
Cu Rghr Conc Condition			100									10.0	+100
Cu 1st Clnr	90	-	-	-	5	-	-	15		1	3	10.5	+95
Cu 2nd Clnr	90	-	-	-	-	-	-	5		1	3	11.0	+90
Cu 3rd Clnr	90	100	-	-	-	-	-	5		1	3	11.0	+90
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	570	-	-	-	-	-	800	-		4		10.5	+40
Zn Rougher	90	-	-	-	-	30	-	5		1	3	10.5	+45
Zn Scavenger	290	-	-	-	-	50	-	5		1	2	10.5	+85
Zinc cleaning on rougher concentrate													
Zn 1st Clnr	110	-	-	-	-	-	-	15		1	3	11.0	+90
Zn 2nd Clnr	60	-	-	-	-	-	-	5		1	2	11.0	+90
Zn 3rd Clnr	180	100	-	-	-	-	-	5		1	2	11.0	+90
Pyrite Flotation on Zn Rougher Tails													
	H2SO4												
Pyrite Rougher 1	1000	-	-	-	-	100	-			1	1	7.0	+190
Pyrite Rougher 2	-	-	-	-	-	100	-			1	3	7.6	+145
Pyrite Rougher 3	-	-	-	-	-	50	-	5		1	3	7.8	+130
Pyrite Rougher 4	-	-	-	-	-	100	-			1	4	7.9	+110

Stage	Cu, Zn, Pyrite Roughers	Cu, Zn, Pyrite Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F3 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: 11/14/1996**

Metallurgical Balance

Product	Weight		Assays, %, g/t						% Distribution					
	grams	%	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Clnr Conc	45.3	4.6	20.10	13.40	20.5	5.05	23.2	835	84.2	4.2	5.4	60.1	58.5	70.5
Cu 3rd Clnr Tail	1.0	0.1	5.73	5.59	20.2	9.62			0.5	0.0	0.1	2.5		
Cu 2nd Clnr Tail	5.2	0.5	2.25	6.60	25.1	6.98			1.1	0.2	0.8	9.5		
Cu 1sr Clnr tail	22.3	2.3	0.93	6.25	37.1	0.92			1.9	1.0	4.8	5.4		
Zn 3rd Clnr Conc	183.2	18.5	0.09	64.70	0.96	0.021	0.16	13.3	1.5	82.2	1.0	1.0	1.6	4.5
Zn 3rd Clnr Tail	6.8	0.7	0.41	59.50	2.58	0.21			0.3	2.8	0.1	0.4		
Zn 2nd Clnr Tail	3.7	0.4	0.95	46.50	7.40	0.55			0.3	1.2	0.2	0.5		
Zn 1st Clnr Tail	8.7	0.9	1.11	31.70	11.9	0.63			0.9	1.9	0.6	1.4		
Zn Scav Conc	282.2	28.6	0.14	2.07	44.5	0.08			3.7	4.1	72.8	6.1		
Pyrite Ro Conc 1	17.1	1.7	0.92	4.66	34.6	0.42			1.5	0.6	3.4	1.9		
Pyrite Ro Conc 2	32.0	3.2	0.75	3.14	31.2	0.45			2.2	0.7	5.8	3.8		
Pyrite Ro Conc 3	8.4	0.9	0.69	2.53	23.2	0.50			0.5	0.1	1.1	1.1		
Pyrite Ro Conc 4	12.1	1.2	0.41	1.75	15.2	0.40			0.5	0.1	1.1	1.3		
Pyrite Ro Tail	359.7	36.4	0.03	0.3	1.38	0.052	0.32	7.75	1.0	0.8	2.9	4.9	6.4	5.2
Head (Calc)	987.7	100.0	1.10	14.60	17.47	0.39			100.0	100.0	100.0	100.0		
Head (direct)			**1.00**	**15.7**	**17.9**	**0.39**	**1.82**	**54.30**					100.0	100.0

Calculated Grades and Recoveries

Cu 3rd Clnr Conc	4.6	20.1	13.4	20.5	5.05	23.2	835	84.2	4.2	5.4	60.1	58.5	70.5
Cu 2nd Clnr Conc	4.7	19.8	13.2	20.5	5.15			84.7	4.2	5.5	62.7		
Cu 1st Clnr Conc	5.2	18.0	12.6	21.0	5.33			85.8	4.5	6.3	72.2		
Cu Rougher Conc	7.5	12.9	10.7	25.8	4.00			87.7	5.5	11.1	77.6		
Zn 3rd Clnr Conc	18.5	0.090	64.7	0.96	0.021	0.16	13.3	1.5	82.2	1.0	1.0	1.6	4.5
Zn 2nd Clnr Conc	19.2	0.10	64.5	1.02	0.028			1.8	85.0	1.1	1.4		
Zn 1st Clnr Conc	19.6	0.12	64.2	1.14	0.038			2.1	86.2	1.3	1.9		
Zn Rougher Conc	20.5	0.16	62.8	1.60	0.063			3.0	88.1	1.9	3.4		
Zn Rougher Feed	92.5	0.15	14.9	16.8	0.093			12.3	94.5	88.9	22.4		
Zn Scavenger Conc	28.6	0.14	2.07	44.5	0.000			3.7	4.1	72.8	6.1		
Pyrite Ro Conc 1	1.7	0.92	4.66	34.6	0.42			1.5	0.6	3.4	1.9		
Pyrite Ro Conc 1-2	5.0	0.81	3.67	32.4	0.44			3.7	1.2	9.2	5.7		
Pyrite Ro Conc 1-3	5.8	0.79	3.50	31.0	0.45			4.2	1.4	10.3	6.8		
Pyrite Ro Conc 1-4	7.0	0.73	3.20	28.3	0.44			4.7	1.5	11.4	8.0		
Py Ro Conc 1-4 & Zn Scav Conc	35.6	0.26	2.29	41.3	0.087	0.73	24.0	8.3	5.6	84.2	14.1	14.3	15.7
Pyrite Rougher Feed	43.5	0.14	0.80	5.74	0.11			5.7	2.4	14.3	13.0		

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Braddick-Grong

Test No: F3

Eh Profile

Time (min)	Condition	Eh	pH
0	Out of Grinding Mill	+30	9.4
1	Air on	+60	9.4
2		+60	9.4
3		+60	9.4
5		+65	9.4
5.5	After air off for 30 sec	+60	9.4
10		+70	9.3
10.5	After air off for 30 sec	+65	9.3
15		+80	9.1
15.5	After air off for 30 sec	+80	9.1
20		+90	8.9
20.5	After air off for 30 sec	+85	8.9

Observations

Test No: F3
Project: Braddick-Grong, 5031
Operator: F.V.
Date: 11/14/1996

Objective:

Initial flotation test, based on Kidd Creek Reagent Scheme
Produce three concentrates: Copper, Zinc and Pyrite.

Observations:

Copper Flotation:

Very nice float but pyrite seems to float easily
Very shiny mineralization which disappears after 2 minutes of froth
Cleaned and almost white 1st Cl tail compared to F1

Zinc Flotation:

Black froth for the first 30 sec, mineralization shows up after
Zinc is floating slowly; longer rougher time?
Additional collector leads to pyrite flotation and no increase in zinc coloration; i.e. zinc not undercollected.
there still too much zinc left in rougher tail, enough to float a scavenger conc
froth in cleaners very thick that stands on its own for the first minute, after that frother needed because the froth is made up of big bubbles bursting at the rim

Pyrite flotation:

Slow floating, not much of material
Added additional collector

Test No: F4 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: 11/15/1996**

Purpose: Cu/Zn/Pyrite Float using Na2SO3

Procedure:

Feed: 1000 grams of SKM-1 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill.

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at end of stage	
	Lime	Na_2CO_3	Na_2SO_3	A3418	R208	Flex-31	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH	Eh
Grind	1500	-	750	-	-	-	-	-	45			10.1	-145
Cu Aeration	-	-	-	-	-	-	-	-		20		9.1	+50
Cu Rougher	240	-	-	10	10	-	-	25		1	5	10.0	+30
Cu Rghr Conc Condition	360		200							4		10.0	-40
Cu 1st Clnr	10	-	-	-	5	-	-	10		1	3	10.5	-50
Cu 2nd Clnr	140	-	-	-	-	-	-	15		1	3	11.0	-10
Cu 3rd Clnr	180	100	-	-	-	-	-	15		1	3	11.0	+80
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	270	-	-	-	-	-	200	-		4		10.5	+10
Zn Rougher	120	-	-	-	-	10	-	10		1	3	10.5	-30.0
Zn Scavenger	100	-	-	-	-	30	-			1	5	10.5	+0
Zinc cleaning on rougher concentrate													
Zn 1st Clnr	190	-	-	-	-	-	-	15		1	3	11.0	+75
Zn 2nd Clnr	220	-	-	-	-	-	-	15		1	2	11.0	+0
Zn 3rd Clnr	200	100	-	-	-	-	-	10		1	2	11.0	+0
Pyrite Flotation on Zn Rougher Tails													
	H2SO4												
Pyrite Rougher 1	720	-	-	-	-	50	-	15		1	1	7.0	+30
Pyrite Rougher 2	-	-	-	-	-	50	-	10		1	3	7.5	+10
Pyrite Rougher 3	-	-	-	-	-	50	-	15		1	3	7.8	+0
Pyrite Rougher 4	-	-	-	-	-	50	-	15		1	6	7.8	+5

Stage	Cu, Zn, Pyrite Roughers	Cu, Zn, Pyrite Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F4 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: 11/15/1996**

Metallurgical Balance

Product	Weight grams	Weight %	Assays, %, g/t Cu	Zn	Fe	Pb	Au	Ag	% Distribution Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Clnr Conc	48.3	4.8	27.0	0.32	31.9	0.55			84.4	5.28	5.46	28.36		
Cu 3rd Clnr Tail	9.3	0.9	4.90	0.28	32.4	1.87			2.95	0.89	1.07	18.56		
Cu 2nd Clnr Tail	12.9	1.3	3.07	0.33	31.2	1.04			2.56	1.45	1.43	14.32		
Cu 1st Clnr tail	27.3	2.7	1.72	0.36	29.2	0.30			3.04	3.36	2.83	8.74		
Zn 3rd Clnr Conc	29.2	2.9	0.81	1.54	45.9	0.069			1.53	15.4	4.75	2.2		
Zn 3rd Clnr Tail	20.5	2.1	0.34	0.70	47.9	0.061			0.45	4.90	3.48	1.33		
Zn 2nd Clnr Tail	36.4	3.7	0.37	0.85	50.2	0.080			0.87	10.6	6.48	3.1		
Zn 1st Clnr Tail	176.2	17.7	0.11	0.76	53.4	0.043			1.25	45.8	33.4	8.1		
Zn Scav Conc	130.4	13.1	0.16	0.18	53.2	0.039			1.35	8.02	24.6	5.43		
Pyrite Ro Conc 1	10.8	1.1	0.13	0.079	50.5	0.041			0.091	0.29	1.93	0.47		
Pyrite Ro Conc 2	25.3	2.5	0.16	0.094	43.0	0.049			0.26	0.81	3.86	1.32		
Pyrite Ro Conc 3	12.9	1.3	0.20	0.094	33.5	0.051			0.17	0.41	1.53	0.70		
Pyrite Ro Conc 4	16.1	1.6	0.17	0.084	26.9	0.047			0.18	0.46	1.54	0.81		
Pyrite Ro Tail	441.3	44.3	0.031	0.016	4.89	0.014			0.89	2.41	7.65	6.60		
Head (Calc)	996.9	100.0	1.55	0.29	28.28	0.094			100.0	100.0	100.0	100.0		
Head (direct)			**1.60**	**0.35**	**30.9**	**0.10**	**1.39**	**19.50**						

Calculated Grades and Recoveries

Product		%	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Clnr Conc		4.85	27.0	0.32	31.9	0.55			84.4	5.28	5.46	28.36		
Cu 2nd Clnr Conc		5.78	23.4	0.31	32.0	0.76	11.8	170	87.4	6.17	6.53	46.92	49.05	50.4
Cu 1st Clnr Conc		7.07	19.7	0.32	31.8	0.81			89.9	7.63	7.96	61.24		
Cu Rougher Conc		9.81	14.7	0.33	31.1	0.67			93.0	11.0	10.8	70.0		
Zn 3rd Clnr Conc		2.93	0.81	1.54	45.9	0.069			1.53	15.4	4.75	2.2		
Zn 2nd Clnr Conc		4.99	0.62	1.19	46.7	0.066			1.98	20.3	8.24	3.5		
Zn 1st Clnr Conc		8.64	0.51	1.05	48.2	0.072			2.85	30.8	14.7	6.6		
Zn Rougher Conc		26.3	0.24	0.85	51.7	0.052			4.11	76.6	48.1	14.7		
Zn Rougher Feed		90.2	0.12	0.29	28.0	0.031			7.04	89.0	89.2	30.0		
Zn Scav Conc		13.1	0.16	0.18	53.2	0.039			1.35	8.02	24.6	5.43		
Pyrite Ro Conc 1		1.08	0.13	0.079	50.5	0.041			0.091	0.29	1.93	0.47		
Pyrite Ro Conc 1-2		3.62	0.15	0.090	45.2	0.047			0.35	1.10	5.79	1.80		
Pyrite Ro Conc 1-3		4.92	0.16	0.091	42.2	0.048			0.52	1.52	7.33	2.50		
Pyrite Ro Conc 1-4		6.53	0.17	0.089	38.4	0.048			0.70	1.98	8.86	3.31		
Py Ro Conc 1-4 & Zn Scav Con		19.6	0.16	0.15	48.3	0.042			2.05	10.0	33.5	8.7		
Pyrite Rougher Feed		50.8	0.048	0.025	2.20	0.018			1.58	4.39	16.5	9.90		

LR-5031
Braddick-Grong

Test No: F4

Eh Profile

Time (min)	Condition	Eh	pH
0	Out of Grinding Mill	-145	10.1
1	Air on	-10	10.0
2		+15	9.8
3		+35	9.7
5		+50	9.6
5.5	After air off for 30 sec	+35	9.6
10		+60	9.5
10.5	After air off for 30 sec	+50	9.5
15		+65	9.3
15.5	After air off for 30 sec	+40	9.3
20		+70	9.2
20.5	After air off for 30 sec	+50	9.1

Observations

Test No: F4
Project: Braddick-Grong, 5031
Operator: F.V.
Date: 11/15/1996

Objective:

Initial flotation test, based on Kidd Creek Reagent Scheme
Produce three concentrates: Copper, Zinc and Pyrite.

Observations:

Copper Flotation:

Stable nice froth black at first but shiny mineralization comes up nicely
Cu seems to float easily in the cleaners
much more copper material in the 3rd Clnr Conc compared than F2

Zinc Flotation:

Instant mineralization after conditioning, very bright color
nice and controllable froth with small bubbles for both rougher and scavenger
very shiny colored froth in cleaners that floats very easily
Still some mineralization visible in the tails but coming up slowly: longer frothing time?
much more zinc material in the 3rd Clnr Conc compared than F2

Pyrite flotation:

Very dark froth with slow floating, still pulling at end of 3 minutes targeted.
Galena (or some left-over zinc) hue visible in froth
Not a lot of material in the 1st rougher concentrate
hue seems to have disappears at the 3rd rougher concentrate

Test No: F5 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: 11/25/1996**

Purpose: Cu/Zn Float using Na2SO3, optimisation of conditions test 1

Procedure:

Feed: 1000 grams of GO-1 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill (NB).

Conditions:

Stage	Reagent added, g/t									Time, minutes			Measured at start of stage
	Lime	Dextrin 1752S	Na_2SO_3	A3418	R208	Flex-31	$CuSO_4$	ZnSO4/ NaCN	MIBC	Grind	Cond.	Froth	pH
Grind	1500	-	750	-	-	-	-	-	-	45			11.2
Cu Aeration	-	-	-	-	-	-	-	-	-		20		10.3
Cu Rougher - 1		-	-	5	10	-	-	-	40		1	3	10.0
Cu Rougher - 2	170			10	20				25		1	4	10.1
comb. Rghr Conc Cond	70		100								4		10.0
Cu 1st Clnr	140	-	-	-	5	-	-	-	20		1	3	10.5
Cu 2nd Clnr	160	50	-	-	-	-	-	-	25		1	3	11.0
Cu 3rd Clnr	130	100	-	-	-	-	-	-	15		1	3	11.0
Cu 4th Clnr	270	-	-	-	-	-	-	200	15		1	3	11.0
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	310	-	-	-	-	-	800		-		4		10.5
Zn Rougher - 1	340	-	-	-	-	20	-		25		1	2	10.5
Zn Rougher - 2	170					25			25		1	4	10.5
Zn Rougher - 3	180					20					1	3	10.5
Zn Scavenger	220	-	-	-	-	20	-		10		1	2	10.5
Zinc cleaning on the combined rougher concentrate													
Zn 1st Clnr	410	-	-	-	-	-	-		25		1	3	11.0
Zn 2nd Clnr	130	-	-	-	-	-	-		15		1	2	11.0
Zn 3rd Clnr	270	-	-	-	-	-	-				1	2	11.0

Stage	Cu, Zn Roughers	Zn Clnr 1	Cu, Zn Clnrs
Flotation Cell	1000gD1	500gD1	250gD1
Speed: r.p.m.	1800	1500	1200

Test No: F5 **Project: Braddick-Grong, 5031** **Operator: F.V.**

Metallurgical Balance

Product	Weight grams	Weight %	Assays, %, g/t Cu	Zn	Fe	Pb	% Distribution Cu	Zn	Fe	Pb
Cu 4th Clnr Conc	1.4	0.1	31.7	3.13	27.2	1.88	4.2	0.0	0.2	0.7
Cu 4th Clnr Tail	7.3	0.7	29.7	5.33	26.0	1.55	20.6	0.3	1.1	2.9
Cu 3rd Clnr Tail	5.5	0.6	22.4	10.5	22.2	4.33	11.7	0.4	0.7	6.1
Cu 2nd Clnr Tail	31.1	3.1	16.0	10.6	17.8	7.68	47.2	2.2	3.1	61.2
Cu 1st Clnr tail	10.5	1.1	2.83	16.3	14.9	2.40	2.8	1.1	0.9	6.5
Zn 3rd Clnr Conc	177.4	17.8	0.22	66.0	1.30	0.04	3.7	78.1	1.3	2.0
Zn 3rd Clnr Tail	16.8	1.7	0.42	61.6	2.85	0.13	0.7	6.9	0.3	0.6
Zn 2nd Clnr Tail	13.6	1.4	0.95	44.4	11.2	0.34	1.2	4.0	0.9	1.2
Zn 1st Clnr Tail	41.0	4.1	0.87	17.1	28.0	0.45	3.4	4.7	6.5	4.7
Zn Scav Conc	97.4	9.8	0.08	0.43	46.5	0.05	0.7	0.3	25.5	1.3
Zinc Scav Tail	592.9	59.6	0.07	0.51	17.9	0.09	3.8	2.0	59.7	12.9
Head (Calc)	994.9	100.0	1.06	15.1	17.9	0.39	100.0	100.0	100.0	100.0
Head (direct)			**1.00**	**15.7**	**17.9**	**0.39**				

Calculated Grades and Recoveries

Cu 4th Clnr Conc	0.1	31.7	3.1	27.2	1.9	4.2	0.0	0.2	0.7
Cu 3rd Clnr Conc	0.9	30.0	5.0	26.2	1.6	24.8	0.3	1.3	3.6
Cu 2nd Clnr Conc	1.4	27.1	7.1	24.6	2.7	36.4	0.7	2.0	9.7
Cu 1st Clnr Conc	4.6	19.5	9.5	19.9	6.1	83.6	2.9	5.1	70.9
Cu Rougher Conc	5.6	16.3	10.8	19.0	5.4	86.5	4.0	6.0	77.3
Zn 3rd Clnr Conc	17.8	0.22	66.0	1.30	0.04	3.7	78.1	1.3	2.0
Zn 2nd Clnr Conc	19.5	0.24	65.6	1.43	0.05	4.4	85.0	1.6	2.6
Zn 1st Clnr Conc	20.9	0.28	64.2	2.07	0.07	5.6	89.0	2.4	3.7
Zn Rougher Conc	25.0	0.38	56.5	6.35	0.13	9.0	93.7	8.9	8.5
Zn Scavenger Conc	9.8	0.08	0.43	46.5	0.05	0.7	0.3	25.5	1.3
Zn Rougher Feed	94.4	0.15	15.3	17.8	0.09	13.5	96.0	94.0	22.7

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Braddick-Grong

Test No: F5

Eh Profile

Time (min)	Condition	Eh	pH
0	Out of Grinding Mill	-20	11.2
1	Air on	-10	11.2
2		-10	11.2
3		-10	11.1
5		-10	10.9
5.5	After air off for 30 sec	-10	10.8
10		-5	10.6
10.5	After air off for 30 sec	+0	10.8
15		+0	10.5
15.5	After air off for 30 sec	+10	10.5
20		+15	10.2
20.5	After air off for 30 sec	+20	10.3

Test No: F5
Project: Braddick-Grong, 5031
Operator: F.V.
Date: 11/25/1996

Objective:

Optimisation of GO-1 sample.
Testing Dextrin for silicate depresssant and Zn/CN depressant for Zn.

Observations:

Copper Flotation:

Black-grey colored froth during roughing stage
copper mineralization visible at the bottom of the rougher concentrate
not much as much material as previous tests (F1 and F3)
same stable black froth in cleaners

Zinc Flotation:

Instant mineralization after conditioning, very bright color
nice and controllable froth with small bubbles for both rougher and scavenger
a lot of zinc pulled out: another roughing stage added to get as much zinc as possible
very shiny colored froth in cleaners that floats very easily
still some zinc mineralization visible in the froth after the allowed time but coming up slowly: longer frothing time?
lots of material in the last concentrate

Test No: F6 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: November 26, 1996**

Purpose: Cu/Zn Float on SKM-2 sample, using Na2SO3

Procedure:

Feed: 1000 grams of SKM-2 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill.

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at end of stage	
	Lime	$ZnSO_4$/NaCN	Na_2SO_3	Dextrin	CA821	Flex-31	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH	Eh
Grind	1500	-	750	-	-	-	-	-	45			8.8	-20
Cu Aeration	-	-	-	-	-	-	-	-		20		8.0	+80
Cu Rougher - 1	500	-	-	-	15	-	-	10		1	3	10.0	+10
Cu Rougher - 2	370	-	-	-	30	-	-	10		1	4	10.0	+20
Cu Rghr Conc Condition	340		200							4		10.0	+60
Cu 1st Clnr		-	-	-	5	-	-	5		1	3	11.0	+55
Cu 2nd Clnr	280	-	-	-	-	-	-	5		1	3	11.0	-15
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	450	-	-	-	-	-	200	-		4		10.5	+30
Zn Rougher	190	-	-	-	-	5	-	10		1	6	10.5	+40
Zinc cleaning on rougher concentrate													
Zn 1st Clnr	260	-	100	-	-	10	-			1	5	11.0	+50
Zn 2nd Clnr	130	-	-	-	-	5	-	5		1	4	11.0	+80
Zn 3rd Clnr	300	-	50	-	-	-	-	5		1	3	11.5	+75

Stage	Cu, Zn Roughers	Cu, Zn Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F6 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: November 26, 1996**

Metallurgical Balance

Product	Weight		Assays, %, g/t						% Distribution					
	grams	%	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 2nd Clnr Conc	32.2	3.23	25.2	11.6	27.0	0.19	5.35	93.0	80.5	18.86	2.62	11.94	45.42	13.3
Cu 2nd Clnr Tail	5.3	0.53	11.7	10.6	31.2	0.61			6.15	2.84	0.50	6.31		
Cu 1st Clnr tail	26.4	2.64	0.66	0.73	44.8	0.094			1.73	0.97	3.56	4.84		
Zn 3rd Clnr Conc	38.2	3.83	0.27	32.5	24.1	0.075	0.62	22.8	1.02	62.7	2.77	5.59	6.24	3.9
Zn 3rd Clnr Tail	1.1	0.11	1.60	14.8	31.6	1.04			0.17	0.82	0.10	2.23		
Zn 2nd Clnr Tail	0.3	0.03	4.41	22.1	20.6	0.30			0.13	0.3	0.02	0.18		
Zn 1st Clnr Tail	2.3	0.23	2.32	7.42	28.5	0.22			0.53	0.9	0.2	1.0		
Zn Rougher tail	892.4	89.4	0.11	0.28	33.6	0.039	0.26	13.5	9.74	12.62	90.2	67.9	61.2	53.6
Head (Calc)	998.2	100.0	1.01	1.98	33.3	0.051			100.0	100.0	100.0	100.0		
Head (direct)			**0.97**	**1.90**	**33.7**	**0.05**	**0.38**	**22.50**					100.0	100.0

Calculated Grades and Recoveries

Cu 2nd Clnr Conc		3.2	25.2	11.6	27.0	0.19	5.35	93.0	80.5	18.86	2.62	11.94	45.4	13.3
Cu 1st Clnr Conc		3.8	23.3	11.5	27.6	0.25			86.7	21.70	3.11	18.25		
Cu Rougher Conc		6.4	13.9	7.03	34.7	0.19			88.4	22.7	6.7	23.1		
Zn 3rd Clnr Conc		3.8	0.27	32.5	24.1	0.075	0.62	22.8	1.02	62.7	2.77	5.59	6.24	3.88
Zn 2nd Clnr Conc		3.9	0.31	32.0	24.3	0.10			1.20	63.5	2.88	7.82		
Zn 1st Clnr Conc		4.0	0.34	31.9	24.3	0.10			1.33	63.8	2.89	8.00		
Zn Rougher Conc		4.2	0.45	30.6	24.5	0.11			1.86	64.7	3.09	8.99		
Zn Rougher Feed		93.6	0.13	1.64	33.2	0.042			11.6	77.3	93.3	76.9		

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Braddick-Grong

Test No: F6

Eh Profile

Time (min)	Condition	Eh	pH
0	Out of Grinding Mill	-20	8.8
1	Air on	+50	8.8
2		+50	8.8
3		+50	8.8
5		+50	8.8
5.5	After air off for 30 sec	+20	8.8
10		+75	8.4
10.5	After air off for 30 sec	+60	8.4
15		+80	8.3
15.5	After air off for 30 sec	+80	8.2
20		+80	8.0
20.5	After air off for 30 sec	+80	8.0

Test No: F6
Project: Braddick-Grong, 5031
Operator: F.V.
Date: November 26, 1996

Objective:

First test on the SKM-2 sample
Based on test conditions for SKM-1

Observations:

Copper Flotation:

Some zinc mineralization shows up at the rougher 1: too much collector?
Stable nice froth black with a shadow of shiny mineralization (copper or pyrite?)
copper mineralization visible at the bottom of the rougher concentrate
Cu seems to float easily in the cleaners
addition of dextrin killed any mineralization (white froth) at the third cleaner
not a lot of material was recovered

Zinc Flotation:

Instant brown mineralization after conditioning so collector dosage cut in two not too float any pyrite
nice brown-burgundy colored froth in rougher 1 that floats easily: increase frothing time from 3 to 6 minutes to get as much zinc as possible
at the 2nd rougher, with half the collector dosage specified of 20g/t, instant bright and shiny pyrite mineraliza
no more rougher or scavenger stage was done: increase the retention time of the first rougher to more than 6 minutes
Same colored froth in cleaners that floats very easily
the 1st clnr tail seems to be only pyrite, but zinc start to appear in the subsequent cleaner tails

Test No: F7 **Project: Braddick-Grong, 5031** **Operator:** **Date: December 3, 1996**

Purpose: As test F6, without aeration and depressants in cleaning.

Procedure: As described below.

Feed: 1000 grams of SKM-2 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill.

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at end of sta
	Lime	$ZnSO_4$/NaCN	Na_2SO_3	Dextrin	A3418	R208	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH
Grind	1500	-	750	-	-	-	-	-	45			8.2
Cu Rougher - 1	750	-	-	-	5	-	-	10		1	3	10.0
Cu Rougher - 2	500	-	-	-	5	10	-	5		1	4	10.0
Cu Rghr Conc Condition	200		200							4		10.0
Cu 1st Clnr	200	-	-	-	5	-	-			1	3	11.0
Cu 2nd Clnr		-	-	-	-	-	-	5		1	3	11.0
Cu 3rd Clnr		10	-	-	-	-	-			1	3	11.0
Cu 4th Clnr	80	20	-	-	-	-	-			1	3	11.0
Zinc Flotation on Cu Rougher Tails												
Zn Conditioning	1420	-	-	-	-	-	200	-		4		11.0
Zn Rougher		-	-	-	-	10	-	10		1	6	11.0
Zn Scavenger	600					10		10		1	3	11.0
Zinc cleaning on rougher concentrate												
Zn 1st Clnr	510	-	100	-	-		-			1	5	11.0
Zn 2nd Clnr	560	-	200	-	-	-	-			1	4	11.5
Zn 3rd Clnr	700		300							1	4	11.5
Zn 4th Clnr	650	-	300	-	-	-	-			1	3	11.5

Stage	Cu, Zn Roughers	Cu, Zn Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F7 **Project: Braddick-Grong, 5031** **Operator:** **Date: December 3, 1996**

Metallurgical Balance

Product	Weight grams	Weight %	Assays, %, g/t Cu	Zn	Fe	Pb	Au	Ag	% Distribution Cu	Zn	Fe	Pb	Au	Ag
Cu 4th Cl Conc	12.5	1.25	28.5	4.22	28.9	0.15	7.67	232	36.9	2.6	1.1	3.5	25.3	12.9
Cu 4th Clnr Tail	9.5	0.95	26.9	3.11	30.4	0.09			26.5	1.4	0.9	1.6		
Cu 3rd Clnr Tail	10.1	1.01	18.0	5.43	33.2	0.11			18.8	2.7	1.0	2.1		
Cu 2nd Clnr Tail	3.9	0.39	3.68	10.0	31.9	0.38			1.5	1.9	0.4	2.8		
Cu 1st Clnr tail	9.2	0.92	1.72	7.96	28.1	0.23			1.6	3.5	0.8	4.0		
Zn 4th Clnr Conc	27.9	2.79	1.45	56.4	9.24	0.21	1.42	66.0	4.2	76.3	0.8	11.0	10.4	8.2
Zn 4th Clnr Tail	0.9	0.09	2.15	25.5	22.2	0.25			0.2	1.1	0.1	0.4		
Zn 3rd Clnr Tail	1.0	0.10	1.48	7.21	30.3	0.22			0.2	0.3	0.1	0.4		
Zn 2nd Clnr Tail	9.6	0.96	0.72	3.04	32.1	0.15			0.7	1.4	0.9	2.7		
Zn 1st Clnr Tail	14.7	1.47	0.39	1.84	31.0	0.12			0.6	1.3	1.4	3.3		
Zn Ro Scav Conc	19.1	1.91	0.82	2.09	31.6	0.11			1.6	1.9	1.8	3.9		
Zn Rougher tail	880.8	88.15	0.08	0.13	34.6	0.039			7.1	5.5	90.9	64.3		
Head (Calc)	999.2	100.0	0.97	2.07	33.5	0.053			100.0	100.0	100.0	100.0		
Head (direct)			**0.97**	**1.90**	**33.7**	**0.050**	**0.38**	**22.5**						

Calculated Grades and Recoveries

Product	Weight grams	Weight %	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 4th Clnr Conc		1.25	28.5	4.22	28.9	0.15			36.9	2.6	1.1	3.5		
Cu 3rd Clnr Conc		2.20	27.8	3.74	29.5	0.12			63.4	4.0	1.9	5.1		
Cu 2nd Clnr Conc		3.21	24.7	4.27	30.7	0.12			82.3	6.6	2.9	7.2		
Cu 1st Clnr Conc		3.60	22.4	4.89	30.8	0.15	3.73	137	83.8	8.5	3.3	10.0	35.4	21.9
Cu Rougher Conc		4.52	18.2	5.51	30.3	0.16			85.4	12.1	4.1	13.9		
Zn 4th Clnr Conc		2.79	1.45	56.4	9.24	0.21	1.42	66.0	4.2	76.3	0.8	11.0	10.4	8.2
Zn 3rd Clnr Conc		2.88	1.47	55.4	9.64	0.21			4.4	77.4	0.8	11.4		
Zn 2nd Clnr Conc		2.98	1.47	53.8	10.3	0.21			4.5	77.7	0.9	11.8		
Zn 1st Clnr Conc		3.94	1.29	41.4	15.6	0.20			5.3	79.1	1.8	14.5		
Zn Rougher Conc		5.41	1.04	30.7	19.8	0.18			5.9	80.4	3.2	17.8		
Zn Rougher Feed		95.5	0.15	1.90	33.7	0.048			14.6	87.9	95.9	86.1		

Test No: F8 **Project: Braddick-Grong, 5031** **Operator:** **Date: 12/5/1996**

Purpose: First flotation test on sample GO-2. Aeration eliminated.

Procedure: As described below.

Feed: 1000 grams of GO-2 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill (NB).

Conditions:

Stage	Reagent added, g/t									Time, minutes			Measured at start of stage
	Lime	Na_2SiO_3	Na_2SO_3	A3418	R208	Flex-31	$CuSO_4$	ZnSO4/ NaCN	MIBC	Grind	Cond.	Froth	pH
Grind	1000	-	500	-	-	-	-	-	-	45			9.3
Cu Rougher - 1	410	-	-	5	10	-	-	-	2.5		1	3	10.0
Cu Rougher - 2				10	20						1	4	10.0
Comb. Ro Conc Cond			150								4		
Cu 1st Clnr	500	-	-	-	5	-	-	-	2.5		1	3	10.5
Cu 2nd Clnr	500	-	-	-	-	-	-	-			1	3	11.5
Clnr Conc Cond x		-	-	-	-	-	-	10	2.5		1		11.5
x		-	-	-	-	-	-	10			1		11.5
x		-	-	-	-	-	-	10			1		11.5
Regrind Conc (PM)	200									20			
Cu 3rd Clnr		200	-	-	-	-	-	10			1	3	11.2
	No discernible Cu conc floated												
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	690	-	-	-	-	-	800		-		4		10.5
Zn Rougher - 1	870	-	-	-	-	5	-		5		1	2	10.5
Zn Rougher - 2	710					10					1	4	10.5
Zn Rougher - 3	630					20					1	3	10.5
Zn Scavenger		-	-	-	-	30	-		2.5		1	2	10.5
Zinc cleaning on the combined rougher concentrate													
Zn 1st Clnr	470	-	-	-	-	-	-				1	4	11.0
Zn 2nd Clnr	480	-	-	-	-	-	-				1	3	11.5
Zn 3rd Clnr	310	-	-	-	-	-	-		2.5		1	2	11.5

Stage	Cu, Zn Roughers	Zn Clnr 1	Cu, Zn Clnrs
Flotation Cell	1000gD1	500gD1	250gD1
Speed: r.p.m.	1800	1500	1200

x No rejection of assumed Zn mineralization

Test No: F8 Project: Braddick-Grong, 5031 Operator: Date: 12/5/1996

Metallurgical Balance

Product	Weight		Assays, %, g/t						% Distribution					
	grams	%	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Clnr Conc	39.8	4.0	21.9	7.14	19.8	1.42	11.1	346	25.3	1.5	5.7	13.4	53.7	27.7
Cu 3rd Clnr Tail	67.3	6.8	26.9	9.0	24.4	1.58			52.4	3.3	11.9	25.2		
Cu 2nd Clnr Tail	25.2	2.5	13.1	18.0	16.3	2.48			9.6	2.4	3.0	14.8		
Cu 1st Clnr tail	32.5	3.3	4.33	27.1	11.7	2.93			4.1	4.8	2.8	22.6		
Cu 2nd&3rd Clnr Tl		9.3					1.79	219					20.1	40.7
Zn 3rd Clnr Conc	248.5	25.1	0.33	62.1	2.91	0.10	0.23	54.8	2.4	83.3	5.3	5.7	7.0	27.4
Zn 3rd Clnr Tail	9.8	1.0	2.15	19.8	16.5	0.55			0.6	1.0	1.2	1.3		
Zn 2nd Clnr Tail	21.8	2.2	1.74	8.8	20.3	0.48			1.1	1.0	3.2	2.5		
Zn 1st Clnr Tail	23.5	2.4	0.98	3.0	13.3	0.36			0.7	0.4	2.3	2.0		
Zn Scav Conc	37.3	3.8	1.15	5.36	20.9	0.33			1.2	1.1	5.7	2.9		
Zinc Scav Tail	483.8	48.9	0.19	0.44	16.8	0.09			2.7	1.1	59.0	9.7		
Head (Calc)	989.5	109.3	3.19	17.1	12.7	0.39			100.0	100.0	100.0	100.0		
Head (direct)			**3.20**	**19.3**	**14.3**	**0.41**	**0.76**	**46.0**						

Calculated Grades and Recoveries

Cu 3rd Clnr Conc		4.0	21.9	7.1	19.8	1.4	11.1	346	25.3	1.5	5.7	13.4	53.7	27.7
Cu 2nd Clnr Conc		10.8	25.0	8.3	22.7	1.5			77.7	4.8	17.7	38.6		
Cu 1st Clnr Conc		13.4	22.8	10.1	21.5	1.7	4.59	257	87.3	7.2	20.6	53.4	73.9	68.4
Cu Rougher Conc		16.7	19.1	13.5	19.5	1.9			91.3	12.0	23.4	75.9		
Zn 3rd Clnr Conc		25.1	0.33	62.1	2.91	0.10	0.23	54.80	2.4	83.3	5.3	5.7	7.0	27.4
Zn 2nd Clnr Conc		26.1	0.40	60.5	3.43	0.11			3.0	84.4	6.4	6.9		
Zn 1st Clnr Conc		28.3	0.50	56.5	4.74	0.14			4.1	85.4	9.6	9.4		
Zn Rougher Conc		30.7	0.54	52.3	5.40	0.16			4.8	85.8	11.9	11.4		
Zn Scavenger Conc		3.8	1.15	5.36	20.9	0.33			1.2	1.1	5.7	2.9		
Zn Rougher Feed		83.3	0.36	19.8	12.8	0.12			8.7	88.0	76.6	24.1		

Test No: F9 **Project: Braddick-Grong, 5031** **Operator:** **Date: December 17, 1996**

Purpose: As test F7, reduced depressants in Cu Ro and Zn Cl stages..

Procedure: As described below.

Feed: 1000 grams of SKM-2 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill.

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at start of stage	
	Lime	$ZnSO_4$/ NaCN	Na_2SO_3	QHS	A3418	R208	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH	eH
Grind	1500	-	250	-	-	-	-	-	45			8.2	-340
Cu Rougher - 1	280	-	-	-	5	-	-	5		1	3	10.0	-400
Cu Rougher - 2	660	-	-	-	5		-	2.5		1	4	10.0	50
Cu Rougher - 3	280				5					1	3	10.0	40
Cu Rghr Conc Condition	330		200							4		11.0	-20
Cu 1st Clnr	260	-	200	-	5	-	-			1	3	11.0	-20
Cu 2nd Clnr	440	-	900	-	-	-	-	2.5		1	3	11.5	
Cu 3rd Clnr	150		-	30	-	-	-			1	3	11.0	-90
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	1200	-	-	-	-	-	200	-		4		11.0	-70
	200						100			3		11.0	
Zn Rougher		-	-	-	-	10	-	5		1	6	11.0	
	870					10		5		1	3	11.0	-90
Zn Scavenger	600					10		2.5		1	3	11.0	
Zinc cleaning on rougher concentrate													
Zn 1st Clnr	460	-	300	-	-		-			1	5	11.0	-110
Zn 2nd Clnr	680	-		-	-	-	-			1	4	12.0	-50
Zn 3rd Clnr	500			10		5				1	4	11.5	

Stage	Cu, Zn Roughers	Cu, Zn Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F9 **Project: Braddick-Grong, 5031** **Operator:** **Date: December 17, 1996**

Metallurgical Balance

Product	Weight		Assays, %, g/t						% Distribution					
	grams	%	Cu	Zn	Fe	Pb	Au *	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Cl Conc	24.4	2.47	30.9	2.81	30.7	0.05	77.1	266	74.4	3.5	2.3	2.3	501	29.2
Cu 3rd Clnr Tail	30.7	3.11	4.17	6.75	38.0	0.16			12.6	10.5	3.6	9.4		
Cu 2nd Clnr Tail	46.0	4.65	0.40	1.37	44.0	0.15			1.8	3.2	6.3	13.2		
Cu 1st Clnr Tail	46.7	4.72	0.46	2.92	36.7	0.11			2.1	6.9	5.4	9.9		
Zn 3rd Clnr Conc	12.5	1.26	1.34	45.0	12.1	0.11			1.7	28.5	0.5	2.6		
Zn 3rd Clnr Tail	9.4	0.95	1.39	16.2	24.6	0.13			1.3	7.7	0.7	2.3		
Zn 2nd Clnr Tail	23.8	2.41	0.50	7.40	29.1	0.094			1.2	8.9	2.2	4.3		
Zn 1st Clnr Tail	82.0	8.30	0.20	3.99	29.5	0.072			1.6	16.6	7.6	11.3		
Zn Ro Scav Conc	35.5	3.59	0.13	2.02	30.1	0.062			0.5	3.6	3.3	4.2		
Zn Rougher tail	677.4	68.54	0.04	0.31	32.2	0.031			2.9	10.6	68.1	40.3		
Head (Calc)	988.4	100.0	1.03	2.00	32.4	0.053			100.0	100.0	100.0	100.0		
Head (direct)			**0.97**	**1.90**	**33.7**	**0.050**	**0.38**	**22.5**						

* Gold assay of Cu 3rd Clnr Conc. was reassayed to the same value: Possible contamination?

<u>Calculated Grades and Recoveries</u>

Cu 3rd Cl Conc		2.47	30.9	2.81	30.7	0.05	77.10	266	74.4	3.5	2.3	2.3	500.9	29.2
Cu 2nd Clnr Conc		5.57	16.0	5.01	34.8	0.11			87.0	14.0	6.0	11.8		
Cu 1st Clnr Conc		10.23	8.9	3.35	39.0	0.13			88.8	17.2	12.3	25.0		
Cu Rougher Conc		14.95	6.2	3.21	38.3	0.12			90.9	24.1	17.7	34.9		
Zn 3rd Clnr Conc		1.3	1.3	45.0	12.1	0.1			1.7	28.5	0.5	2.6		
Zn 2nd Clnr Conc		2.22	1.36	32.6	17.5	0.12			2.9	36.2	1.2	5.0		
Zn 1st Clnr Conc		4.62	0.91	19.5	23.5	0.11			4.1	45.1	3.4	9.3		
Zn Rougher Conc		12.92	0.46	9.5	27.4	0.08			5.7	61.7	10.9	20.6		
Zn Rougher Feed		85.0	0.11	1.78	31.4	0.040			9.1	75.9	82.3	65.1		

Observations

Test No: F9
Project: Braddick-Grong, 5031
Operator:
Date: December 17, 1996

Objective:

2nd test on the SKM-2 sample
Based on test conditions of F7

Observations:

Copper Flotation:

Py mineralization apparent in Cu circuit, needs the Na2SO3 in the grind
Floated Ro as per conditions, attempt to depress in cleaning circuit
Increased additions of Na2SO3 in cleaners do nothing, we need to hit Py in the Ro's, once it is activated, very difficult to depress
Elevated pH drops some Fe sulphide, believe not enough to gain grade
Use QHS in conjunction with high pH to depress Py, seems to work, try earlier?
Attempt to clean 3rd Conc with QHS, NaCN sol'n, resulted in depression of entire float
Added 10 g/t

Zinc Flotation:

Carry over of lack of depressant from the Cu circuit leads to flotation of Py with the Zn rougher.
Extended Ro float time due to slower kinetics, possibly from the decrease of collector in Cu Ro
No R 208, only 3418A
Attempt to drop Py out with Na2SO3 didn't produce amiable results, in 1st Clnr
Elevated pH helped in the 2nd Clnr, decent Py rejection
Use of QHS appears to have some benefit to Py rejection, increase in future test

Test No: F10 **Project: Braddick-Grong, 5031** **Operator:** **Date: December 17, 1996**

Purpose: As test F9, with adjustments to reagents to increase recovery in Cu and Zn circuits.

Procedure: As described below.

Feed: 1000 grams of SKM-2 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill.

Conditions:

Stage	Reagent added, g/t								Time, minutes			Measured at start of stage	
	Lime	$ZnSO_4$/NaCN	Na_2SO_3	QHS	A3418	R208	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH	eH
Grind	1500	-	500	-	-	-	-	-	45			8.7	-320
Cu Rougher - 1	640	-	-	-	5	-	-	5		1	3	10.0	-300
Cu Rougher - 2	500	-	-	-	5	10	-	2.5		1	4	10.0	-70
Cu Rougher - 3	280									1	3	10.0	70
Cu Rghr Conc Conditio	260									1		10.3	50
Cu 1st Clnr		-		15		-	-			1	3	10.0	-20
Cu 2nd Clnr	300	-		30	-	-	-	2.5		1	3	11.0	
Cu 3rd Clnr	400		-		-	-	-			1	3	12.0	20
Zinc Flotation on Cu Rougher Tails													
Zn Conditioning	1140	-	-	-	-	-	300	-		4		11.0	30
										3		11.0	
Zn Rougher	385	-	-	-	-	10	-	5		1	6	11.0	-20
	800					10		5		1	3	11.0	
Zn Scavenger	700					10		2.5		1	3	11.0	
Zinc cleaning on rougher concentrate													
Zn 1st Clnr	460	-		5		10	-			1	5	11.0	-30
Zn 2nd Clnr	680	-		10	-	-	-			1	4	11.0	
Zn 3rd Clnr	655			10						1	4	12.0	50

Stage	Cu, Zn Roughers	Cu, Zn Clnrs
Flotation Cell	1000gD1	250gD1
Speed: r.p.m.	1800	1200

Test No: F10 Project: Braddick-Grong, 5031 Operator: Date: December 17, 1996

Metallurgical Balance

Product	Weight		Assays, %, g/t						% Distribution					
	grams	%	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Cl Conc	8.3	0.84	29.7	2.57	29.6	0.076	8.1	155	25.0	1.1	0.7	1.2	17.8	5.8
Cu 3rd Clnr Tail	17.8	1.80	20.4	4.15	31.7	0.091			36.9	3.6	1.7	3.2		
Cu 2nd Clnr Tail	26.9	2.71	5.23	4.83	35.6	0.17			14.3	6.4	2.9	9.0		
Cu 1st Clnr Tail	90.4	9.12	1.71	3.94	35.1	0.12			15.7	17.6	9.5	21.3		
Zn 3rd Clnr Conc	13.6	1.37	0.81	36.5	16.1	0.12			1.1	24.5	0.7	3.2		
Zn 3rd Clnr Tail	15.9	1.60	0.68	12.2	26.8	0.11			1.1	9.6	1.3	3.4		
Zn 2nd Clnr Tail	34.7	3.50	0.29	1.26	31.2	0.077			1.0	2.2	3.3	5.3		
Zn 1st Clnr Tail	84.6	8.53	0.18	1.35	30.5	0.061			1.5	5.6	7.8	10.2		
Zn Ro Scav Conc	32.5	3.28	0.14	3.61	29.3	0.06			0.5	5.8	2.9	3.8		
Zn Rougher tail	666.8	67.25	0.04	0.72	34.6	0.030			2.8	23.7	69.3	39.3		
Head (Calc)	991.5	100.0	0.99	2.04	33.6	0.051			100.0	100.0	100.0	100.0		
Head (direct)			**0.97**	**1.90**	**33.7**	**0.050**	**0.38**	**22.5**					100.0	100.0

Calculated Grades and Recoveries

Cu 3rd Cl Conc	0.84	29.7	2.57	29.6	0.08	8.10	155	25.0	1.1	0.7	1.2	17.8	5.8
Cu 2nd Clnr Conc	2.63	23.4	3.65	31.0	0.09			61.9	4.7	2.4	4.4		
Cu 1st Clnr Conc	5.35	14.2	4.25	33.4	0.13			76.2	11.1	5.3	13.4		
Cu Rougher Conc	14.46	6.31	4.05	34.5	0.12			91.9	28.7	14.8	34.8		
Zn 3rd Clnr Conc	1.4	0.8	36.5	16.1	0.1			1.1	24.5	0.7	3.2		
Zn 2nd Clnr Conc	2.98	0.74	23.4	21.9	0.11			2.2	34.1	1.9	6.7		
Zn 1st Clnr Conc	6.48	0.50	11.4	26.9	0.09			3.2	36.2	5.2	11.9		
Zn Rougher Conc	15.01	0.32	5.70	29.0	0.08			4.8	41.8	12.9	22.1		
Zn Rougher Feed	85.5	0.09	1.70	33.4	0.039			8.1	71.3	85.2	65.2		

Observations

Test No: F10
Project: Braddick-Grong, 5031
Operator:
Date: December 17, 1996

Objective:

Third test on the SKM-2 sample
Based on visuals of test F 9

Observations:

Copper Flotation:

Increased Na2SO3 reduced Py flotation from F 9. Not as active
Use of QHS in opposition of Na2SO3 produced more rejectiion of Py in the cleaner circuit. Elevation of pH in the3rd Cl dropped weight and slowed the overall kinetics.
Have to address the Py problem of being active out of the mill.

Zinc Flotation:

Increased CuSO4 in the Zn Ro to speed kinetics. The froth appears more mineralized than in test F9. Probably the carry over of the increased collector in the Cu Ro circuit. QHS in conjunction with elevating the pH aided with rejection of the travelling Py. Overall recovery gained nothing in the Zn Ro due to active Zn floating in the Cu Ro circuit.

Test No: F11 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: December 30, 1996**

Purpose: Repeat of test F7, for the production of a pyrite concentrate

Procedure: As described below.

Feed: 1000 grams of SKM-2 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill.

Conditions:

Stage	Reagent added, g/t						Time, minutes			Measured at start of stage
	Lime	Na_2SO_3	A3418	R208	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH
Grind	1500	750	-	-	-	-	45			
Cu Rougher - 1	590	-	5	-	-	50		1	4	10.0
Cu Rougher - 2	710	-	5	10	-			1	6	10.0
Cu Rghr Conc Condition	180	200						4		10.0
Cu 1st Clnr	300	-	5	-	-			1	4	11.0
Cu 2nd Clnr	230	-	-	-	-			1	4	11.0
Zinc Flotation on Cu Rougher Tails										
Zn Conditioning	1030	-	-	-	200	-		4		11.0
Zn Rougher	440	-	-	10	-	50		1	6	11.0
Zn Scavenger	530	-	-	10	-			1	3	11.0
Zinc cleaning on rougher concentrate										
Zn 1st Clnr	100	100	-	10	-	10		1	7	11.0
Zn 2nd Clnr	340	200	-	-	-	10		1	5	11.5
Zn 3rd Clnr	280	300	-	-	-	10		1	4	11.5
	H_2SO_4			**Flex-31**		**MIBC**				
Pyrite Conc - 1	1410			50				1	2	8.5
Pyrite Conc - 2	-			50				1	4	8.5
Pyrite Conc - 3	-			100				1	6	8.4
Combine the rougher concentrates for cleaning										
Pyrite 1st Cleaner								1	6	8.5

Stage	Cu, Zn Roughers	Cu & Pyrite 1st Clnr	Cu, Zn Clnrs
Flotation Cell	1000gD1	500gD1	250gD1
Speed: r.p.m.	1800	1500	1200

Test No: F11 **Project: Braddick-Grong, 5031** **Operator: F.V.** **Date: December 30, 1996**

Metallurgical Balance

Product	Weight		Assays, %, g/t					% Distribution				
	grams	%	Cu	Zn	Fe	Au	Ag	Cu	Zn	Fe	Au	Ag
Cu 2nd Clnr Conc	61	6.14	13.0	5.13	36.4	4.25	85.7	80.0	15.5	6.8	68.6	23.4
Cu 2nd Clnr Tail	36.1	3.63	0.43	2.62	43.5			1.6	4.7	4.8		
Cu 1st Clnr tail	70.2	7.06	0.54	3.14	32.9			3.8	10.9	7.0		
Zn 3rd Clnr Conc	5.1	0.51	11.8	31.9	17.2	2.32	57.2	6.1	8.0	0.3	3.1	1.3
Zn 3rd Clnr Tail	0.4	0.04	4.43	12.0	26.3			0.2	0.2	0.0		
Zn 2nd Clnr Tail	2.4	0.24	1.11	3.27	34.9			0.3	0.4	0.3		
Zn 1st Clnr Tail	11.4	1.15	0.26	0.57	31.6			0.3	0.3	1.1		
Zn Ro Scav Conc	5.3	0.53	2.04	13.8	26.8			1.1	3.6	0.4		
Pyrite 1st Clnr Conc	474.1	47.7	0.085	2.31	44.5	0.17	9.30	4.1	54.1	64.3	21.3	19.7
Pyrite 1st Clnr Tail	46.3	4.66	0.170	0.29	38.0			0.8	0.7	5.4		
Pyrite Ro Tail	281.9	28.4	0.063	0.12	11.3			1.8	1.7	9.7		
Head (Calc)	994.2	100.0	1.00	2.04	33.0			100.0	100.0	100.0		
Head (direct)			**0.97**	**1.90**	**33.7**	**0.38**	**22.5**					

Calculated Grades and Recoveries

Cu 2nd Clnr Conc	6.1	13.0	5.13	36.4	4.25	85.7	80.0	15.5	6.8	68.6	23.4
Cu 1st Clnr Conc	9.8	8.33	4.20	39.0			81.6	20.1	11.5		
Cu Rougher Conc	16.8	5.06	3.75	36.5			85.4	31.0	18.6		
Zn 3rd Clnr Conc	0.5	11.8	31.9	17.2	2.32	57.2	6.1	8.0	0.3	3.1	1.3
Zn 2nd Clnr Conc	0.6	11.3	30.5	17.9			6.3	8.3	0.3		
Zn 1st Clnr Conc	0.8	8.18	22.2	23.0			6.5	8.7	0.6		
Zn Rougher Conc	1.9	3.50	9.42	28.1			6.8	9.0	1.7		
Zn Scav. Conc	0.5	2.04	13.8	26.8			1.1	3.6	0.4		
Pyrite 1st Clnr Conc	47.7	0.09	2.31	44.5	0.17	9.30	4.1	54.1	64.3	21.3	19.7
Pyrite Rougher Conc	52.3	0.09	2.13	43.9			4.9	54.7	69.6		
Pyrite Rougher Feed	80.7	0.08	1.42	32.5			6.7	56.4	79.3		

Test No: F12 **Project: Braddick-Grong, 5031** **Date: Jan. 10, 1997**

Purpose: Repeat of test F11, for the production of a pyrite concentrate

Procedure: As described below. **Operator: G.C.**

Feed: 1000 grams of SKM-2 Sample, crushed to 10 Mesh

Grind: 45 minutes/1kg at 50% solids in a lab ball mill.

Conditions:

Stage	Reagent added, g/t						Time, minutes			Measured at start of stage
	Lime	Na_2SO_3	A3418	R208	$CuSO_4$	MIBC	Grind	Cond.	Froth	pH
Grind	1500	750					45			8.3
Cu Rougher - 1	600		5			5		1	4	10.0
Cu Rougher - 2	400		5	10				1	6	10.0
Cu Rghr Conc Condition	160	200						4		10.0
Cu 1st Clnr	420		5					1	4	11.0
Cu 2nd Clnr	250							1	4	11.0
Cu 3rd Clnr	320	100						1	2	11.0
Zinc Flotation on Cu Rougher Tails										
Zn Conditioning	1270				600			4		11.0
Zn Rougher	600			10		7.5		1	6	11.5
Zn Scavenger	340			10				1	3	11.5
Zinc cleaning on the combined rougher & scavenger concentrate										
Zn 1st Clnr	190	100		10				1	5	11.5
Zn 2nd Clnr	250	200						1	4	11.5
Zn 3rd Clnr	210	300						1	2.5	11.5
	H_2SO_4			**Flex-31**		**MIBC**				
Pyrite Conc - 1	1200			50		5		1	2	8.5
Pyrite Conc - 2				50		5		1	4	8.5
Pyrite Conc - 3				100		5		1	6	8.4
Pyrite 1st Cleaner								1	6	8.5

Stage	Cu, Zn Roughers	Cu & Pyrite 1st Clnr	Cu, Zn Clnrs
Flotation Cell	1000gD1	500gD1	250gD1
Speed: r.p.m.	1800	1500	1200

Test No: F12 **Project: Braddick-Grong, 5031** **Date: Jan. 10, 1997**

Metallurgical Balance

Product	Weight grams	Weight %	Assays, %, g/t Cu	Zn	Fe	Pb	Au	Ag	% Distribution Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Clnr Conc	17.9	1.86	**27.4**	**4.64**	**29.6**				48.9	4.1	1.6			
Cu 3rd Clnr Tail	7.6	0.79	4.3	1.87	42.0				3.3	0.7	1.0			
Cu 2nd Clnr Tail	32.3	3.35	1.89	2.1	43.5				6.1	3.4	4.3			
Cu 1st Clnr tail	36.3	3.77	2.34	2.6	42.4				8.5	4.7	4.7			
Zn 3rd Clnr Conc	33.2	3.45	5.36	45.9	12.2				17.7	76.0	1.2			
Zn 3rd Clnr Tail	4.2	0.44	5.57	16.2	26.3				2.3	3.4	0.3			
Zn 2nd Clnr Tail	5.0	0.52	3.02	5.86	30.9				1.5	1.5	0.5			
Zn 1st Clnr Tail	12.3	1.28	2.03	3.06	31.1				2.5	1.9	1.2			
Pyrite 1st Clnr Conc	562.1	58.4	0.130	0.11	46.2				7.3	3.1	78.8			
Pyrite 1st Clnr Tail	33.8	3.51	0.190	0.19	26.8				0.6	0.3	2.8			
Pyrite Ro Tail	236.2	24.5	0.055	0.08	5.16				1.3	0.9	3.7			
Head (Calc)	963.0	100.0	1.04	2.08	34.2				100.0	100.0	100.0			
Head (direct)			**0.97**	**1.90**	**33.7**	**0.050**	**0.38**	**22.5**						

Calculated Grades and Recoveries

Product	Weight grams	Weight %	Cu	Zn	Fe	Pb	Au	Ag	Cu	Zn	Fe	Pb	Au	Ag
Cu 3rd Clnr Conc		1.9	27.4	4.64	29.6				48.9	4.1	1.6			
Cu 2nd Clnr Conc		2.6	20.5	3.81	33.3				52.2	4.9	2.6			
Cu 1st Clnr Conc		6.0	10.1	2.86	39.0				58.3	8.2	6.8			
Cu Rougher Conc		9.8	7.11	2.76	40.3				66.7	12.9	11.5			
Zn 3rd Clnr Conc		3.4	5.4	45.9	12.2				17.7	76.0	1.2			
Zn 2nd Clnr Conc		3.9	5.4	42.6	13.8				20.1	79.4	1.6			
Zn 1st Clnr Conc		4.4	5.10	38.2	15.8				21.6	80.8	2.0			
Zn Rougher Conc		5.7	4.41	30.3	19.2				24.1	82.7	3.2			
Pyrite Clnr Conc		58.4	0.13	0.11	46.2				7.3	3.1	78.8			
Pyrite Rougher Conc		61.9	0.13	0.11	45.1	0.037	0.24	12.8	7.9	3.4	81.6	45.8	39.1	35.2
Pyrite Rougher Feed		86.4	0.11	0.10	33.8				9.2	4.3	85.3			

Test to confirm oxidation of the SKM-2 sample

Assumption:

The oxidation of the sulphides will produce sulphates
The sulphates will increase the acidity of the sample when slurried with water.

Samples

Aged Sample: Test Charge (1 kg) of SKM-2, stored in freezer.
The charge was prepared on November 26, by crushing from 3/4" to 10 mesh
Fresh Sample Crush 1 kg of the 3/4" reject sample to minus 10 mesh

	Aged Sample	Fesh Sample
6-Nov-96	Crush to -3/4" Store in Freezer	Crush to -3/4" Store in Freezer
26-Nov-96	Crush to 10 M	
24-Jan-97		Crush to 10 M
24-Jan-97	Measure pH	Measure pH

Procedure:

On each of the two samples:
Mix the identical weights of each sample with identical volumes of water to obtain 65% solids
Measure the pH of the slurry

	Aged Sample	Fresh Sample
Sample Weight	941	941
pH	3.1	3.5

APPENDIX No. 4

Basis of Smelting & Refining Charges for Copper/Zinc Concentrates

NORSULFID AS
Avd. Grong Gruber

Member of
Outokumpu Metals & Resources OY

Postadresse:
N-7894 Limingen
Norway

Telefon:
74 33 52 00

Telefax:
74 33 58 45

Bank: DnB
7460.05.05460

Postgiro:
0809.2.085520

FAX TRANSMISSION

Date: Dec 6th 1995 **Fax no.:** 00 1 416 947 0807

No. of pages (incl.cover): 1

To: Braddick Resources Ltd.

Attention: Derek Bartlett

From: Arve Haugen

<u>If you do not receive all the pages, please telephone immediately</u>

Derek,
Thank you for your fax of Dec 4th 95.

As you problably understand, I am not at liberty to give you the exact n[redacted] asking for. But:

- TC is approx US $ 180, we are negotiating this every year.
- As for what we are getting back in return, is dependent on price, TC, escalating factor penalties (Fe, Cd, MgO) conc. content etc. Since 1990 our return has varied between 67% and 55% of the LME-price value (in NOK), of course dependent on every year's TC. But as for our return, it is safely to say between US $ 270 and 320.
- We have a budget for Zn in conc. of 53%. Production values differ in average from 52-54%.

Although you do not ask, the operating costs in our mill are at the moment US $ 7,15-7,95.

Hope you can make use of this information.

Best regards

Arve

Arve

NORSULFID AS
Avd. Grong Gruber

Member of
Outokumpu Metals & Resources OY

Postadresse:	Telefon:	Telefax:	Bank: DnB	Postgiro:
N-7894 Limingen Norway	74 33 52 00	74 33 58 45	7460.05.0546	0809.2.085520

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- As for what we are getting back in return, is dependent on price, TC, escalating factor penalties (Fe, Cd, MgO) conc. content etc. Since 1990 our return has varied between 67% and 55% of the LME-price value (in NOK), of course dependent on every year's TC. But as for our return, it is safely to say between US $ 270 and 320.
- We have a budget for Zn in conc. of 53%. Production values differ in average from 52-54%.

Although you do not ask, the operating costs in our mill are at the moment US $ 7,15-7,95.

Hope you can make use of this information.

Best regards

Arve

Arve

Geologiske Tjenester a.s.
Hovfaret 8
N-0275 Oslo, Norway
Telephone: 22 50 65 30
Telefax: 22 50 91 30
Bankgiro: 7029.05.15767
Company Reg.: 937 746 571

FAX

Date: 28.10.1996

To: Louis M. Bernard, Fax no. 00.1.905.849-9622

From: Boye Flood

Subject: Your request sent by Frank October 30, 1996

Please find attached the answers to your request as sent us by Outokumpu.

We have been informed that the Borregaard sulphuric acid plant in Sarpsborg around 150 km SSE of Oslo is buying 250,000 tons of pyrite concentrate annually from Finland, the Pyhæsalmi deposit which will be closed in year 2000.

Most probably they will then convert their plant for sulphur as raw material, but this has not yet been decided. They expect to pay around 20% less for 100% S by using sulphur compared to pyrite.

The concentrate they buy holds 50-52% S and 45% Fe. They manage to sell their iron oxide to the cement industry on the continent.

Maximum accepted values for other elements are in %:

Cu	0.1	
Co	0.025	
Zn	0.1	
Ni	0.01	
Pb	0.075	
As	0.06	
Hg	0.25	ppm
Cd	3	ppm
water		6%

The concentrate is transported by road or rail from the mine to Botniska Viken, and from there by ship to Sarpsborg.

Regards

Boye

Outokumpu
HARJAVALTA METALS
Espoo - HW/MR

TELEFAX

November 1, 1996

total pages: 2

TO: Geologiske Tjenester a.s 47-22-509130

Mr Boye Flood

FROM: Heikki Welling
Outokumpu Harjavalta Metals Oy, Espoo
Tel: +358-9-421 3158 Fax: +358-9-421 2520

RE: Copper concentrates, your fax 31.10.1996

Many thanks for your fax yesterday.

Below please find our answers:

1 Payable copper content
= the assayed copper content less 3.5 % or less 1.0 percentage unit whichever deduction is greater.

2 Payable gold content
= the assayed gold content less 5 % or less 1.0 g/dmt whichever deduction is greater.

Payable silver content
= the assayed silver content less 5 % or less 30 g/dmt whichever deduction is greater.

3 It's difficult to give any absolute minimum copper grade. The lowest we ever have used was 15 % Cu.

4 Max. Zn content roughly 5 %. Zn penalty trigger level 2 %.

Moisture content shall not exceed the International Maritime Organization limit (Transportable Moisture Limit, TML). Otherwise, max. moisture about 10 %.

5 TC, RC Cu, RC Au and RC Ag are negotiated based on market circumstances.

In long-term agreements, the terms applicable in 1997 are estimated to be as follows:

TC	100-115 \$/dmt
RC Cu	10-11.5 c/lb
RC Au	6 \$/tr.oz.
RC Ag	0.4 \$/tr.oz.

Price participation for copper:
10 % of the portion of copper price in excess of 90 c/lb.

At the moment, in the spot market, TC is about 130 \$/dmt and RC Cu approx. 13 c/lb.

6 There is a long list of penalizable elements. Could you please let us know the expected assay of the concentrates so we could comment it.

Best regards

Heikki Welling

Heikki Welling

APPENDIX No.5

Quote on Used 1000 tonne/day Flotation Plants

M.C. LA BARR

6262 N. SWAN RD.-SUITE 170 / TUCSON, ARIZONA 85718 / TELEPHONE (520) 577-2414 / FAX (520) 577-0952 / e-mail: labarr@worldnet.att.net

DATE: October 31, 1996

TO: Mr. Luke Bernard
Ontario
Canada

FAX: 905-849-9622

PHONE: 905-849-9971

Dear

We can offer two plants that are the closest to the capacity requirements for your project in Norway. Both 60Hz plants were built in the 1970's, operated 4 years and shut down in excellent condition.

Our office suffered a hard disc failure and a backup glitch on Monday. The hard disc is at a lab and we should have the support data available by early next week. We do include the prices "As is-where is" for your current reference. A complete set of engineering drawings and manuals are included with both plants.

With best regards,

M.C. LaBarr

Encl:

850 TPD Copper Zinc. Price-$950,000
1000TPD Copper Concentrator. Price-$1,250,000

M.C. LA BARR

6262 N. SWAN RD.-SUITE 170 / TUCSON, ARIZONA 85718 / TELEPHONE (520) 577-2414 / FAX (520) 577-0952 / e-mail: labarr@worldnet.att.net

DATE: November 22, 1996

TO: Mr. Louise Bemard
Senior Mining Consultant
331 Maplehurst Avenue
Oakville, Ontario L6L 4Y3
Canada

FAX: 905-849-9622

PHONE: 905-849-9971

Dear Mr. Bemard,

We apologize in the delay of getting the following information to you.

Please give us a call if you have questions on the following information.

With best regards,

M.C. LaBarr

MCL/wrh

Encl: 750 TPD Copper Zinc, d750tpd.doc, Price-$950,000
1000TPD Copper Concentrator, d1000 tpd.doc, Price-$1,250,000

M.C. LA BARR

6262 N. SWAN RD.-SUITE 170 / TUCSON, ARIZONA 85718 / TELEPHONE (520) 577-2414 / FAX (520) 577-0952 / e-mail: labarr@worldnet.att.net

1000 TPD COPPER CONCENTRATOR

Feeder, Hydrostroke, 42" x 14', NICO 550
Crusher, Jaw, 24" x 36", Universal WRB
Stacker, Belt, 36" x 100', 50 TPH, 10HP
Feeder, Apron, 3' x 8', NICO, Cast Pans, Cat Trac
Hopper, Steel, Feeder, 7'-5" x 8'-7" x 5' high, with 3/4" AR liner plate and 20# rail wear bars
Conveyor, Belt, 24" - 30" x 1500' (9 total)
Grizzley, Vibrating, 36" x 12', Telsmith, HD, 4 bars, 15HP
Screen, Vibrating, 5' x 12', DD, Hewitt Robins, 20HP
Electromagnet, 19" x 22 1/2" x 9" high, Silicon 707 with rectifier
Crusher, Cone, 5 1/2', Nordberg, SH, 200HP
Transfer Tower, Steel
Bin, Steel, 1300T, Fine Ore Bin, 35'D x 32'H, Cone Top
Feeder, Pioche, 36" x 20', with 3/4" AR liners (2)
Mill, Rod, 6'-6" x 12'-1", with 250HP/380/3/60/440V
Trommel, Discharge, 4'-4 1/2" OD x 5'-5" long x 3/16" steel with 1/2" round holes
Sampler, Belt, 24", Denver S2
Jig, Mineral, 24" x 36", Denver Duplex with #3 Dausett valves and S/S shafts
Pump, Centrifugal, Horizontal & Vertical, 1/2-8", Misc. HP (9)
Cyclone, Wet, 20", Krebs D20-865
Sump, Steel, 5' x 4' x 4' x 1/4" PL
Mill, Ball, 11'D x 4' x 6' x 10' Tricone, overflow, Double scoop feed, Fawick Air Flex clutch, gear reducer, 700HP
Jig, Mineral, 24" x 30", Denver Duplex S2 with #3 Dausett valves and S/S shafts
Sampler, Belt, Denver S2-18" Cut, Type B, 1/4HP gear motor with magnetic brake (2)
Sampler, Belt, Denver S2-18" Cut, Type B, 1/2HP gear motor with magnetic brake (2)
Belt Scale, Howe Richardson, Continuous Weigh for 24" belt
Dust Collector, Wet, Ducon 58 MUL Model 1, 10,000 CFM, 50HP
Dust Collector, Wet, Fisher-Kosterman WM750, 1/4" wear plate, 260MH American Std. blower, 100 HP
Classifier, Spiral, Denver, 12" x 8'-6", 1HP
Flotation Cells, Denver DR-24, 2 banks, 8 cells, 40CF cells
Flotation Cells, Denver DR-18, 2 cell unit, 25CF per cell, 7.5HP
Thickener, 20' x 8', Bridge Type, 2HP with reducer
Thickener, 75' x 12', Bridge Type, 5HP with reducer
Filter, Drum, 8' x 4', Dorr Oliver 8F8C, 304SS, with nylon backer panels and polypropylene filter media
Pump, Vacuum, Nash Hy-tor 703, 40HP
Pump, Filtrate, 1", Worthington 20NF74, 3HP
Furnace, Heating, 2.5MBTU, Tjemlund HH410, Model #280, Gas fired
Tank, Steel, 2' x 2', reagent steady head tanks (5)
Blower, 1550 CFM, Spencer Turbo Compressor 1520-1AD, 20HP
Building, Steel, 50' x 60' x 30' high, with bridge crane, 10T (Crushing)
Building, Steel, 50' x 110' x 30', with (2) monorail systems with S10 beam x 214', S8 beam x 80' (Concentrator)
Building, Steel, 15' x 50' x 14' (Electrical)
Electric Substation, 500KVA, 2400V/480 each with (3) 100 amp cutouts (3)
Electric Switchgear, LV, 1 Westinghouse DR50, 1200Amp ACB, (3) Westinghouse DR25, 500 Amp ACB
Electrical System for Mill with various control panels, tranformers, distribution panels, starters, CB's and lighting
Heater, Gas/Oil, 4.2MBTU/Hr (Crusher Building)
Heater, Gas, 1 Rezner XB300, 300,000BTU/Hr (2) XB250, 250,000BTU/Hr, (3) XB400, 400,000 BTU/Hr
Parts, 1 lot, Mill Process Equipment Parts
Parts, 1 lot, Process, Piping, Fittings and Valves

d1000tpd.doc, 11/22/96

750 TPD PB/ZN FLOTATION CONCENTRATOR

CONDITIONERS, 6' x 7', Galigher, 18" Prop, 7½HP, (2) (ZN Conditioners)
PUMP, HORIZONTAL, 3" x 4" Galigher Vacseal 3VRG200, 15HP (ZN Pumps)
FLOTATION MACHINE, 60CF, 1 Bank of 8, Galigher 20HP ea. (ZN Rougher Cells)
FLOTATION MACHINE, Wemco 300CF, 30HP (3)
PUMP, VERTICAL, Galigher 2 ½", 10HP (2)
PUMP, HORIZONTAL, Galigher 3 x 4, 30HP
FLOTATION MACHINE, 60CF, 2 Banks of 2, Galigher, 20HP ea. (ZN 1st Cleaners)
PUMP, VERTICAL, 2½" x 48" Galigher Vacseal (ZN 1st Cleaner Sump Pump)
FLOTATION MACHINES, 60CF, 2 Banks of 2, Galigher, 20HP ea. (ZN 2nd Cleaners)
PUMP, VERTICAL, 2½" x 48", Galigher Vacseal, (ZN 2nd Cleaner Sump Pump)
THICKENER, 36' Dia. x 10' Deep, Eimco Bridge Type w/Rake Positioners, 2HP (ZN Cons)
PUMP, DIAPHRAGM, Warren-Rupp Sandpiper SA-2, Neoprene/SS (ZN Concentrates)
FILTER, DRUM, VACUUM, 10' x 10', Eimco, 3HP, (ZN Filter)
PUMP, VACUUM, Nash CL2003, 125HP (ZN Filter Vacuum Pump)
PUMP, FILTRATE, Eimco Krogh, 5HP (ZN Filtrate Pump)
BLOWER, Roots 59AF (ZN Filter Blower)
SAMPLER, Galigher CSA-900 (ZN Concentrate)
PUMP, HORIZONTAL, 3" x 4", Galigher Vacseal 3VRG200
THICKENER, 75', Eimco Bridge Type w/Lifter (Tailings)
PUMPS, HORIZONTAL, 6' x 6" Galigher Vacseal (Tailings Water Recycle)
PUMP, DIAPHRAGM, Warren Rupp Sandpiper SA-4 (Tailings Underflow)
PUMP, HORIZONTAL, Galigher Vacseal, (Tailings Overflow)
SAMPLER, Galigher (ZN Conditioner Sampler)
SAMPLER, Galigher (ZN Thickener Sampler)
PUMP, VERTICAL, 4" x 48, Galigher, (Sump Pump) (Various)
PUMP, VERTICAL, 2 ½" x 48, Galigher (Sump Pump) (Various)
HOIST, TROLLEY, 10T (Various)
HOIST, TROLLEY, 5T, (Various)
HOIST, TROLLEY, 2T, (Various)
TANK, REDWOOD, 34' x 20', 125,000 Gallons (Water Fire Line)
TANK, REDWOOD, 20' x 14', 3000 Gallons (Mill Supply)
TANK, REDWOOD, 6' x 6', 1000G. (Steady Head Tank)
AIR COMPRESSOR, 50HP (Mill Air)
BLOWER, Suterbilt 3100 Series, Silencer, (Mill Flotation Air)
TANK, STEEL, 6'D. x 8'H. w/Agitators, (Lime Tank) (2)
TANK, STEEL, 10' x 10', w/Agitators, Pumps & Feeders (2) ($ZnSO_4$ System)
TANK, STEEL, 10' x 10', w/Agitators, Fans, Pumps & Feeders (2) (NaCN System)
TANK, STEEL, 54" x 6', w/Agitators & Pumps. (R-343 Zanthate System)
TANK, STEEL, 8' x 8', w/Agitator, Feeder & (3) Pumps ($CuSO_4$ System)
PUMP, VERTICAL, 3" x 4", Galigher Vacseal (Reagent Sump)
SILO, STEEL, 50T (Lime Silo)
ELECTRICAL MCC, Nelson SBI Selectric, Westinghouse Breakers
ELECTRICAL SUBSTATIONS, 1500KVA, 4160V Delta 480/277, Westinghouse Feeder Breakers
ELECTRICAL SWITCHGEAR, GTE Sylvania, Zinsco

THIS LIST IS REPRESENTATIVE BUT NOT COMPLETE.
THE COMPLETE PLANT IS OFFERED FOR $950,000 U.S. INCLUDING DRAWINGS AND PARTS.

M.C. LA BARR

6262 N. SWAN RD.-SUITE 170 / TUCSON, ARIZONA 85718 / TELEPHONE (520) 577-2414 / FAX (520) 577-0952 / e-mail: labarr@worldnet.att.net

750 TPD PB/ZN FLOTATION CONCENTRATOR

BIN, STEEL, 30T, A/R Lined (Coarse Ore Dumping Hopper)
FEEDER, BELT, 42" x 20', Reversible, 5HP (Coarse Ore Feeder)
CONVEYOR, BELT, 42" x 12' (Coarse Ore Conveyor)
CONVEYOR, BELT, 42" x 100' (Waste Conveyor) w/ Magnetic Pulley, Eriez, 42" x 44"
BIN, STEEL, 500T, A/R Lined (Coarse Ore Storage)
HOPPER, STEEL, A/R Lined (Coarse Ore Hopper)
FEEDER, APRON, 42" x 20', NICO FD-4275, 10HP (Coarse Ore Feeder)
HOIST, MONORAIL, 10T, Manual Travel, Pendant Control
GRIZZLEY, VIBRATING, 42" x 5', Symons K42-97, 10HP, 3" Spacing
CRUSHER, JAW, 25" x 40", Telsmith, Roller Bearing, Hydraulic Adjust, Automatic Lube, 100HP
FEEDER, BELT, 30" x 163', 46' Lift, 20HP, 150FPM, w/7'x10' Gallery Covering
METAL DETECTOR, Tectron
SCREEN, VIBRATING, 5' x 15' DD, TyRock F1004X, 1" Top, 3/8" to ½" Bottom Screen, 15HP
BELT MAGNET, Eriez 16-SP-610, Suspended, w/Controller and Rectifier
CRUSHER, CONE, 5 ½ SH, Nordberg HD, Hydraulic Clamping and Adjustment, Automatic Lube- 200HP
HOIST, MONORAIL, 10T, Manual Travel, Pendant Control
CONVEYOR, BELT, TRIPPER, 24" x 155' x 25' Rise, 20HP,
SCALE, BELT, Autoweigh 40024 w/Totalizer
TRIPPER, BELT, 24" x 24" x 24" w/2HP Electric Tugger
BINS, STEEL, 700T, 26'D x 30'H, Concrete Base, (Fine Ore Bins) (3)
SCRUBBERS, Centrifugal Wash, Ducon Size 72 UW-4, w/66" Fan & 2x2 Galigher Pump (2)
SCRUBBERS, Centrifugal Wash, Ducon 60UW4 (Various)
FEEDER, BELT, 30" x 18'-4", 2.5HP, Variable (Fine Ore Feeders) (6)
CONVEYOR, BELT, 24" x 155' x 15' Lift, 5HP, (Fine Ore Collecting Conveyor)
CONVEYOR, BELT, 24" x 20' x 6" Lift, 2HP (Fine Ore Feed Conveyor)
BELT SCALE, 24", Autoweigh E400
MILL, BALL, 9'-6" x 9', Grate Discharge, 500HP w/Reducer, Lubestay, Jacking, Trommel
PUMP, HORIZONTAL, 4" x 6", Galigher Vacseal (Ball Mill Discharge)
PUMP, HORIZONTAL, 6" x 6", Galigher Vacseal (Ball Mill Discharge)
PUMP, HORIZONTAL, 3" x 4", Galigher Vacseal, (Flotation Feed)
CYCLONE, WET, Krebs D20B
SAMPLER, Galigher CSA 900, ¼HP
FLOTATION MACHINE, 60CF, 1 Bank of 8, Galigher Agitators, 20HP ea. (PB Roughers)
PUMP, VERTICAL, 2 ½" x 48", Galigher Sump Pumps, 7.5HP, (Various)
FLOTATION MACHINE, 60CF, 2 Banks of 2, Galigher Agitators, 20HP ea. (PB 1st & 2nd Cleaners)
THICKENER SAMPLER, Galigher (PB Concentrate Thickeners) (2)
THICKENER, 36' Dia. x 10' Deep, Eimco Type B, Bridge Type w/Rake Indicator (PB Thickener)
PUMP, Diaphragm, 2.5", Warren Rupp Sandpiper SA-2, Neoprene/SS Seats, (PB Thickener U/F)
FILTER, DRUM, VACUUM, 10' x 10', Eimco, 2HP, 3HP Agitator w/Wire Wrapping Machine (PB Filter)
FILTER PUMP, VACUUM, Nash CL2003, 125HP
FILTER AGITATOR, EIMCO 3HP
FILTRATE PUMP, Eimco Krogh, 5HP, w/Filtrate Receiver
FILTRATE BLOWER, Roots Model 75, 5HP

APPENDIX No. 6

Cost of New & Used Buildings/Trailers

TEL. (403) 245-3371

FAX (403) 229-1980
INTERNET: http://www.sprung.com

Sprung Instant Structures Ltd.

1001 - 10TH AVENUE S.W. CALGARY, ALBERTA T2R 0B7 CANADA

October 25, 1996

Louis Bernard
331 Maplehurst Avenue
Oakville, ON
L6L 4Y3

Dear Mr. Bernard,

We are pleased to submit the following quotation for a custom structure to be located in Norway.

STRUCTURE SIZE: *Approximately 130 feet wide by 165 feet long structure transitioning into a custom 130 feet wide by 120 feet long structure on 10 foot leg extensions.*

FABRIC: *Regular white or tan opaque fabric.*

PURCHASE PRICE: *Structure, including the following accessories:*

4 - Bays of Cable Bracing
1 - Custom Engineered Flat End c/w Insulation and center braced bay
1 - Engineered Flat Ends c/w Insulation and center braced bay
6 - Single Personnel Doors c/w Hood
2 - Full Height Interior Fabric Partitions
3 - 14' x 14' Insulated Steel Roll-Up Doors placed in center panels
3 - 10' x 14' Insulated Steel Roll-Up Door placed in center panels
1 - Transition wall
- Insulation to the Peak c/w Interior Fabric Liner
- Opaque Fabric - No Skylight

NOTE: ***The portion of the structure mounted on 10' leg extensions has the beams placed 10' on center.***

Total Purchase Price,
C & F nearest seaport Norway *US $1,657,527.00*

TERMS: *Confirmed irrevocable Letter of Credit with drawings at sight against shipping documents.*

TEL. (403) 245-3371

FAX (403) 229-1980
INTERNET: http://www.sprung.com

Sprung Instant Structures Ltd.

1001 - 10TH AVENUE S.W. CALGARY, ALBERTA T2R 0B7 CANADA

October 25, 1996

Louis Bernard
331 Maplehurst Avenue
Oakville, ON
L6L 4Y3

Dear Mr. Bernard,

We are pleased to submit the following quotation for a structure to be located in Norway.

STRUCTURE SIZE:	*Approximately 130 feet wide by 285 feet long.*
FABRIC:	*Regular white or tan opaque fabric, complete with translucent skylight.*
PURCHASE PRICE:	*Structure, including the following accessories:*

6 - 14' x 14' Insulated, Steel Roll-Up Doors placed in center bays
2 - Bays of Cable Bracing
2 - Engineered Flat Ends c/w Insulation and center braced bays
6 - Single Personnel Door(s) c/w Hood
2 - Full Height Interior Fabric Partitions
- Insulation to the Peak c/w White Interior Liner

Total Purchase Price,
C & F Seaport Norway — *US $1,104,724.00*

TERMS:	*Confirmed irrevocable Letter of Credit with drawings at sight against shipping documents.*
DELIVERY:	*Normally from inventory.*
DATE REQUIRED:	*To be determined.*
ERECTION:	*We will supply one Technical Consultant, equipped with all hand tools, free of charge, to supervise the erection of this structure by your work force. It will be your responsibility to administer your worker safety procedures. It will also be your responsibility to supply the following:*

TEL. (403) 245-3371

FAX (403) 229-1980
INTERNET: http://www.sprung.com

Sprung Instant Structures Ltd.

1001 - 10TH AVENUE S.W. CALGARY, ALBERTA T2R 0B7 CANADA

October 29, 1996

Louis Bernard
331 Maplehurst Avenue
Oakville, ON
L6L 4Y3

Dear Mr. Bernard,

We are pleased to submit the following quotation for a structure to be located in Norway.

STRUCTURE SIZE: Approximately 50 feet wide by 70 feet long.

FABRIC: Regular white or tan opaque fabric.

PURCHASE PRICE: Structure, including the following accessories:

1 - 10' wide x 10' high Insulated Metal Roll-Up Door placed in End Panel
1 - Single Personnel Door c/w Hood
- Insulation to the Peak c/w White Interior Liner
- Opaque Fabric-No Skylight

Total Purchase Price,
C & F nearest seaport Norway US$73,060.00

TERMS: Confirmed irrevocable Letter of Credit with drawings at sight against shipping documents.

DELIVERY: Normally from inventory.

DATE REQUIRED: To be determined.

ERECTION: We will supply one Technical Consultant, equipped with all hand tools, free of charge, to supervise the erection of this structure by your work force. It will be your responsibility to administer your worker safety procedures. It will also be your responsibility to supply the following:

a) Scaffolding or manlifts.
b) Electrical power to site.
c) 6 unskilled workmen for approximately 6, 8 hour working days.

DELIVERY: Normally from inventory.

DATE REQUIRED: To be determined.

ERECTION: We will supply one Technical Consultant, equipped with all hand tools, free of charge, to supervise the erection of this structure by your work force. It will be your responsibility to administer your worker safety procedures. It will also be your responsibility to supply the following:

a) Manlifts and scissorlifts.
b) Electrical power to site.
c) 14 unskilled workmen for approximately 60, 8 hour working days.

HAND TOOLS: You will be charged in advance for a set of specialized hand tools required for erection at an extra cost of $12,000.00. These will be shipped with the structure. If you choose to return them following erection you will be credited accordingly.

CRANE: We request that you supply two large cranes, with operators, to assist in raising the free span aluminum beams. Both will be required at the beginning of the erection sequence. They will be needed for approximately 12 and 36 hours respectively.

TECHNICAL CONSULTANT: We would ask that Louis Bernard, supply meals, accommodation and ground transportation for our technical consultant while on location, as well as the return airline ticket from Calgary, Alberta, Canada. If circumstances dictate overtime charges may occur with your approval.

ANCHORAGE: Concrete footings. Specifications will be provided where required.

PERMITS, LICENSES AND TAXES: It will be your responsibility to obtain all permits and licenses and pay all applicable taxes. Standard pre-engineered drawings are available upon request.

This quotation is valid for 60 days.

*Thank you for the opportunity to submit this quotation. To demonstrate our confidence in the integrity of the Sprung Instant Structure we enclose for your review, Guarantee Certificate No:**B-1909**. We look forward to being of service to you.*

Yours very truly,

SPRUNG INSTANT STRUCTURES LTD.

Garry Blashyn
Sales Manager
GAB/ng

GE Capital Modular Space
Bob Kucherawy
P.O. Box 89
13932 Woodbine Ave.
Gormley, Ontario L0H 1G0
905-713-2826, Fax 905-841-3640

Friday, October 25, 1996

FAXED TO 905 849-9622

Lou Bennard
331 Maplehurst Ave
Oakville, Ontario L6L 4Y3

Dear Mr. Bennard :

Please find enclosed budgetary pricing for office trailers and a washcar as per your request.

We currently have 4 10x40 Office Trailers the have no wheels available and a 10x52 washcar as per attached layout.

Outright Purchase price per 10x40 as is	$3,000.00 -$5,000.00
Delivery to Toronto	$ 450.00 each

Outright Purchase Price for a 10x52 Washcar see attached layout $21,235.00.
Delivery to Toronto (This unit is currently in Our Sudbury Yard) $ 850.00.

Offloading by crane and placing on the ship would be the responsibility of the client.

As per our telephone conversation Modifications are as follows if you require us to do them;

New Floor per 10x40	$1,600.00 approx each
Re-panell Walls	$2,065.00 approx. each
New Ceiling	$2,000.00 approx. each
New Doors	$ 800.00 each door installed

If any other modifications are required they would be extra as needed.

All taxes are extra.

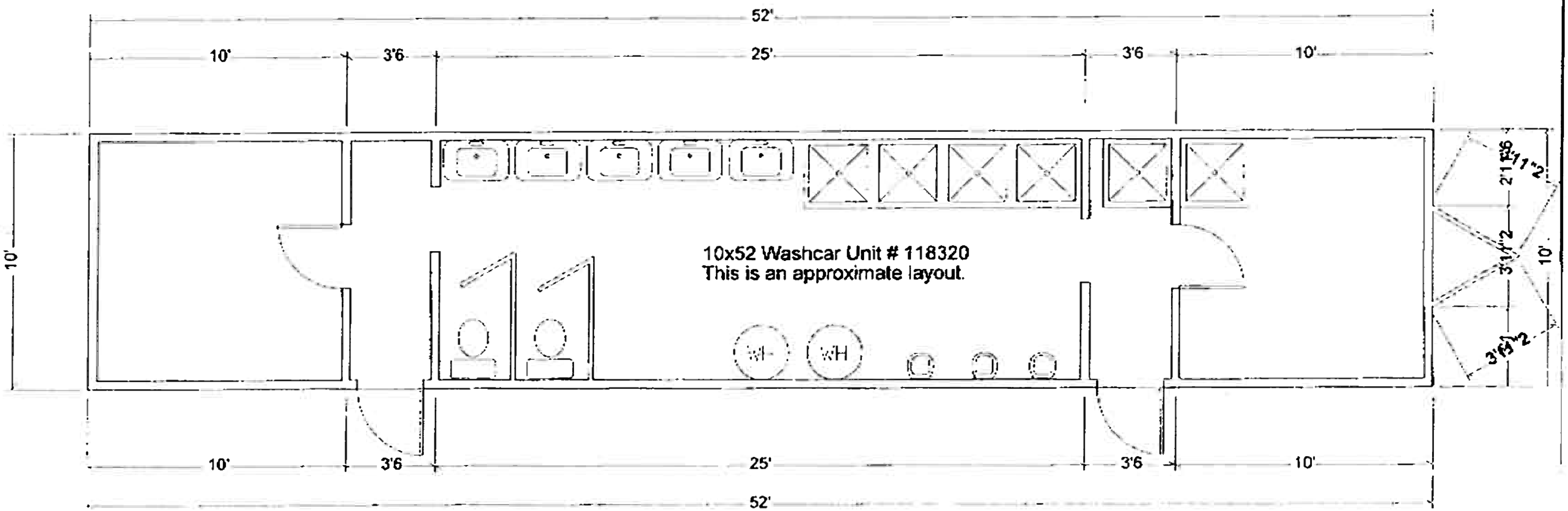
All equipment is subject to availability.

I am sure this information will help, I will not be available until Wednesday of next week. If you have any questions please call my Mobile at 905-308-4984.

We look forward to being of service to you.

Sincerely,

Bob Kucherawy
Senior Sales Representative


52'
10'
3'6
25'
3'6
10'
10'
10x52 Washcar Unit # 118320
This is an approximate layout.
WH
WH
10'
3'6
25'
3'6
10'
52'

APPENDIX No.7

Supplier Quotes for Cost of New/Used Equipment

GRONG COPPERZINC PROJECT

MINE CAPITAL / EQUIPMENT COSTS

ITEM	No.	SIZE	Motor HP / KW	Cdn.$ COST	
ORE BIN	1	100 tonne, 3m.dia X5m	/0		(New)
GRIZZLY FEEDER	1	1X4m.	15/ 11	000	(used)
JAW CRUSHER	1	24X36	100/75	97,000	(used)
COMPRESSOR	1	1000 cfm	250 /185	$39, 000	( )
SCOOP TRAM	2 of ea	2(5)yd. 5yd or 6yd.	/0	$196,000	EACH (used)
u/g TRUCK	2	36 tonne	/0	$380, 000	EACH (used)
DRILL(mobile)	2	Longhole/drifter, in. wide	/0	$115, 000	EACH
DRILL JUMBO	1	2-Boom (elec./hydraulic)	/	$296 000	(used)
MAIN VENT FAN	2	3-4000cfm	2x100/150	000	(used)
U/G VENT FANS	6	200 cfm	6x10/45	000	(used)
U/G VENT TUBE	10m lengths	600&800mmFlex	/0	00ea.	( )
VENT TUBE Couplings		(as above)		00ea.	(new)
Dewatering Pump	1	Gorman Rupp S-4J1	60/45	000	( )
U/G Maint. Equipt.	?	(various tools)		000	(est.)
Survey Equipment	1	total station		000	(est.)
Cap Lamp Charger	1	20 Lamps , 6volts	/0.2	000	(new)
Cap Lamps	20	normal	/0	000	(new)
PIPE	n/a	500 m. of 4in. steel, 3in.&2in. plastic		000	(new)
ANFO Hole Loader	1	Mobile using Scoop Tram. 500 lb. capacity		$125, 000	(new)
Front end Loader(surface)	1	2Yd Bucket		$79, 000	( )
u/g Transformer	2	(to serve elct/hyd drills)		$18, 000	EACH ( )
TOTAL			**Kw**	**$,000**	

FEED Rate = 1030 tonnes/day ore, 3.5% Zn, 1.0% Cu , 63% Pyrite

PRODUCTS : Cu conc. =40 tpd, Zn conc. =60 tpd Pyrite conc. = 650 tpd

NOTE: **All Electrics to be 50 HZ for European Use.**

* 3/8" (9.5mm) steel fabricated to tanks = $ 00/tonne steel

** Concrete estimated @ $ 00/m3

MILL CAPITAL / EQUIPMENT COSTS

ITEM	No.	SIZE	Motor H.P / KW	CdnS COST
Coarse Ore Bin	1	2000 tonne(714 cu.m.)	/0	000 ( )
CONE CRUSHER	1	Allis 4000 MF.	150/113	000 ( )
SCREEN	1	3X6 m	15/11	000 ( )
CONVEYORS	5	0.6m W X 110 m. L	5 x 5/19	000 ( )
Fine Ore Bin	1	2000 tonne(714 cu.m)	/0	000 ( )
Weightometer	1	10-20 tph.cap.	/0	000 (new)
BALL MILL	1	3X4.25 m.	1000/750	196,000 ( )
CYCLONEs	2	250mm. Dia.	/0	000 ( )
Process Pumps	9	1.5x 1.25 SRL	8x 2h.p./12	000 ( )
	3	1.5x1.25 SRL	3x3h.p./6.75	000 ( )
	2	5 x 5 SRL.	2x40hp/60	000 ( )
	4	5 x 5 SRL	2x30hp/45	000 ( )
	3	Vertical Sump-2inch	5/3.75	000 ( )
CONDITIONER	3	2.5m x 2.5m.	/0	* 000 ( )
AGITATOR	1	small	3/2.25	000 ( )
FLOT. CELLS	56/14	Denver DR100/18sp		000 ( )
Vacuum Pump	1		15/11.5	000 ( )
Air Blower	2	000 scfm	5/3.75	000 ( )
Conc. Thickeners	3	7/ 7/30m.diam.		000 ( )
Filters, Drum	3	m x m	/	000 ( )
Conc. Hold. Tanks	2	1 day-(4m.dia.x4m. high)	0	* 000 ( )
Conc.Tank Agitator	2	small	5/3.75	000 ( )
Conc. Conveyor	3	0.6m.W x20m. L.	5/3.75	000 ( )
Conc. Storage Pad	2	20x20m.X 0.15m(covered) + 2m.wall	/0	** 000 ( )
Samplers	10		10 X 1/10	000 ( )
Reagent Feeders	10	Clarkson	10 X 0.25/2.5	000 ( )
Return Water Tank	1	40,000 gal., 6m.dia.x6m.high	/0	* 000 ( )
Fresh Water Tank	1	25,000 gal.,5m.dia.x 5m.high	/0	* 000 ( )
Reagent Mixer	3	Barrel Type	3/2.25	000 ( )
TOTAL			Kw	$ 000

ATT LOUIS . M. BERNARD.

I have located a complete Zinc, Mill in Europe. 1000 T.P.D price $ 1,280,000 CAN. Cost to dismantel aprox $600,000 includes buildings delivered to docks. let me know if it is of interest.

John Conlon

TOTAL P.03

Comments: Authenticated sender is <manhatt@jhb-pop.iafrica.com>
From: "MANHATTAN MINING EQUIPMENT" <manhatt@iafrica.com>
To: wanderers@sysconc.com
Date: Fri, 1 Nov 1996 18:15:35 +0000
Subject: 2nd-hand Equipment
Reply-to: manhatt@iafrica.com
X-Confirm-Reading-To: manhatt@iafrica.com
X-pmrqc: 1
Return-receipt-to: manhatt@iafrica.com
Priority: urgent
X-Info: Evaluation version at mail-server

Manhattan Mining Equipment (Pty)Ltd
Tele: (011) 884 2897, fax: (011) 883 4701, Int. code + 2711, email: manhatt@iafrica.com, www.os2.iaccess.za/mme/ Suite 183, Private Bag X9, Benmore, 2010, South Africa
CK95/03942/07

RECONDITIONED & USED MINING, QUARRYING AND MINERAL PROCESSING EQUIPMENT

ATTENTION: REF: MME19898
MR. LOUIS M. BERNARD
1/11/96
WANDERERS IMPORTS LTD
wanderers@sysconc.com

RE: 2nd-hand Equipment

Thank you for your enquiry. We have a large variety of used equipment listed on our data-bank. We have reviewed your request and compiled a budget price list. As you know our currency has taken a turn for the worse and overseas companies are taking advantage of the low cost 2nd hand equipment available in South Africa.

ITEM 1 (coarse ore bin) REF:516 CANADIAN $
100 ton silo on steel legs
Condition: Very good
Our Price (budget) R35 000 10,500

ITEM 2 (Allis 4000MF) REF 130
Allis Chalmers 1650 gyratory crusher(equivalent to a 4.25' Symons), includes: crusher, motor, oil lubrication system, v-belts Condition: Good Note: Please advise on the size unit
Our Price (as is) R185000
Our Price (reconditioned) R280 000 84,000

ITEM 3 (conveyor) REF:423
(five) 0.6m W x 110m L conveyor, complete with drive, belt, idlers, frames Condition: Reconditioned
Our Price (budget) R1650/m 54,500
X 110m

CANADIAN $

ITEM 4 (fine ore bin) REF:324
2293m3 silo, 10m diameter x 34m high
Condition: Fair-good
Our Price (budget) R75 000
Dismantling cost (excludes re-erection), budget R200 000 } 82,500

ITEM 5 (weightometer) REF:460
Electronic weightometer, 40-50t/h
Condition: New
Our Price (budget) R25 000 7,500

ITEM 6 (ball mill) REF:80
4 off Vecor 12x16' pebble mills, 6600V 1000HP, liners also available, vapourmatic starter, trunion and pinion bearings still good,
Condition: Good..fair
Our Price (budget) R850 000 each 2556

ITEM 7 (cyclones) REF:
500mm diameter
Condition: Good
Our Price (budget) R4 000 1200

ITEM 8 (process pumps) REF:
1.5x1.25", 5x5" & vertical sump pump 2", rubber lined, complete with motors Condition: Reconditioned
Our Price (budget) R7 500 each 2300
(1.5x1.25")
R24 000 each (5x5" 60kW)
R20 000 each (5x5" 45kW)
R14 000 each (2" sump)

ITEM 9 (conditioner) REF:
2.5x2.5m conditioner tanks
Condition: Good
Our Price (budget) R40 000 each 12,00

ITEM 10 (agitator) REF:
Small size agitator, +-1m diameter 1.5m high
Condition: Good
Our Price (budget) R18 000 5500

ITEM 11(flotation cells) REF:
Various flotation cells, 2 to 8m3 listed
Condition: Reconditioned
Our Price (budget)- Wemco 66, 1.7m3 R15 000/cell 4500
Our Price (budget) Denver 300, 8m3 R20 000/cell 6000
Our Price (budget) Denver 500, 13m3 R25 000/cell 7500

ITEM 12 (vacuum pump) REF:
50HP vacuum pumps
Condition: Reconditioned
Our Price (budget) R20 000 each 6000

CANADIAN $

ITEM 13 (thickeners) REF:
7m diameter, includes tank
Condition: Reconditioned
Our Price Each (budget) R30 000 (drive & bridge) 9000
R40 000 (tank only) 12,000

ITEM 14 (drum filters) REF:
Drum filters, +-2m diameter x 2.5m long
Condition: Reconditioned
Our Price (budget) R60 000 each 18,000

ITEM 15 REF:261/9
(two) Rotary kilns, 2.7m diameter x 12m long, candle wax, tar and oil fired, 180 deg. C, one unit has been stripped Condition: Good
Our Price (budget) R175 000 each 52,500

ITEM 16 REF:
4m diameter x 4m high conc. tanks
Condition: Good
Our Price (budget) R60 000 18,000

ITEM 17 REF:
Conc. Tank agitator, +-2m diameter 3m high
Condition: Good
Our Price (budget) R45 000 13,500

ITEM 18 REF:
Return water tank, 6m diameter x 6m high AND 5m diam. X 5m high
Condition: Good (dismantled)
Our Price (budget) R45 000 each 13,500

Equipment can be refurbished upon request. C.O.D. Price Excludes V.A.T. & is ex works.

Current exchange rate: R4.74 = US$1.00

PAYMENT
By prior arranged and agreed form of payment.

CONDITION OF UNITS
All equipment is second-hand unless specified otherwise. Customer to inspect units prior to purchase to ensure they are in a satisfactory condition. MME accepts no responsibility for equipment purchased "AS IS" and not in an acceptable condition.

WARRANTY
Equipment is purchased "AS IS" unless otherwise specified.

DELIVERY
Please note that delivery quotations exclude insurance. If the customer wants goods to be insured, please inform us prior to shipment for our quotation. Manhattan does not take responsibility for goods damaged or lost in transit.

VALIDITY
Equipment is sold on a first come first served basis unless otherwise arranged. Quote is valid for 14 days. Subject to confirmation thereafter.

Best Regards
Many thanks for the enquiry.
We have received it and will be contacting you once we have compiled the information.

Best Regards

Chris Pouroullis
Managing Director BSc Eng (Mech)

Tel: +27 11 8842897
Fax: +27 11 8834701

Suite 183, Private Bag X9, Benmore, Sandton, 2010, South Africa

MINING TECHNOLOGIES INTERNATIONAL INC.

Mailing: P. O. Box 2845 Station A, Sudbury, Ontario, Canada P3A 5J3
Physical: 25 Fielding Road, Copper Cliff, Ontario, Canada P0M 1N0
Telephone: (705) 682-0623 Fax: (705) 682-4508

FAX TRANSMISSION

Page 1 of 7

Date: Oct 31, 1996

To:

Att: Mr. Louis Bernard

Fax: 905-849-9622

c.c.:

From: MARCEL DEMERS

Re: GRONG Project – NORWAY

Pricing, MTI 400 M & MTI 700 M.

We don't have a 5 yd machine.

Remote Control

$15,000 Receiver
18,000 Operator
33,000
say $35,000 total.

Regards
Marcel Demers

MINING TECHNOLOGIES INTERNATIONAL INC.

MTI PRICE LIST
EFFECTIVE 3/28/96

MINING TECHNOLOGIES INTERNATIONAL INC. Proposes to supply the following equipment:

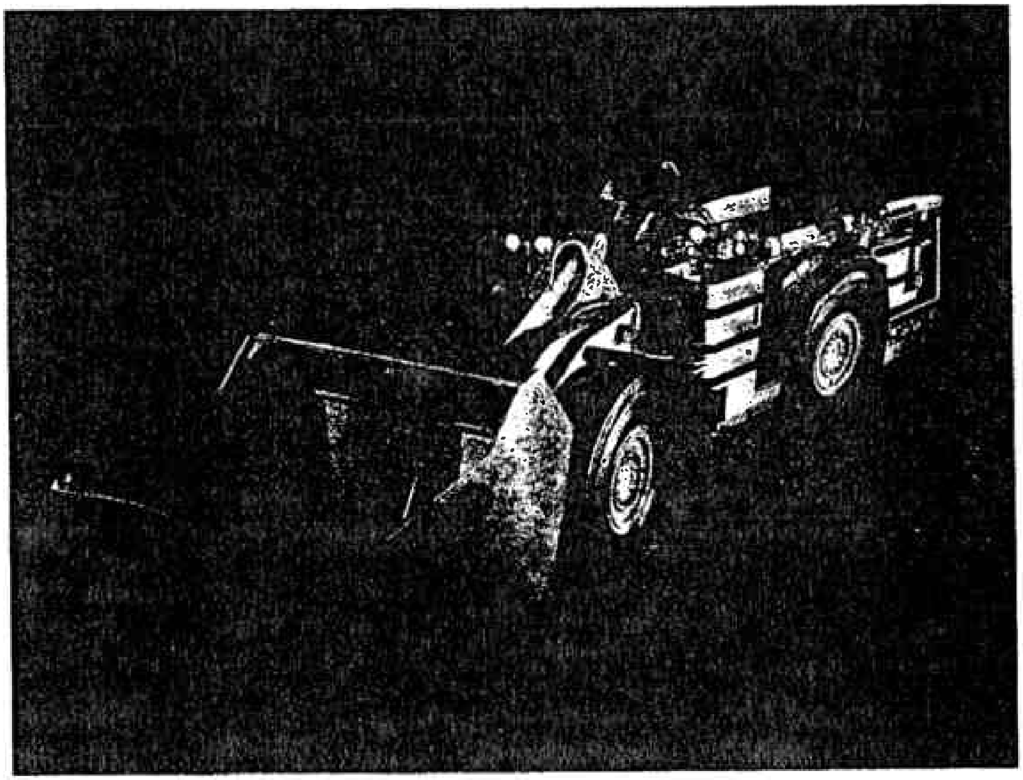
One only **MTI/JCI Model 401M** diesel powered, four wheel drive, center articulated 4.0 cubic yard LHD with a tramming capacity of 14,000 lbs (6,363 kg.) generally in accordance with the standard specifications:

Deutz F8L413FW diesel engine rated at 185 hp;
Engine pan protection;
Donaldson dry air cleaner;
Catalytic exhaust conditioner;
24 volt electric system;
Four lights, 2 front and 2 rear;
Clark C323 series single stage torque convertor;
Clark 32,000 series "modulated" four speeds
forward and reverse, powershift transmission;
Clark 16D planetary drive axles with no-spin
differentials front and rear;
Trunnion oscillated rear axle;
Totally enclosed wet disc service brake;
Spring applied emergency/park drive line disc brakes;
Variable displacement load sensing hydraulic system;
Heavy duty standard 4.0 cubic yard bucket;
17.5 x 25 L-5 slick tires;
Side seated operator compartment with fully
adjustable mechanical suspension seat;
Hydraulic joystick steering control;
Monostick boom and bucket control system;
Audio visual alarm system;
Standard gauges and instrumentation.

PRICE .... $ 310,000

"Three hundred ten thousand dollars"

Load Haul Dump Vehicles



351M – 401M Series LHD

Standard Features

- 13,500–14,500 lbs. (6,136–6,590 kg) Rated Load Capacity
- 3.5–4.0 yd³ (2.7–3.0 m³) Bucket Capacity
- 21,500 lbs. (9,772 kg) Breakout Force
- 75" (1.93 m) OA Width
- Heavy Duty Powertrain
- Option of Detroit Diesel or Deutz Engine
- Clark C320 Series Torque Converter
- Clark Modulated 32,000 Series Transmission
- Clark 16D Planetary Drive Axles with Posi-Stop Brakes
- Trunnion Mounted Rear Axle Oscillation
- Load Sensing Variable Displacement Hydraulics
- Full Flow Pressure and Return 10 Micron Hydraulic Filtering
- Extended Boom Reach for Truck Loading
- Largest Operator Compartment in Its Class

MINING TECHNOLOGIES INTERNATIONAL INC.

Innovative technologies. Unparalleled support.

Performance Specifications

Rated Load Capacity

351M	13,500 lbs. (6,136 kg)
401M	14,500 lbs. (6,590 kg)

Standard Bucket Capacity

Semi Nominal (heaped)

351M	3.5 yd³ (2.7 m³)
401M	4.0 yd³ (3.1 m³)

Struck

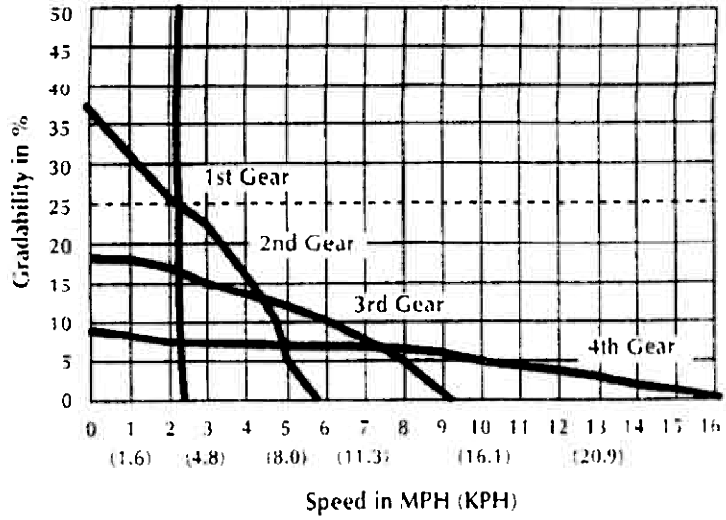
351M	2.9 yd³ (2.2 m³)
401M	3.3 yd³ (2.5 m³)

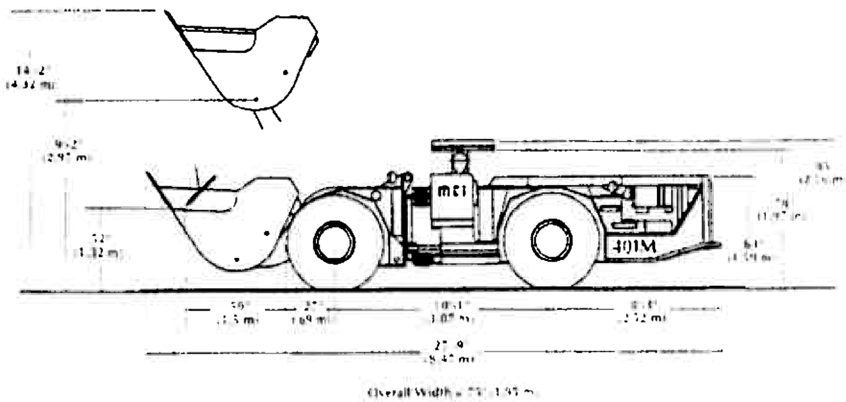
Maximum Breakout Force
(bucket roll back)

351M	31,915 lbs. (14,506 kg)
401M	31,915 lbs. (14,506 kg)

Operating Weight (empty)

351M	39,000 lbs. (17,727 kg)
401M	40,000 lbs. (18,182 kg)





MINING TECHNOLOGIES INTERNATIONAL INC.

351M–401M Series LHD

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Fax: (705) 692-4850

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P3Y 1L7
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Fax: (705) 682-4308
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U.S. Sales Office:
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Fax: (303) 892-1408
Toll free: (Canada & U.S.A.)
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Fax: (303) 892-2830
Toll free: (Canada & U.S.A.)
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25 Soetdoring Str. Dalpark
Brakpan, 1540
Republic of South Africa
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Fax: 27-11-915-8330

Australia & South East Asia:
26-42 Cooper Road
Jandakot 6164, W.A.
Australia
Telephone: 61-9-417-8488
Fax: 61-9-417-8747

Authorized Representative

MINING TECHNOLOGIES INTERNATIONAL INC

MTI PRICE LIST
EFFECTIVE 3/28/96

MINING TECHNOLOGIES INTERNATIONAL INC. Proposes to supply the following equipment:

One Only

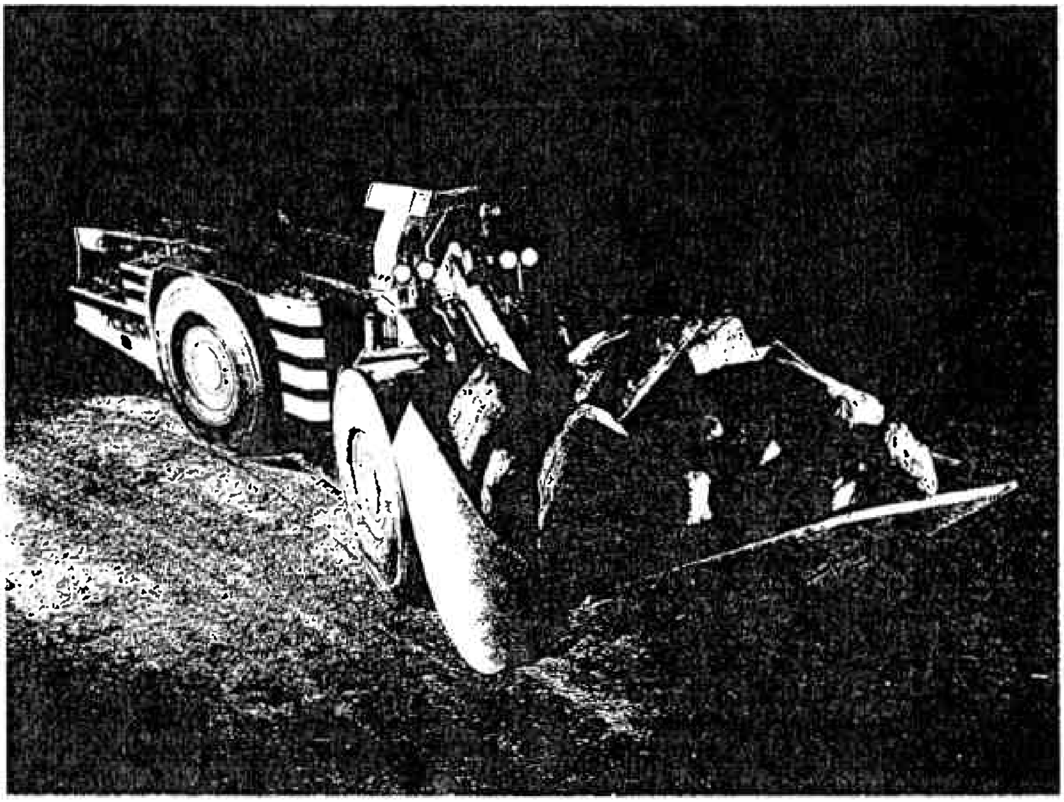
MTI/JCI Model 700M diesel powered, four wheel drive, center articulated 7.0 cubic yard 21,000 lb. (9,545 kg) tramming capacity LHD unit generally in accordance with the standard specifications and fitted with:

Caterpillar model 3306 DITA engine rated at 250 HP;
Mesabi double pass replaceable core radiator;
Engine skid pan;
Dry type air cleaner;
24 volt electric system;
Lights - 4 front and 4 rear;
Heavy duty marine type alternator;
Catalytic exhaust scrubber;
Clark 8000 series industrial type torque convertor;
Clark 4000 series "modulated" powershift transmission with 4 speeds forward and reverse;
JCI Rockwell extra heavy duty planetary drive axles with no-spin differentials front and rear;
Twenty degree rear axle oscillation;
Independent totally enclosed wet disc service brakes;
Spring applied emergency/park driveline disc brake system, mounted on front axle;
Bearing mounted center pin type hydraulic steering;
Heavy duty standard 7.0 cubic yard capacity bucket;
18:00 x 25 L-5 tires;
Side seated operators position with fully adjustable ergonomic seat;
Hydraulic joystick steering control;
Monostick boom and bucket control;
Audio visual alarm system;
Standard gauges and instrumentation;
Gear pump hydraulic system (Note extra price option 22).

PRICE . . . . $ 425,000 US

" FOUR HUNDRED TWENTY FIVE DOLLARS

Load Haul Dump Vehicles



700M LHD

Standard Features

- 21,000 lbs. (9,545 kg.) Rated Load Capacity
- 7.0 yd³ (5.4 m³) Bucket Capacity
- 36,780 lbs (16,718 kg) Breakout Force
- 96"–104" (2.44–2.64 m) OA Width
- Heavy Duty Powertrain
- Option of Caterpillar, Detroit Diesel, or Deutz Engine
- Clark C8000 Series Torque Converter
- Rockwell 5225 Planetary Axles with Optional SAHR Brakes
- Trunnion Mounted Rear Axle Oscillation
- Optional Load Sensing Variable Displacement Hydraulics
- Optional Full Flow Pressure and Return 10 Micron Hydraulic Filtering
- Optional High Lift Boom for Truck Loading
- Largest Operator Compartment in Its Class

MINING TECHNOLOGIES INTERNATIONAL INC.

Innovative technologies. Unparalleled support

Performance Specifications

Rated Load Capacity

21,000 lbs.
(9,545 kg)

Standard Bucket Capacity

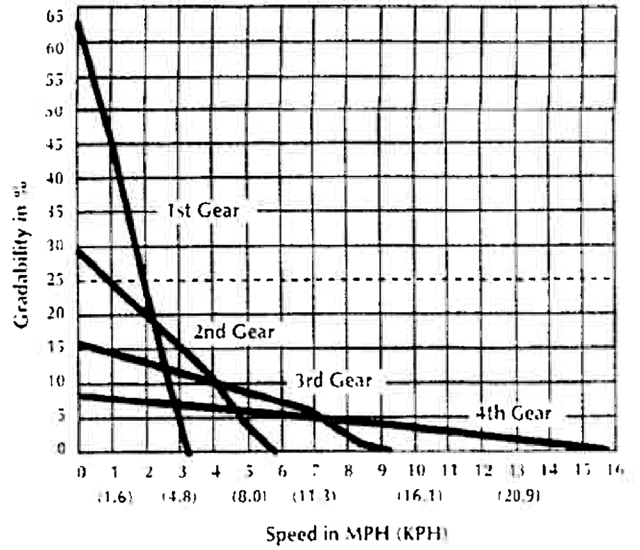
Semi Normal Heaped
7.0 yd³
(5.4 m³)
Struck
6.2 yd³
(4.8 m³)

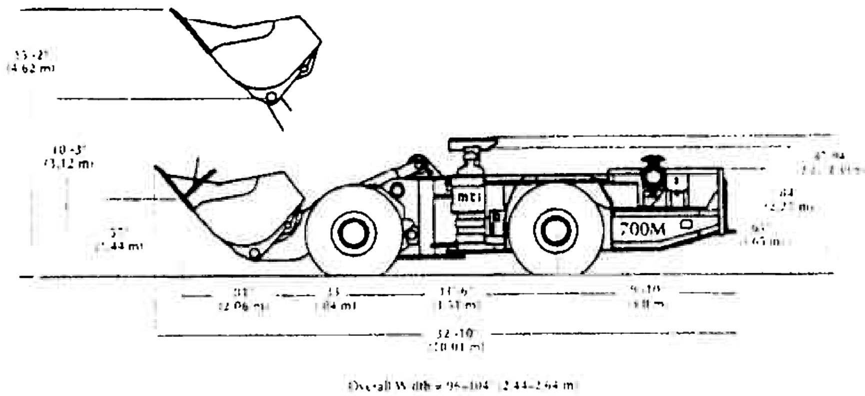
Maximum Breakout Force
(bucket roll back)

36,780 lbs
(16,718 kg)

Operating Weight (empty)

61,500 lbs
(27,957 kg)





MINING TECHNOLOGIES
INTERNATIONAL INC.

700M LHD

Authorized Representative

Head Office:
P.O. Box 2097, Station A
Sudbury, Ontario, Canada
P3A 4R8
145 Magill Street
Lively, Ontario, Canada
P3Y 1K6
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Toll free: (Canada & U.S.A.)
1-800-659-4955

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25 Soerdonne Str. Dalpark
Brakpan, 1540
Republic of South Africa
Telephone: 27-11-915-2770
Fax: 27-11-915-8330

Australia & South East Asia:
26-42 Cooper Road
Jandakot 6164 W.A.
Australia
Telephone: 61-9-417-8488
Fax: 61-9-417-8747

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Physical: 25 Fielding Road, Copper Cliff, Ontario, Canada POM 1NO

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FAX TRANSMISSION

Page 1 of 18

Date: Oct 31, 1996

To:

Att: Mr. Louis Bernard

Fax: 905-849-9622

c.c.:

From: Marcel Desmers

Re: Grong Project - Norway

MINING TECHNOLOGIES INTERNATIONAL INC.

MTI PRICE LIST
EFFECTIVE 3/28/96

MINING TECHNOLOGIES INTERNATIONAL INC., Proposes to supply the following equipment:

One only

MTI/JCI, Model C1504 diesel powered, four wheel drive, center articulated, center oscillation, 15 ton capacity **"Container Vehicle"**, with the JCI "J" hook pick up system as outlined in attached proposal and complete with:

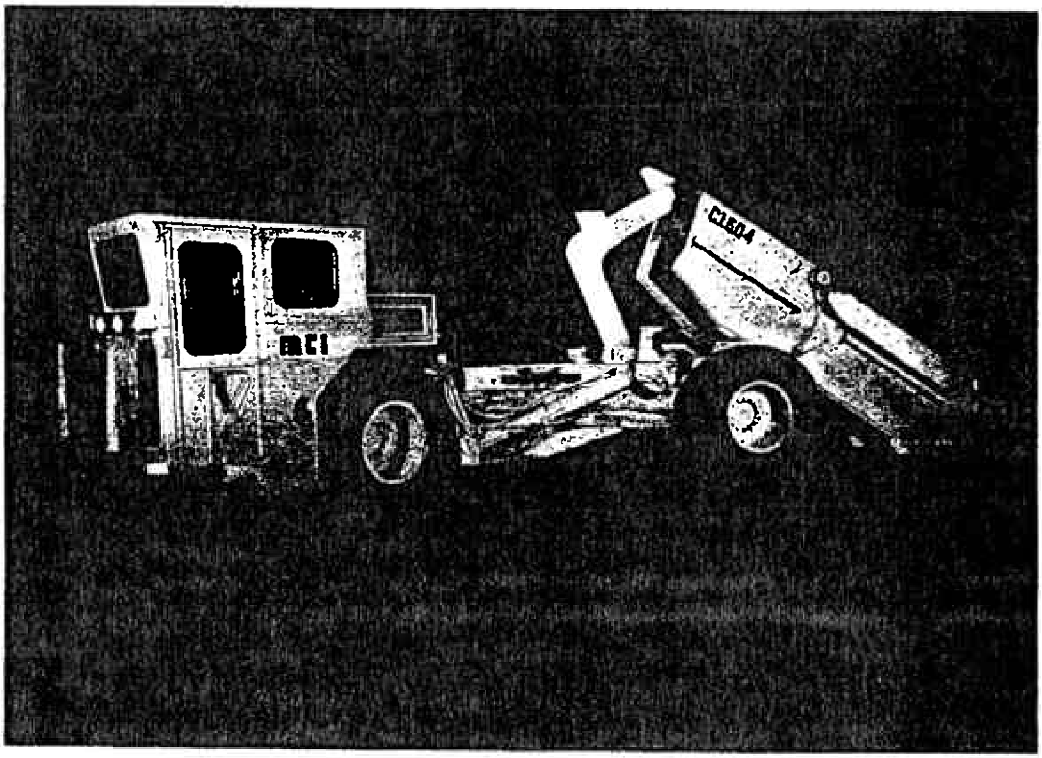
Deutz F8L-413FW diesel engine rated at 185 HP
Catalytic exhaust conditioner;
Donaldson dry type air filter;
24 volt electric system;
Lights - 4 front and 2 rear;
Clark CL323 industrial type torque convertor with manual lock up;
Clark 32000 series powershift transmission, four speeds forward and reverse;
John Deere 1400 series planetary drive axles front and rear;
Independent hydraulic wet disc service brakes;
Spring applied hydraulic released emergency/park driveline disc brake mounted on rear axle;
15 degree oscillation between front and rear frame;
17.5 x 25, L-5 tires;
Side seated operators compartment with adjustable mechanical suspension seat;
Hydraulic stick steering control;
Prime mover includes "J" hook assembly and sub frame complete with necessary cylinders, valves, pumps and hoses;

Two containers complete with automatic tailgate, hook pick up system, end skids, suitable for carrying 15 tons of material weighing a minimum of 3,000 lbs per cubic yard.

PRICE $485,000

"FOUR HUNDRED EIGHTY FIVE THOUSAND"

Four Wheel Drive Container Truck



C1504 Container Truck

Standard Features

- 30,000 lbs. (13,636 kg) Rated Load Capacity
- Containers Sized to Material Density
- "J" Hook Pick Up with Patented Dump Mechanism
- 8' (2.43 m) OA Width
- Heavy Duty Powertrain
- Option of Detroit Diesel or Deutz Engine
- Clark CL323 Lock Up Torque Converter
- Clark 32,000 Series Transmission
- John Deere Inboard Planetary Axles with Wet Disc Brakes
- Center Hinge Oscillation
- Load Sensing Variable Displacement Hydraulics
- Full Flow Pressure and Return 10 Micron Hydraulic Filters
- Large Side Seated Ooerators Compartment Captains Chair

MINING TECHNOLOGIES
INTERNATIONAL INC.
Innovative technologies. Unparalleled support.

Performance Specifications

Rated Load Capacity

30,000 lbs.
(13,636 kg.)

Standard Box Capacity

Semi Nominal Heaped

8.9 yd³
(6.8 m³)

Struck

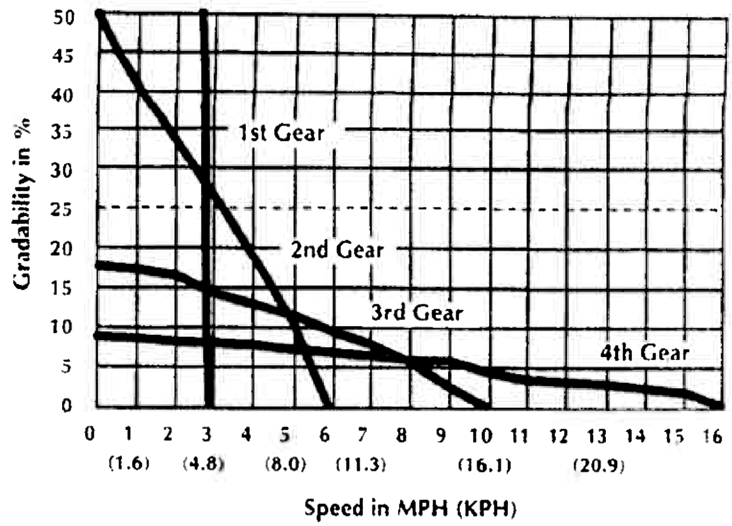
7.5 yd³
(5.7 m³)

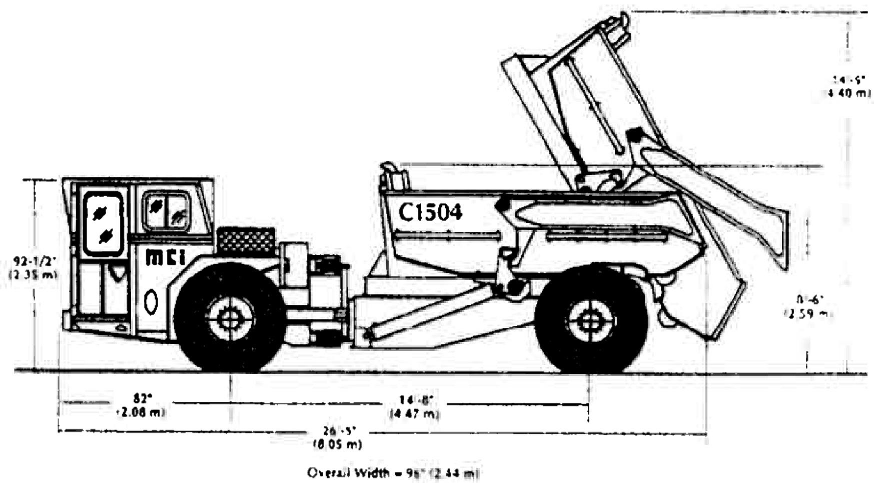
Box Dump Height

14'-5"
(4.4 m)

Operating Weight (empty)

40,300 lbs.
(18,318 kg.)





MINING TECHNOLOGIES
INTERNATIONAL INC.

C1504 Container Truck

Head Office:
P.O. Box 2097, Station A
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Fax: (705) 692-4850

Canadian Sales Office:
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25 Fielding Road
Lively, Ontario, Canada
P3y 1L7
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Fax: (705) 682-4508
Toll free: (Canada & U.S.A.)
1-800-461-4094

U.S. Sales Office:
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Fax: (303) 892-1408
Toll free: (Canada & U.S.A.)
1-800-659-4955

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Telephone: (303) 892-5800
Fax: (303) 892-2830
Toll free: (Canada & U.S.A.)
1-800-659-4955

South Africa:
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Fax: 27-11-915-8330

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Australia
Telephone: 61-9-417-8488
Fax: 61-9-417-8747

Authorized Representative

MINING TECHNOLOGIES INTERNATIONAL INC.

MTI PRICE LIST
CDJ-2SHBE
EFFECTIVE 1/1/96

Mining Technologies International proposes the following for your consideration:

One only MTI model CDJ-2SHBE hydraulic two boom, four wheel drive, centre articulated, parallel drilling diesel/electric jumbo complete with:

MTI carrier complete with HC40 drifter and HCCF14 feed and including the following:

- F6L912W diesel engine
- 3 speed 18000 Clark auto transmission
- oil bath air cleaner c/w pre-cleaner and cleanout
- ECS exhaust
- heavy duty inboard planetary axles complete with no spin differentials Clark 176 service brakes, fully hydraulic applied totally enclosed inboard wet disc brakes
- 12:00 x 24 Trojan core tires
- audio visual instrumentation including: oil pressure gauge, hourmeter, engine temperature, ammeter, horn activated by high engine temperature, low engine oil pressure or blower belt failure
- closed centre hydraulic system for operation of brakes and steering
- two (2) boom end jacks
- two (2) engine end jacks

Electrical (575 volt, 3phase)

- 2 only 50 hp main electric motor
- lubricator for front head using mine air
- main electrical panel including; hourmeter, amp meter, all annunciation systems and one 110 volt, 15 amp utility receptacle (ground fault style)
- manual sequential motor starting panel adjacent to the operator c/w emergency stop button for system shutdown
- safetys shutdowns monitor low oil level and high oil temperature, low water pressure and low air pressure
- easily accessible collector rings suitable for 3 phase power ground fault and ground check systems
- two adjustable 500 watt quartz halogen drilling lights
- 55 amp alternator (90 amp optional)
- 2 only 90 amp hour batteries
- 4 halogen lights boom end and 2 at engine end
- electric wiring wrapped in fire resistant sleeves
- master disconnect switch
- diesel driven hydraulic powered cable reel (less cable)

Booms, Feeds, Drills and Controls

- 2 only SHBE auto parallel drill boom of X-Y movement configuration c/w 270 deg feed rotation, 1.0 meter boom extension and universal head to allow drilling 90 degrees to one side or the other
- feed automatics to return the drill to the collaring position following hole completion
- automatic collaring control to prevent rods from sticking when poor ground is encountered

MINING TECHNOLOGIES
INTERNATIONAL INC.

MTI PRICE LIST
CDJ2-SHBE
EFFECTIVE 1/1/96

Booms, Feeds, Drills, and Controls(con't)

- 2 only HCCF feed suitable for drilling 13ft long hole using 14 ft steel
- 2 only monoblock control system incorporating these features:
  1)automatic collaring
  2)anti-jamming
  3)automatic water monitoring with feed return should low volume or pressure be encountered
  4)auto return to start position following hole completion
  5)control levers for operating percussion, rotation, bit removal and feed control

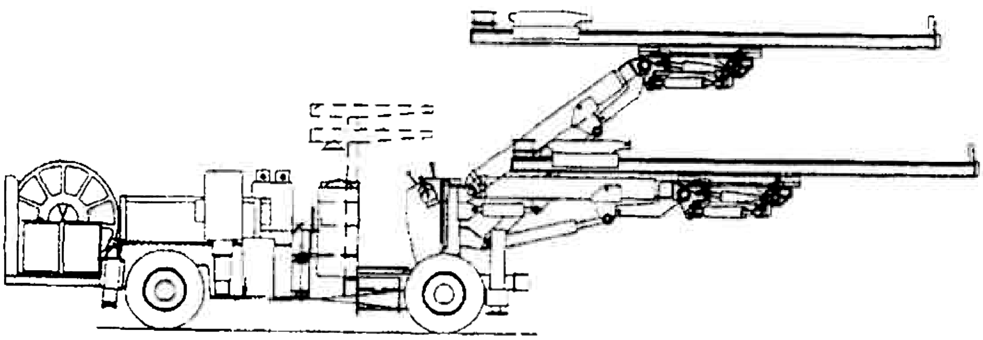
Drills and Hydraulics

- 2 only third generation I.R. Montabert HC40 drifter incorporating energy recuperation system
- 2 only 50 hp motor , driving a gear pump for drilling rotation, percussion and a piston pump for boom movements

Price...$ 525,000 CDN

"Five hundred twenty thousand CDN"

Two-Boom Hydraulic Jumbo



CDJ-2SHBe

The CDJ-2SHBe is a two-boom hydraulic rig designed for drilling vertical, horizontal and angled holes in underground production headings. It features two hydraulic percussion drifters and can drive headings 16 ft. by 22 ft. (4.88 m by 6.71 m).

Standard Features

- Deutz F6L 912W diesel engine, 82 hp (54 kw), air cooled, catalytic exhaust purifier, dry element air cleaner
- Clark 18000 series powershift transmission with torque converter, 3 speeds forward/reverse
- 50 hp (37 kw) electric power pack driving a tandem gear pump for percussion/rotation and a pressure compensated piston pump for feed/boom movements
- 24v negative ground engine electrics
- SHBe booms for 352 sq. ft. (32.7 sq. m) of coverage
- HC-40 Montabert drifters
- HCCF-14 aluminium feed (14 ft. (4.27 m) rod)
- 10 micron pressure and return filters
- Front leveling jacks
- Safety shut-downs for:
  - engine temperature
  - engine oil pressure
  - hydraulic oil temperature
  - hydraulic oil level
- Cable reel with capacity for 350 ft. (107 m) of cable
- Two 500 watt drill lights

MINING TECHNOLOGIES INTERNATIONAL INC.

Innovative technologies. Unparalleled support.

General Specifications

Total Length
43 ft. 1 in. (13.13 m)

Width
78 in. (1.98 m)

Height
84 in. (2.13 m)

Weight
39,000 lb. (17,727 kg)

Articulation Angle
40 degrees

Ground Clearance
12 in. (0.30 m)

Gradeability
25% (15°)

Turning Radius
Inside 13 ft. 2 in. (4.01 m)
Outside 23 ft. 7 in. (7.19 m)

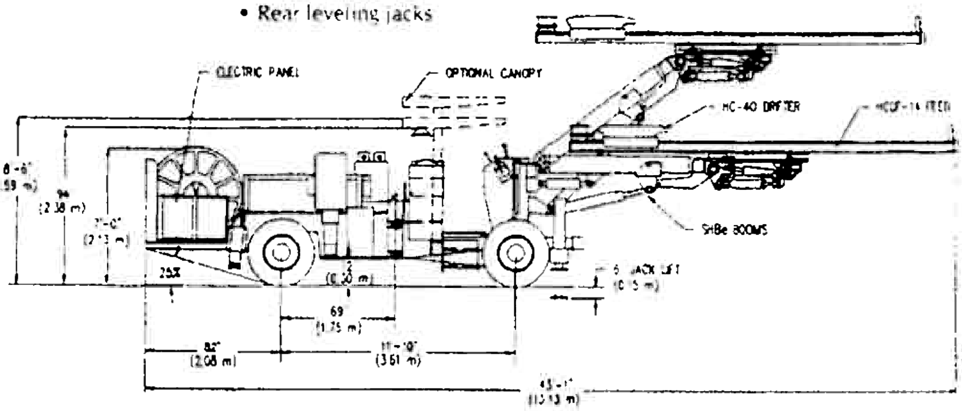
Tire Size
10.00 x 20 solid

Tramming Speed
6 mph (9.7 km/hr.)

Face Coverage (H x W)
16 ft. x 22 ft. (4.88 m x 6.71 m)

Options

- BUC-20 or BUC-24 booms at 72 in. (1.83 m) wide
- BUC-40 booms at 84 in. (2.13 m) wide
- MHB booms at 84 in. (2.13 m) wide
- HC-80 drifter (c/w a 60 hp (45 kw) power pack)
- Fire suppression system
- Automatic actuation for fire suppression system
- Operator's canopy
- Air compressor
- Water booster pump
- Rear leveling jacks



MINING TECHNOLOGIES INTERNATIONAL INC.

CDJ-2SHBe

All specifications, while accurate at time of publication, are subject to change without notice. Some features shown in photos and illustrations may be optional.

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Telephone: 1-705-692-3661
Fax: 1-705-692-4850

Canadian Sales Office:
P.O. Box 2845, Station A
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P3A 5J3
25 Fielding Road
Lively, Ontario, Canada
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Telephone: 1-705-682-0623
Fax: 1-705-682-4508
Toll free: (Canada & U.S.A.)
1-800-461-4094

U.S. Sales Office:
4955 Bannock Street
Denver, Colorado, U.S.A.
80216
Telephone: 1-303-892-5800
Fax: 1-303-892-1408
Toll free: (Canada & U.S.A.)
1-800-659-4955

International Sales Office:
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1-800-659-4955

South Africa:
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25, Soetdoring Str. Dalpark
Brakpan 1540
Republic of South Africa
Telephone: 27-11-915-2770
Fax: 27-11-915-8330

Australia & South East Asia:
26-42 Cooper Road
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Telephone: 61-9-417-8488
Fax: 61-9-417-8747

Authorized Representative

MINING TECHNOLOGIES INTERNATIONAL INC

MTI PRICE LIST
EFFECTIVE 3/28/96

MINING TECHNOLOGIES INTERNATIONAL INC. Proposes to supply the following equipment:

One only

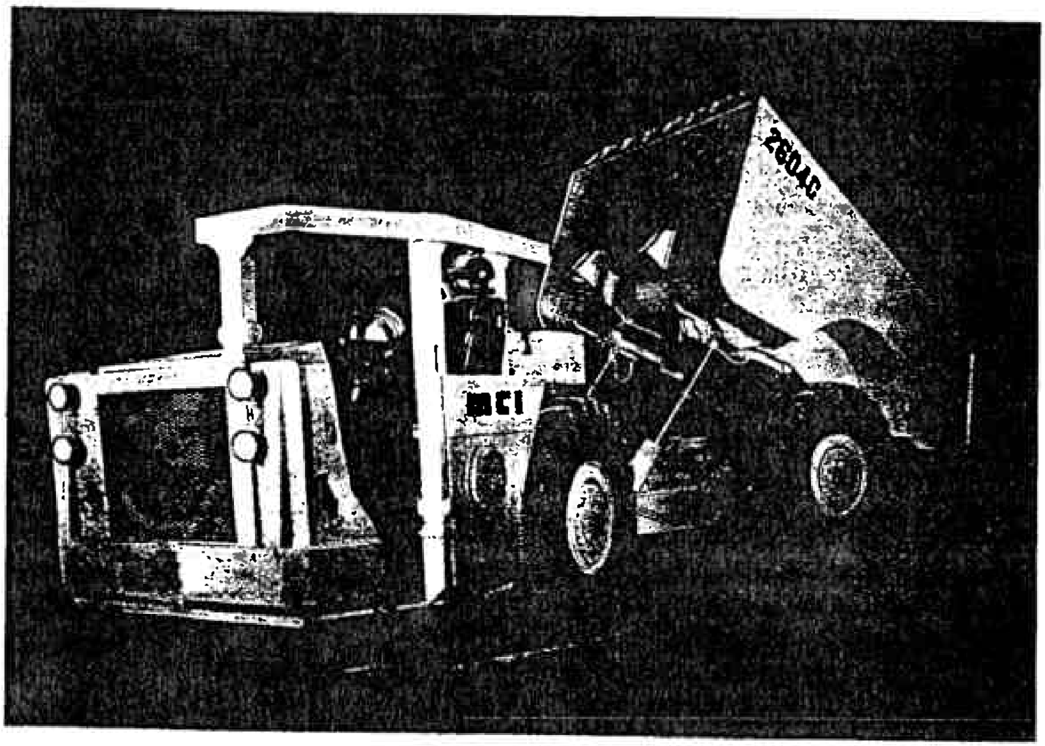
MTI/JCI Model 3004C diesel powered, four wheel drive, center articulated, center oscillation 30 ton capacity End Dump Truck, in accordance with standard specifications and fitted with:

Caterpillar 3306DITA diesel engine rated at 270HP;
Double pass, side mounted, Mesabi type radiator with hydraulically driven fan;
Catalytic exhaust conditioner;
Donaldson dry type air cleaner;
24 volt electric system;
Lights - 4 front and 2 rear;
Caterpillar industrial type torque convertor, complete with automatic lock-up;
Caterpillar powershift transmission, five speeds forward and two speed reverse;
Caterpillar planetary drive axles front and rear;
Independent hydraulically actuated wet disc type service brakes on front and rear axle;
Spring applied, hydraulically released emergency/park driveline disc brake mounted on rear axle;
15° center hinge oscillation;
Heavy duty 20.7 cubic yard SAE heaped capacity rated dump box;
21:00 x 25, L-5 tires;
Side seated operator's compartment with adjustable spring suspension seat;
Hydraulic stick steering controls;
Audio/visual alarm system;
Standard gauges and instrumentation.

PRICE .... $ 420,000 CDN

"FOUR HUNDRED TWENTY THOUSAND"

Four Wheel Drive End Dump Truck



2604C–3004C Series Truck

Standard Features

- 52,000–60,000 lbs. (23,636–27,272 kg) Rated Load Capacity
- 18.6–20.7 yd^3 (14.2–15.9 m^3) Box Capacity
- 110"–116" (2.79–2.95 m) OA Width
- Heavy Duty Powertrain
- Option of Caterpillar or Detroit Diesel Engine
- Side Mounted Swing Out Radiator
- CAT Torque Converter with Automatic Lock Up
- CAT Hydraulic Retarder
- CAT Planetary Transmission
- CAT Inboard Planetary Axles with Wet Disc Brakes
- Center Hinge Oscillation
- High Capacity Bowl Design Box
- Load Sensing Variable Displacement Hydraulics
- Full Flow Pressure and Return 10 Micron Hydraulic Filters
- Large Side Seated Operators Compartment Captains Chair

MINING TECHNOLOGIES
INTERNATIONAL INC.
Innovative technologies. Unparalleled support

Performance Specifications

Rated Load Capacity

2604 52,000 lbs.
(23,636 kg.)
3004 60,000 lbs.
(27,273 kg.)

Standard Box Capacity

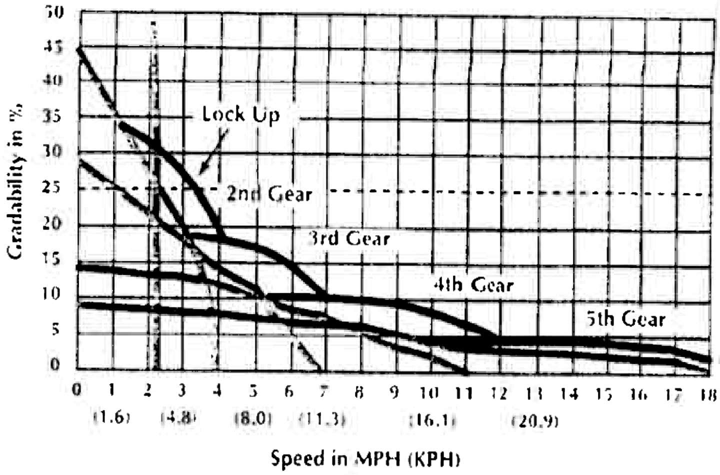
Semi Nominal Heaped
2604 18.6 yd³
(14.2 m³)
3004 20.7 yd³
(15.9 m³)
Struck
2604 14.9 yd³
(11.4 m³)
3004 16.8 yd³
(12.9 m³)

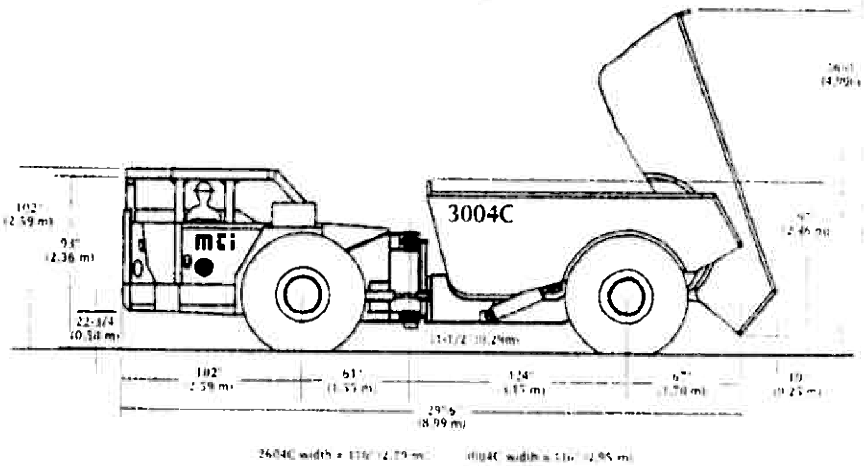
Box Dump Height

2604 15'-8"
(4.7 m)
3004 16'-1"
(4.9 m)

Operating Weight (empty)

2604 52,400 lbs.
(23,818 kg.)
3004 55,000 lbs.
(25,000 kg.)





MINING TECHNOLOGIES
INTERNATIONAL INC.

2604C–3004C Series Trucks

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Fax: (705) 682-4508
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Authorized Representative

MINING TECHNOLOGIES INTERNATIONAL

MTI PRICE LIST
EFFECTIVE 3/28/96

MINING TECHNOLOGIES INTERNATIONAL INC. Proposes to supply the following equipment:

One only

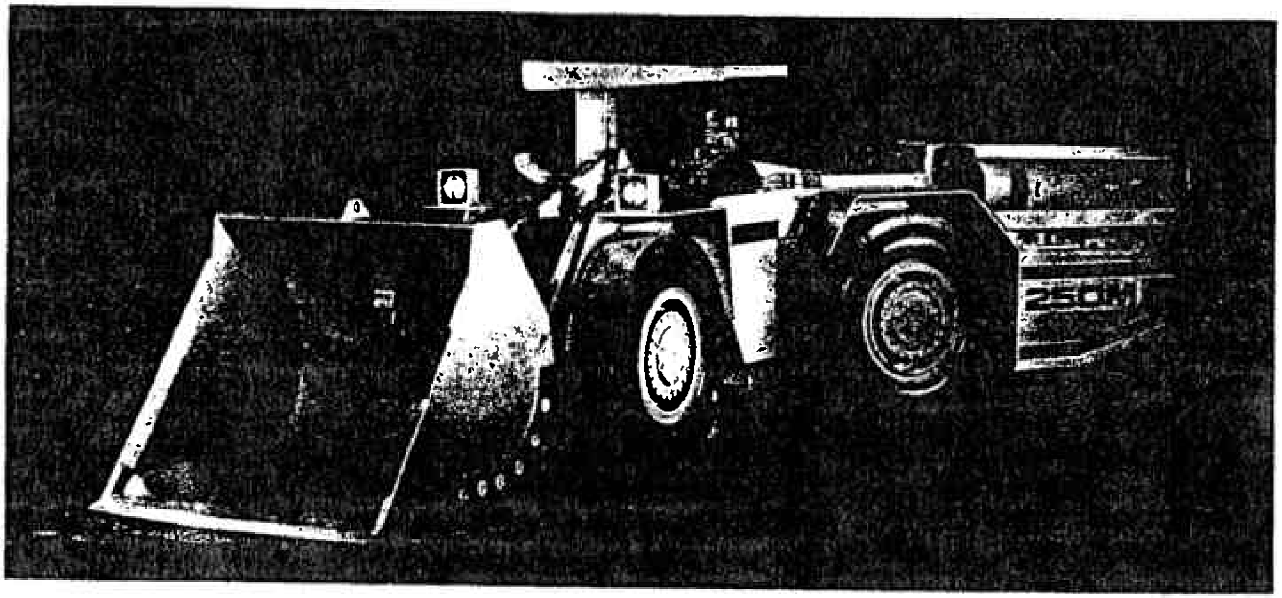
MTI/JCI Model 250M diesel powered, four wheel drive, center articulated 2.5 cubic yard LHD unit, generally in accordance with the standard specifications and to be supplied complete with:

Deutz F5L413FRW diesel engine rated at 116 hp;
Donaldson dry type air cleaner;
Catalytic exhaust conditioner;
Straight 24 volt electric system;
Four lights - 2 front and 2 rear;
Heavy duty marine type alternator;
Clark C-270 series industrial torque convertor;
Clark 32000 series "modulated" four speeds powershift transmission;
Clark 14D outboard planetary drive axles with no-spin differentials front and rear;
Trunnion mounted rear axle;
Totally enclosed wet disc service brakes;
Spring applied emergency/park driveline disc brake;
Variable displacement load sensing hydraulic system;
2.5 cubic yard standard bucket;
12:00 x 24, L-5 slick tires;
Side seated operator compartment with fully adjustable mechanical suspension seat;
Hydraulic joystick steering control;
Monostick boom and bucket control system;
Audio/visual alarm system;
Standard gauges and instrumentation.

PRICE...$ 250,000 CDN.

"TWO HUNDRED FIFTY THOUSAND"

Load Haul Dump Vehicles



250M–300M Series LHD

Standard Features

- 7,500–9,000 lbs (3,409–4,090 kg) Rated Load Capacity
- 2.5-3.0 yd^3 (1.9–2.3 m^3) Bucket Capacity
- 16,870 lbs. (7,668 kg) Breakout Force
- 61"–68" (1.55–1.73 m) OA Width
- Heavy Duty Powertrain
- Deutz F5L-413-FRW Engine
- Clark C270 Series Torque Converter
- Clark Modulated 32,000 Series Transmission
- Clark 14D Planetary Drive Axles with Wet Disc Service Brakes
- Trunnion Mounted Rear Axle Oscillation
- Load Sensing Variable Displacement Hydraulics
- Full Flow Pressure and Return 10 Micron Hydraulic Filtering
- Extended Boom Reach for Truck Loading
- Largest Operator Compartment in Its Class

MINING TECHNOLOGIES INTERNATIONAL INC.

Innovative technologies. Unparalleled support

Performance Specifications

Rated Load Capacity

250M 7,500 lbs.
(3,409 kg.)
300M 9,000 lbs.
(4,090 kg.)

Standard Bucket Capacity

Semi Nominal Heaped

250M 2.5 yd³
(1.91 m³)
300M 3.0 yd³
(2.30 m³)

Struck

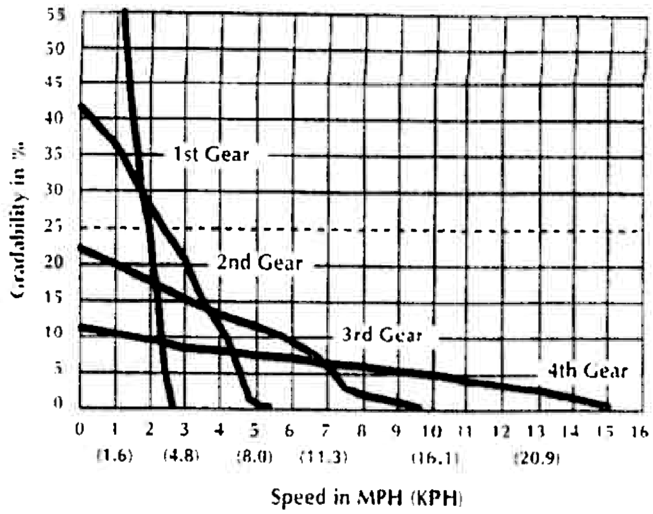
250M 2.1 yd³
(1.60 m³)
300M 2.4 yd³
(1.83 m³)

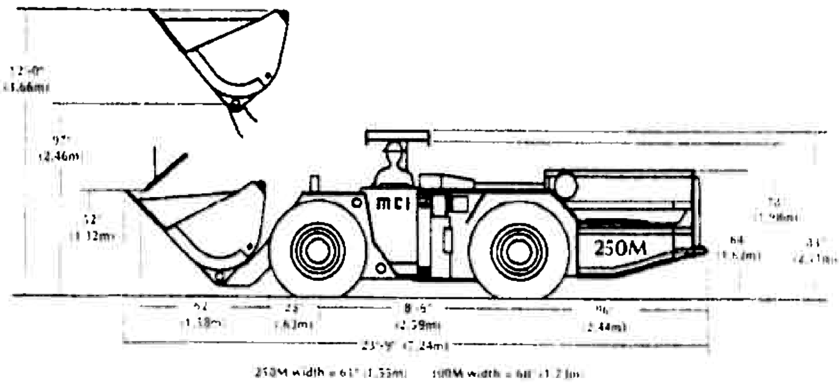
Maximum Breakout Force
(bucket roll back)

250M 21,210 lbs.
(9,641 kg.)
300M 16,870 lbs.
(7,668 kg.)

Operating Weight (empty)

250M 26,000 lbs.
(11,818 kg.)
300M 26,450 lbs.
(12,023 kg.)





MINING TECHNOLOGIES
INTERNATIONAL INC.

250M – 300M Series LHD

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Authorized Representative

MINING TECHNOLOGIES
INTERNATIONAL INC.

MTI PRICE LIST
CD360M
EFFECTIVE 1/1/96

MTI proposes the following equipment for your consideration:

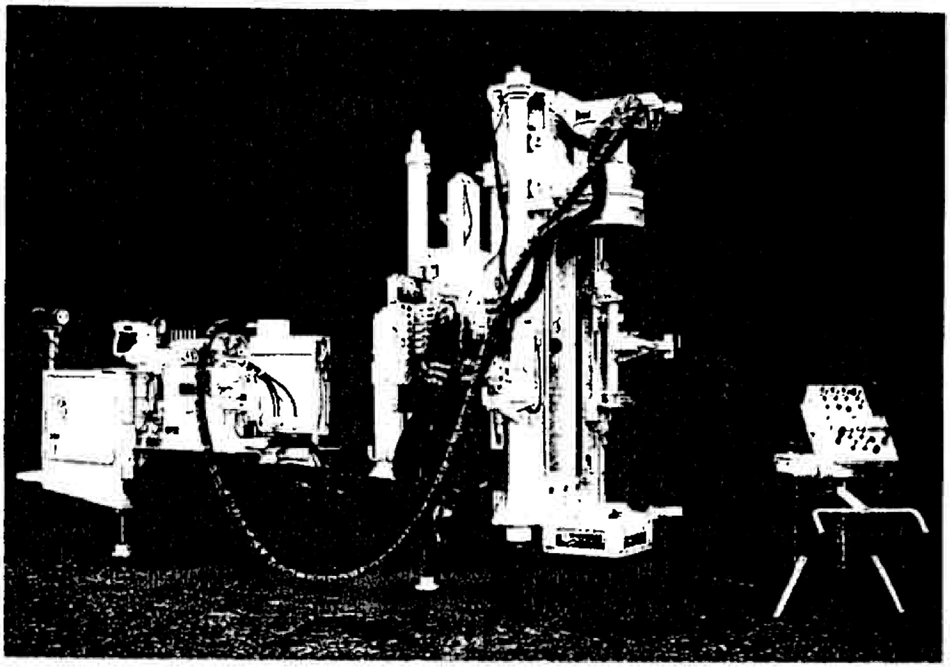
One only "Continuous" model CD360M In-The-Hole drill complete with:

- crawler track drive carrier
- spring applied, hydraulically released, totally enclosed wet disc brakes
- modular frame construction
- waterproof start-stop station on console
- dual hydraulic mast cylinder
- moveable tram controls for front, rear, left or right side operation
- hydraulic rotation motor suitable for up to 6.5" holes
- mechanized breakout at worktable and drivehead
- drive motor carrier
- high strength formed channel mast c/w double mast stingers
- patented double acting feed cylinder mounted in a trunnion block with gimbal movement
- four independent 10 stroke stabilizer jacks for easy set up
- 360 degree mast rotation
- 23" lateral mast side shift and 25" of mast anchoring allow the driller to properly anchor the mast for better collaring in any drill setup. These features also enable a blooie to make a better seal
- main stinger column to keep the machine stable regardless of the mast's rotation
- non pressurized lubricator to allow unit to be operated with 350 psi of air
- hydraulic circuits complete with 100 mesh suction strainers
- 10 micron absolute, no bypass pressure filter c/w visual indicator
- 10 micron absolute, 50 psi bypass return filter c/w visual indicator
- 30 hp TEFC electric motor, 60 cycle, 575 volts
- hourmeter
- float valve
- three parts/service manuals
- two bar

Price...$ 250,000 CDN

1) One only Montabert HC-80 Top
HAMMER DRILL.
Price . . . . $ 123,000 CDN

In-the-Hole Production Drills



CD 360M

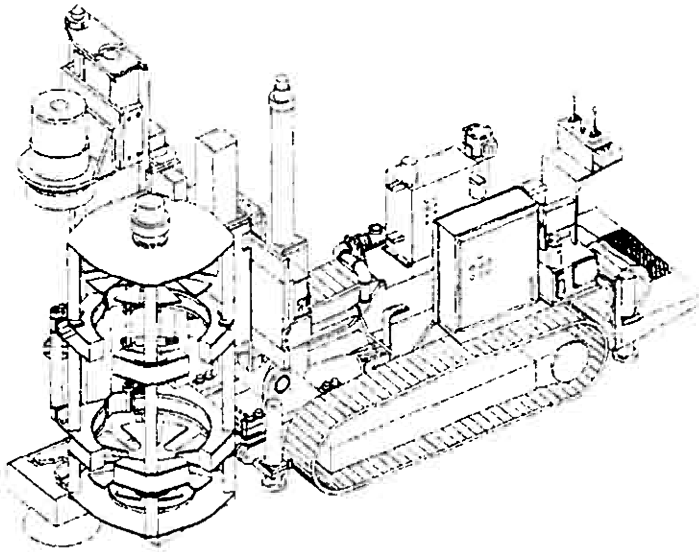
- ITH production drilling from 3.5 in. (89 mm) to 8.5 in. (216 mm)
- Hole depths to 750 ft. (229 m) x 8in. (203 mm) hole
- 360° tower rotation enables full-circle drilling without repinning
- Hydraulic rod lock, breakout wrench and optional mechanical arm for easy rod changing
- Towers for rod lengths from 4 ft. (1.22 m) to 6 ft. (1.82 m)
- Dual tower stabilizing jacks and hydraulic tower extension
- Hydraulic remote control for all drilling functions
- Electric drill light packages
- High pressure water injection

MINING TECHNOLOGIES
INTERNATIONAL INC.

Innovative technologies. Unparalleled support

Leading the way in blasthole drilling technology.

The basic CD 360M evolved with cooperation from some of Canada's leading mining operations. This sturdy, stable ITH drill is capable of withstanding the harsh demands of high productivity underground drilling. The modular design permits field upgrades of most options to meet any changing production needs or to exploit opportunities for increased productivity.



Features and Benefits

- Transportable in a 12 ft. by 5 ft. (3.66 m by 1.42 m) cage with 20,000 lb. (9,070 kg) capacity
- Diesel power for tramming and drilling
- Mechanical rod lock and breakout wrench
- Side shift 26 in. (66 cm)
- Four hydraulic levelling jacks
- Carrier and mast stingers
- Positive feed stability with patented gimble cylinder mounting and cam roller slide system
- Load sensing hydraulic system
- Electric hydraulic control capability
- On-board screw booster compressor
- High torque head

- Level to level mobility without ramp access
- Tramming and operating capabilities independent of electricity source
- Operator safety and convenience
- More accurate and effective collaring and hole layout
- Improved set-up accuracy and hole accuracy
- Increased drilling stability and hole accuracy
- Smoother drilling extends component life and improves hole accuracy
- Optimum power distribution and remote drilling
- Automation capability
- Self-contained
- Larger, longer hole capability

General Specifications

Drilling

Hole diameter	3.5 in. (89 mm) to 8.5 in. (216 mm) (ask about larger hole and cluster drill capabilities)
Rod diameter	2.88 in. (73 mm) to 6.5 in. (165 mm)
Rod length	6 ft. (1.83 m) standard, 5 ft. (1.52 m) and 4 ft. (1.22 m) optional
Hole length	750 ft. (229 m) (ask about longer hole capabilities)

Feed

Gimbal mounted compound feed cylinder

Feed force	18,000 lb. (8,181 kg), optional 30,000 lb. (13,636 kg)

Drive Head

Direct drive hydraulic motor

Speed	0-39 rpm
Torque	5400 ft.-lb. (7,321 Nm), optional 10,000 ft.-lb. (13,557 Nm) hi-torque head

Mast Position

Horizontal tram

Tilt	15° forward (45° optional)
Rotation	360°
Side Shift	26 in. (660 mm) total, 13 in. (330 mm) offset from centre

Carrier

Crawler

Drive	hydraulic motor c/w motion control valves
Brakes	spring applied/hydraulic released on each track
Towing	brake disengage on torque hubs

Hydraulics

Reservoir	21 US gal. (80 l)
Pump	variable displacement piston pump c/w load sense and horsepower control
Filters	suction strainer, 10 micron pressure filter, 10 micron return filter

Dimensions

Tramming	length 12 ft. (3.66 m), width 56 in. (142.2 cm) or optional 48 in. (121.9 cm)
Dry weight	standard 20,500 lb. (9320 kg) with mechanical rod handling 24,000 lb. (10,910 kg) with diesel motor 20,500 lb. (9320 kg)

Air/Water System

Lube Oil	hydraulic driven rock drill oil injection pump c/w sight gauge and volume adjustment
Water	air driven water pump, 4 US gpm (15 L/min.) maximum volume adjustable. Optional electric driven piston pump: 4 US gpm (15 L/min.), 8 US gpm (30 L/min.), or 12 US gpm (45 L/min.)

Electrical

Power	30 hp (23 kw)/3 phase/575V/60Hz standard (ask about larger power options)
Controls	all electrical panels NEMA 4 rated (stainless steel optional), ground fault duplex receptacle

Controls

Hydraulic pilot drill and wrenching
Electric pilot set-up
Hydraulic gauges for feed, rotation, air and pilot pressure
Separate tram console
Optional electric pilot controls
Remote control
Electric umbilical
Radio remote

Options

10 in.² feed cylinder
High-torque head
Diesel power pack
Narrower unit, 4 ft. (1.2 m) wide (electric and diesel)
Single rod mechanical handler
20 rod handler for 3.5 in. (89 mm) to 4 in. (102 mm) rods
14 rod handler for 5.25 in. (133 mm) rods
On board screw air pressure booster to 350 psi (24 bar)

Contact an MTI representative or call our head office to discuss your specific requirements.

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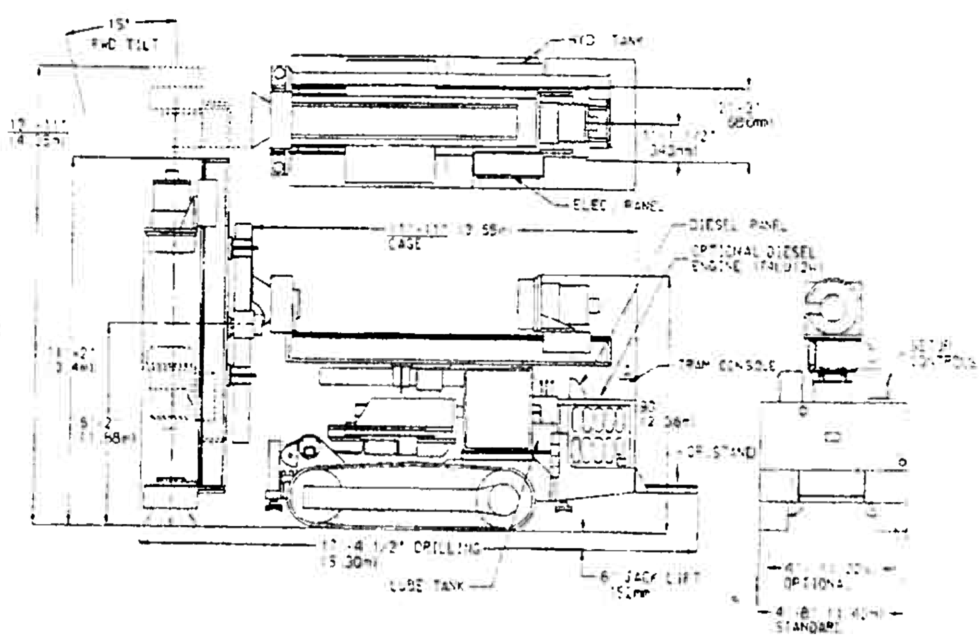

TECHPRO MINING PRODUCTS LIMITED

November 1, 1996

LOUIS BERNARD Fax: (905)-849-9622

Ref: T96-772-B

Dear Louis,

We are pleased to submit the following budget prices for your consideration:

One (1) only Return Water Storage Tank 21' 6 1/2" diameter x 18' high, nominal capacity 40,000 gallons

Price $14,700.00

One (1) only Fresh Water Storage Tank 18' 5 1/2" diameter x 16' high, nominal capacity 26,650 gallons

Price $13,550.00

Funds:	U.S. Dollars
Pricing:	Ex Works
Taxes:	Extra if Applicable
Duty:	Extra
Delivery:	TBA
Terms:	TBA

STEEL THICKNESS IN TANKS = 12 GAUGE
BOLT-TOGETHER SECTIONS

Unfortunately we do not get into transformers.

Should you have any questions please feel free to contact us.

Yours truly,

TECHPRO MINING PRODUCTS LIMITED

Brian Burrows
Vice President & General Manager

2485 SPEERS ROAD, OAKVILLE, ONTARIO, CANADA L6L 2X9
PHONE (905) 847-6620 FAX (905) 847-9052

10/28/96 11:52 ☎9058479052 001/002

TECHPRO MINING PRODUCTS LIMITED

October 28, 1996

LOUIS BERNARD

Fax: (905) 849-9622

Ref.: T96-772-A

Dear Louis:

Further to your fax of October 22, 1996 we are pleased to submit the following budget proposal for your consideration:

Qty.	Items		Unit Price
1	2000 ton coarse ore bin		No quote
1	Allis 4000 MF cone crusher (reconditioned)	150 HP	160,000.00
1	3 x 6m screen	5 HP	45,000.00
5	.6m x 110m Conveyor	20 Hp	130,000.00
1	2000 ton fine ore bin		No quote
1	10 - 20 TPH weightometer		9,000.00
1	3 x 4.25m ball mill (reconditioned)	900 HP	225,000.00
2	250mm cyclones		1,975 00
12	1.5 x 1.25 SRL pumps	2 HP	4,500.00
6	5 x 5 SRL pumps	15 HP	7,600.00
3	2" vertical sump pump	10 HP	8,500.00
3	2.5m x 2.5m conditioner		12,000.00
1	3 Hp agitator		2,500.00
56	DR 100 flotation cells	25 HP	10,500.00
14	18SP flotation cells	10 HP	7,500.00
1	15 Hp vacuum pump		7,500.00
2	5 Hp blowers		5,000.00
2	7m ϕ thickeners (reconditioned)	1 HP	27,000.00
1	30m ϕ thickener (reconditioned)	3 HP	90,000.00
3	6' x 8' drum filters	3 HP	75,000.00

2485 SPEERS ROAD, OAKVILLE, ONTARIO, CANADA L6L 2X9
PHONE (905) 847-6620 FAX (905) 847-9052

2	4m x 4m conc. hold tanks		8,500.00
2	5 Hp agitators		3,500.00
3	.6m x 20m conveyors	5 HP	25,000.00
10	1 Hp samplers		7,200.00
10	clarkson reagent feeders	1/12 HP	2,700.00
1	6m x 6m water tank		
1	5m x 5m water tank		
3	3 Hp reagent mixers		2,500.00
1	100 ton ore bin (3m x 5m)		No quote
1	1 x 4m grizzly feeder	40 HP	22,000.00
2	24" x 36" jaw crushers	100 HP	75,000.00
1	24" x 620m Conveyor		650,000.00
1	60 TPD Dryer (reconditioned)	10 HP	60,000.00
1	10,000 CFM dust collector (reconditioned)	5 HP	70,000.00
1	2 TPH lime slaking system	combined - 40 HP	100,000.00

Commercial Terms:

Funds: Canadian Dollars
Pricing: Ex-works
Taxes: Extra if applicable
Duty: Extra if applicable
Delivery: TBA
Terms: TBA

Note: All items quoted are new unless otherwise stated.

We trust this quotation is adequate for your present requirements. However, should you have any questions, please do not hesitate to contact us.

Yours very truly,

TECHPRO MINING PRODUCTS LIMITED

Brian A. Burrows
Vice-President & General Manager

TECHPRO MINING PRODUCTS LIMITED

October 23, 1996

LOUIS BERNARD Fax: (905) 849-9622

Ref.: T96-772

Dear Louis:

Further to your fax of October 22, 1996 we are pleased to submit the following budget proposal for your consideration:

Qty.	Items	Unit Price
1	2000 ton coarse ore bin	No quote
1	Allis 4000 MF cone crusher	
1	3 x 6m screen	45,000.00
5	6m x 110m Conveyor	130,000.00
1	2000 ton fine ore bin	No quote
1	40 - 50 TPH weightometer	9,000.00
1	3 x 4.25m ball mill (reconditioned)	225,000.00
2	250mm cyclones	1,975.00
12	1.5 x 1.25 SRL pumps	4,500.00
6	5 x 5 SRL pumps	7,600.00
3	2" vertical sump pump	8,500.00
3	2.5m x 2.5m conditioner	12,000.00
1	3 Hp agitator	2,500.00
56	DR 100 flotation cells	10,500.00
14	18SP flotation cells	7,500.00
1	15 Hp vacuum pump	7,500.00
2	5 Hp blowers	5,000.00
2	7m ϕ thickeners (reconditioned)	27,000.00
1	30m ϕ thickener (reconditioned)	90,000.00
3	drum filters	size??

2485 SPEERS ROAD, OAKVILLE, ONTARIO, CANADA L6L 2X9
PHONE (905) 847-6620 FAX (905) 847-9052

2	4m x 4m conc. hold tanks	8,500.00
2	5 Hp agitators	3,500.00
3	.6m x 20m conveyors	25,000.00
10	1 Hp samplers	7,200.00
10	clarkson reagent feeders	2,700.00
1	6m x 6m water tank	
1	5m x 5m water tank	
3	3 Hp reagent mixers	2,500.00
1	100 ton ore bin (3m x 5m)	No quote
1	1 x 4m grizzly feeder	22,000.00
2	24" x 36" jaw crushers	75,000.00

Commercial Terms:

Funds: Canadian Dollars
Pricing: Ex-works
Taxes: Extra if applicable
Duty: Extra if applicable
Delivery: TBA
Terms: TBA

Note: All items quoted are new unless otherwise stated.

We trust this quotation is adequate for your present requirements. However, should you have any questions, please do not hesitate to contact us.

Yours very truly,

TECHPRO MINING PRODUCTS LIMITED

Brian A. Burrows
Vice-Presicent & General Manager

BAB/cem

SVEDALA

November 15, 1996

Fax No. (905) 849-9622

Total Pages 3

Louis M. Bernard
Senior Mining Consultant
331 Maplehurst Avenue
Oakville, Ontario
L6L 4Y3

Subject: Ball Mill
Pre-Feasibility Inquiry
Grong Project, Norway
Our Ref: B96-066

Gentlemen,

In response to your fax request of October 18, 1996, we are pleased to submit for your consideration our budget proposal based on your preliminary information for the following equipment:

1. Ball Mill

We have selected the Ball Mill size based on your horsepower requirement of 1,000 HP. A preliminary calculation based on the data provided, indicates that the grinding power required is approximately 800 HP. This assumes a throughput of 48 MTPH, a feed size of 80% passing 12,000 micrometers, a product size of 80% passing 120 micrometers and a Ball Mill work Index of 12 kWhr/ST. The size of this Ball Mill will have to be confirmed with grindability tests on the ore.

Item 1.0 One (1) only 11' x 17' F/F (16.5' EGL) Overflow Ball Mill arranged for 1000 HP/1000 RPM induction motor complete with fabricated steel shell, cast iron heads, steel gear and forged alloy steel pinion, feed spout, feed and discharge trunnion liners, trash trommel, trunnion bearings, main bearing lubrication system, trunnion bearing, trunnion bearing and pinion bearing soleplates and automatic gearspray system. Speed reducer and high speed coupling and air clutch are also included.

Not including: Mill Motor and Controls
Liners and Liner Hardware
Initial Ball Charge
Installation and Field Assembly
Foundation & Foundation Bolts
Interconnecting Piping

Approximate Shipping Weight:..144,056 lbs/lot

Budget Price @ Point of Manufacture:..............................$CAN 848,100/lot

Svedala Industries Canada Inc.
16777 Hymus, Kirkland, Québec, Canada H9H 3L4
Tel: (514) 694-9400 Fax: (514) 694-9413

SVEDALA

Subject: Ball Mill
Pre-Feasibility Inquiry
Grong Project, Norway
Our Ref: B96-066

Item 1.1 One (1) set of Cr-Moly steel shell, feed and discharge end liners c/w filler bars and necessary hardware for item 1.0.

Liner Weight:..95,300 lbs/set

Budget Price @ Point of Manufacture:..............................$CAN 175,360/set

Item 1.1A One (1) set of rubber shell, feed and discharge end liners c/w filler bars and necessary hardware for item 1.0.

Liner Weight:..24,600 lbs/set

Budget Price @ Point of Manufacture:..............................$CAN 82,105/set

Item 1.2 One (1) 1000 HP/1000 RPM squirrel cage induction motor (3.3KV/3PH/50HZ); TEAAC enclosure; 1.0 s.f.; for Ball Mill item 1.0. Motor starters and controls are not included.

Approximate Shipping Weight:.......................................10,200 lbs/lot

Budget Price @ Point of Manufacture:..............................$CAN 135,030/lot

Note:

1. Above prices are in Canadian funds; today's budget; exclusive of custom duty and all taxes.
2. Prices are at point of manufacture, freight charges are extra.
3. Mill sizes quoted are Svedala Standard Mill sizes closest to your requirement.
4. The Grinding Mills offered will be supplied all in accordance with Svedala's standard for this class of equipment.
5. For export, allow 10% to cover the cost of export preparation and freight to ports of export.

SVEDALA

Subject: Ball Mill
Pre-Feasibility Inquiry
Grong Project, Norway
Our Ref: B96-066

We trust that the above will meet with your current requirements. Please do not hesitate to contact us if you require additional information.

Yours very truly,

J. Gougeon
Product Engineer

JG/ah

cc:	B. See Hoye	(Svedala, Kirkland)
	M. Fu	(Svedala, Kirkland)
	H. Holmes	(Svedala, Derby)
	S. Crawford	(Svedala, Mississauga)
	G. Langlands	(Svedala, Uxbridge)

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SVEDALA INDUSTRIES CANADA
PUMPS AND PROCESS EQUIPMENT

3136 MAVIS ROAD, MISSISSAUGA, ON L5C 1T8, CANADA
TELEFAX No.: (905) 270-9996 TELEPHONE No.: (905) 270-2170

Fax Cover Sheet

PAGE 1 OF 3

DATE: **October 23, 1996**

REF. No.:

COMPANY:

FAX No.: **849-9622**

ATTN.: **Louis Bernard**

COPY TO:

FROM: **Scott Crawford**

SUBJECT: **Budget List for "Grong" Project, Norway**
Svedala Pumps & Process Canada Budget Quotation No. 96/6429

Dear Mr. Bernard:

We are pleased to provide budget pricing as follows based on your equipment list:

Item	Qty.	Description	Power (HP)	Estimated Weight (kg)	Budget Price (CDN$/Each)
1	12	Process Pumps, SRL 1.5 x 1.25 x 9	3	450	$4,500.00
2	6	Process Pumps, SRL 5 x 5 x 14	30	800	$9,600.00
3	3	Process Pumps, VASA G 2"	3	500	$4,700.00
4	3	Conditioner, 2.5m x 2.5m (tank by others)	15	300	$11,000.00
5	1	Agitator (tank by others)	7.5	600	$13,700.00
6	56	Flotation Cells, DR100	840	26,500	$1,368,000.00
7	14	Flotation Cells, 18 SP DR	70	6,500	$187,000.00
8	2	Concentrate Thickeners, 7m (tank by others)	5	8,200	$60,000.00
9	1	Concentrate Thickener, 30m (tank by others)	7.5	27,000	$295,000.00
10	2	Conc. Tank Agitator (tank by others)	7.5	600	$13,700.00

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Page 2. October 23, 1996
"Grong" Project, Norway
Svedala Pumps & Process Canada Budget Quotation No. 96/6429

VALIDITY

The above quoted prices are for budgetary purposes only and are FOB Mississauga, Ontario. Sales taxes, if applicable, are not included in the prices.

Should you require any further information, please do not hesitate to contact us.

Yours truly,

Scott Crawford

Scott Crawford
Application Engineer

Encl. Conditions of Sale

SVEDALA

CONDITIONS OF SALE

1. WARRANTY — There are no warranties or representations by Company with respect to the Product(s) or affecting the rights of the parties other than those specifically set out hereafter. No other warranty, agreement or representation made hereto, nor any modification hereof, shall be binding upon Company (or its assigns), unless endorsed hereon in writing. The present conventional warranty is agreed to as the only warranty of Company with respect to the Product(s), legal and implied warranties and conditions being specifically excluded and waived by Purchaser.

   (a) Company warrants title to the Product(s) and, except as noted below, with respect to items not of Company's manufacture, also warrants the Product(s) on date of shipment to Purchaser to be of the kind and quality described herein, merchantable, and free of defects in workmanship and material;

   (b) Any item of the Product(s) which is not manufactured by Company is not warranted by Company. Any warranty which is due to Company by the manufacturer in question is hereby transferred to Purchaser, without recourse against Company;

   (c) If within one year from date of initial operation, but not more than eighteen months from date of shipment by Company of any item of Product(s), any such item is discovered by Purchaser not to be of the kind and quality described herein, or to be defective in workmanship or material, and prompt written notification is made in by (at Company's option) adjustment or repair or replacement of item or of any affected part of the Product(s). Purchaser shall assume all responsibility and expense for removal, reinstallation and freight in connection with the foregoing. The foregoing conditions shall extend to replacement parts furnished by Company hereunder. Company shall have the right of disposal of parts replaced by it;

   (d) Company does not warrant that any item of Product(s) is fit for any particular purpose, nor does it warrant design. Likewise, Company and its suppliers shall have no obligation as to any item which has been improperly stored or handled, or which has not been operated or maintained according to good practice or according to instructions in any manuals, nor shall they be liable for the fault, negligence, want of skill, or wrongful acts of Purchaser, of its employees, or of other contractors or suppliers of Puchaser;

   (e) The limit of Company's warranty being stated as above, Purchaser waives any claim against Company or its suppliers, whether in contract or in tort or under any other legal theory, and whether arising out of warranties, representations, conditions, or defects from any cause for loss of use, revenue of profit, as well as for any incidental or consequential losses or damages, and for claims for damages of Purchaser's customers.

2. PATENTS — Company shall pay costs and damages finally awarded in any suit against Purchaser or its vendees to the extent based upon a finding that the design or construction of the product(s) as furnished infringes a Canadian patent (except infringement occurring as a result of incorporating a design or modification at Purchaser's request) provided that Purchaser promptly notifies Company of any charge of such infringement, and Company is given the right at its expense to settle such charge and to defend or control the defense of any suit based upon such charge. This paragraph sets forth Company's exclusive liability with respect to patents.

3. PURCHASER DATA — Timely performance by Company is contingent upon Purchaser's supplying to Company when needed, all required technical information, including drawing approval, and all required commercial documentation.

4. NUCLEAR — Purchaser represents and warrants that the product(s) covered by this contract shall not be used in or in connection with a nuclear facility or application.

5. NONCANCELLATION — Purchaser may not cancel or terminate for convenience, or direct suspension of manufacture, except on mutually acceptable terms.

6. DELAYS — If Company suffers delay in performance due to any cause beyond its control, including but not limited to act of God, war, act or failure to act of government, act or omission of Purchaser, fire, flood, strike or labor trouble, sabotage, or delay in obtaining from others suitable services, materials, components, equipment of transportation, the time of performance shall be extended a period of time equal to the period of the delay and its consequences. Company will give to Purchaser notice in writing with a reasonable time after Company becomes aware of such delay.

7. STORAGE — Any item of the product(s) on which manufacture or shipment is delayed by causes within Purchaser's control, or by causes which affect Purchaser's ability to receive the product(s), may be placed in storage by Company for Purchaser's account and risk.

8. SHIPMENT — The term "shipment" means delivery to the initial carrier in accordance with the delivery terms of this order. The shipping date is based upon conditions at the factory on the date hereof, and is subject to revision to meet conditions on date of Purchaser's acceptance. Company may make partial shipments. Company shall select method of transportation and route, unless terms are f.o.b. point of shipment and Purchaser specifies the method and route and is to pay the freight costs in addition to the price. When terms are f.o.b. destination or freight allowed to destination, "destination" means common carrier delivery point (within Canada) nearest the destination. For movement outside Canada, Company shall arrange for inland carriage to port of exit and shall cooperate with Purchaser's agents in making necessary arrangements for overseas carriage and preparing necessary documents.

9. SPECIAL SHIPPING DEVICES — On shipments to a destination in Canada, Company has the right to add to the invoice, as a separate item, the value of any special shipping device (oil barrel, reel, tarpaulin, cradle, crib and the like) used to contain or protect the product(s) invoiced, while in transit. Except as to oil barrels, full credit will be given on the return to Company of the device in a reusable condition, f.o.b. destination, freight prepaid. As to oil barrels, arrangements for return and credit must be made by Purchaser with the refiner.

10. TERMS OF PAYMENT — Unless otherwise stated, all payments shall be in Canadian dollars, and a pro rata payment shall become due as each readiness for shipment shall be deemed to be date of shipment for payment purposes.

    On late payment, the contract price shall, without prejudice to the Company's right to immediate payment, be increased by 1-1/2% per month (18% annual percentage rate) on the unpaid balance due and owing, but not to exceed the maximum permitted by law. Interest shall accrue and be calculated to each statement date from the due date of payment.

    If at any time in Company's judgment Purchaser may be or may become unable or unwilling to meet the terms specified, Company may require satisfactory assurances or full or partial payment as a condition to commencing or continuing manufacture or making shipment; and may, if shipment has been made, recover the product(s) from the carrier, pending receipt of such assurances.

11. TITLE AND INSURANCE — Title to the product(s) and risk of loss or damage shall pass to Purchaser at the f.o.b. point, except that a security interest in the product(s) and proceeds and any replacement shall remain in Company, regardless of mode of attachment to realty or other property, until the full price has been paid in cash. Purchaser agrees to do all acts necessary to perfect and maintain said security interest and to protect Company's interest by adequately insuring the product(s) against loss or damage from any external cause with Company named as insured or co-insured.

12. QUALITY ASSURANCE - This equipment is a standard product subject only to Sala Machine Works. Quality Assurance procedures. No additional Purchaser quality requirements will apply unless agreed upon.

13. CONTAINERS — The value of each special shipping device (oil barrel, reel, tarpaulin, cradle, crib and the like) used by the Company to contain or protect the product(s) in shipment will be invoiced to Purchaser as a separately stated addition to the contract price.

    If the Company's proposal or quotation or other contract documents stipulate the return of any such device, it shall be returned by the Purchaser in good condition for credit f.o.b. Purchaser's plant freight collect, within thirty (30) days after receipt by Purchaser.

    Return of any such device as to which there is no stipulation but which has been separately invoiced is at the option of the Purchaser. If returned promptly in usable condition, f.o.b. destination, freight prepaid, Company will grant Purchaser a credit in the invoiced amount (except oil barrels as to which arrangements for return and refund must be made by the Purchaser with the refiner).

    The foregoing provisions as to special shipping devices shall not apply to any such device shipped outside continental Canada and the United States.

14. TAXES — Any applicable Sales, Use, Excise or similar taxes will be added to quoted price and invoiced separately (unless acceptable exemption certificate is furnished).

15. GENERAL — Purchaser hereby waives any claim against Company or its suppliers (whether contractual or delictual) for loss of use, revenue or profit, as well as for any incidental or consequential losses or damages, and for claims for damages of Purchaser's customers.

    Company will comply with all laws applicable to Company. Compliance with federal, provincial, or local laws during any operation or use of the product(s) is the sole responsibility of Purchaser.

    The laws of the Province of Ontario shall govern the validity, interpretation and enforcement of any contract of which these provisions are a part.

    Assignment may be made only with written consent of both parties.

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SVEDALA INDUSTRIES CANADA
PUMPS AND PROCESS EQUIPMENT

3136 MAVIS ROAD, MISSISSAUGA, ON L5C 1T8, CANADA
TELEFAX No.: (905) 270-9996 TELEPHONE No.: (905) 270-2170

Fax Cover Sheet

PAGE 1 OF 2

DATE: **October 28, 1996**

REF. No.:

COMPANY:

FAX No.: **849-9622**

ATTN.: **Louis Bernard**

COPY TO:

FROM: **Scott Crawford**

SUBJECT: **Budget List for Grong Project**
Correction to Our Budget Quotation No. 96/6429

Dear Mr. Bernard:

I realized I made a mistake in the quotation I sent you last week. The budget prices for the flotation cells are the "lot" prices for the machines, not "each" prices as the heading for the price column indicates. Also, the estimated weights should be as follows: 56x100DR: 106,000 kg and 14x18 SP DR: 13,000 kg.

Attached please find the corrected first page to the quotation.

Yours truly,

Scott Crawford
Application Engineer

SVEDALA

SVEDALA INDUSTRIES CANADA
PUMPS AND PROCESS EQUIPMENT

3136 MAVIS ROAD, MISSISSAUGA, ON L5C 1T8, CANADA
TELEFAX No.: (905) 270-9996 TELEPHONE No.: (905) 270-2170

Fax Cover Sheet

PAGE 1 OF 3

DATE: **October 23, 1996**

REF. No.:

COMPANY:

FAX No.: **849-9622**

ATTN.: **Louis Bernard**

COPY TO:

FROM: **Scott Crawford**

SUBJECT: **Budget List for "Grong" Project, Norway**
Svedala Pumps & Process Canada Budget Quotation No. 96/6429

Dear Mr. Bernard:

We are pleased to provide budget pricing as follows based on your equipment list:

Item	Qty.	Description	Power (HP)	Estimated Weight (kg)	Budget Price (CDN$/Each)
1	12	Process Pumps, SRL 1.5 x 1.25 x 9	3	450	$4,500.00
2	6	Process Pumps, SRL 5 x 5 x 14	30	800	$9,600.00
3	3	Process Pumps, VASA G 2"	3	500	$4,700.00
4	3	Conditioner, 2.5m x 2.5m (tank by others)	15	300	$11,000.00
5	1	Agitator (tank by others)	7.5	600	$13,700.00
6	1	56 Flotation Cells, DR100	840	106,000	$1,368,000.00
7	1	14 Flotation Cells, 18 SP DR	70	13,000	$187,000.00
8	2	Concentrate Thickeners, 7m (tank by others)	5	8,200	$60,000.00
9	1	Concentrate Thickener, 30m (tank by others)	7.5	27,000	$295,000.00
10	2	Conc. Tank Agitator (tank by others)	7.5	600	$13,700.00

Cont'd.../2

technequip limited

VIA FAX

297 Garyray Drive
WESTON, Ontario, Canada, M9L 1P2
Tel. 416-749-3991 Fax 416-749-9767
SUBSIDIARY OF FULLER-TRAYLOR INC.

To: **Mr. Louis Bernard**
Fax: 905-849-9622
From: **Campbell McClure**
Date: October 24, 1996
Pages: 4 page(s) including this page.

Re: Grong Copper/Zinc Project - Norway

Louis;

You have asked for a relatively coarse grind. The high S.G. of this ore demands the use of a large cyclone to produce the desired overflow product. Unfortunately, we are limited by the tonnage. As long as the customer can operate with a feed of 64% solids by wt., I would suggest using 1 20" cyclone equipped with a 5.25" vortex finder and 2.75" apex @8-10 PSI to produce the 250% C.L. at the 55%-325 mesh grind. Please find attached a mass balance outlining the operating conditions.

I am pleased to submit a budget quotation as follows;

1 only model H209C-AM8 cyclone as illustrated on the enclosed drawing #LCAARH01 consisting of;

- 5.25" Nihard vortex finder
- 2.75" fixed ceramic apex.
- 1 operating 20" cyclone @ 8-10 PSI
- all slurry handling surfaces are lined with dense molded pure gum rubber. The feed inlet and overflow adapter will both be victaulic. The feed inlet adapter will be lined with ¼" hot vulcanized pure gum rubber. The overflow adapter will be unlined.
- all exterior surfaces to be painted with 1 coat of Alkyd based primer and 1 coat of high gloss yellow Alkyd machinery enamel.
- The cyclone will be factory assembled and prepared for shipment.

Cyclone as described above:

Unit Price $ 4770.00 CDN

Price is;
FOB Plant Toronto, ON
Canadian funds
All taxes extra
Terms net 30 days
Delivery 4-6 weeks ARO and approved drawings.

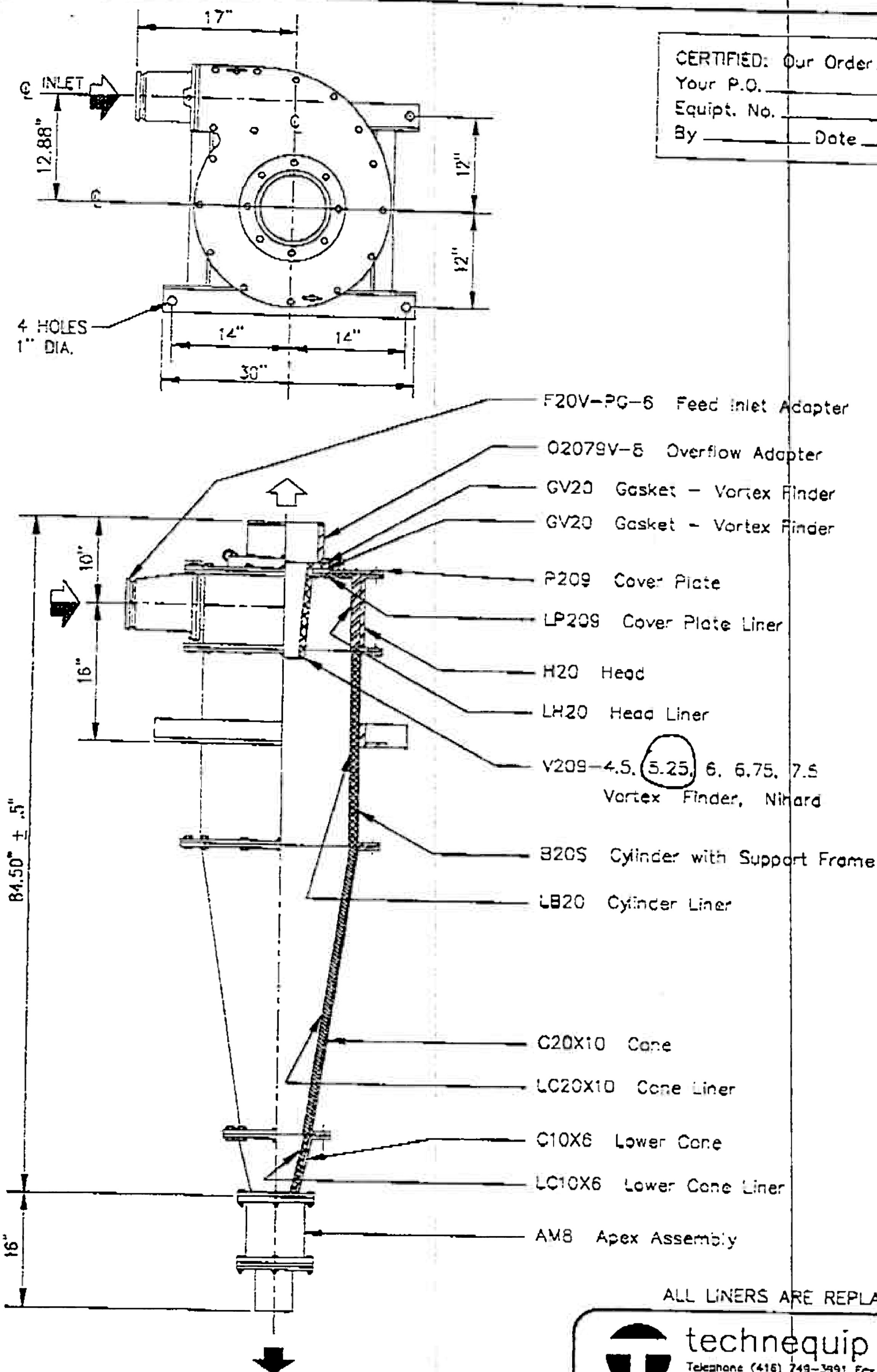
The attached drawing is for illustrative purposes only. If you have any questions, please don't hesitate to give me a call.

Best regards,

C. McClure

Campbell McClure

CERTIFIED: Our Order
Your P.O.
Equipt. No.
By Date



ALL LINERS ARE REPLACEABLE

technequip limited
Telephone (416) 749-3991 Fax (416) 749-9767
297 Garyray Drive, Weston, Ont., Canada M9L 1P2

DESCRIPTION:
HYDROTECH CYCLONE
MODEL H209C-AM8

DRWG NO: LCAARH01 | DATE AUG 24/95 | A

technequip limited

CYCLONE MASS BALANCE

DATE: OCT. 24/'96
FOR: GRONG
PROJECT: COPPER/ZINC PROJECT
APPLICATION: 55% – 325 MESH
1 OP'G H209C–AM8 CYCLONE @ 8–10 PSI

	WEIGHT			VOLUME		
	STPH	MTPH	S.G.	USGPM	M^3/HR	LPS
OVERFLOW						
Solids	47.3	42.9	4.00	47.3	10.7	3.0
Liquid	53.8	48.8	1.00	215.0	48.8	13.5
Slurry	101.0	91.7	1.54	262.3	59.5	16.5

Overflow % Sol./Wt. ***46.8%*** *Overflow % Sol./Vol.* ***18.0%***

CIRCULATING LOAD **250.0%**

	STPH	MTPH	S.G.	USGPM	M^3/HR	LPS
UNDERFLOW						
Solids	118.2	107.3	4.00	118.2	26.8	7.4
Liquid	39.4	35.7	1.00	157.6	35.7	9.9
Slurry	157.6	143.0	2.29	275.8	62.6	17.4

Underflow % Sol./Wt. ***75.0%*** *Underflow % Sol./Vol.* ***42.9%***

	STPH	MTPH	S.G.	USGPM	M^3/HR	LPS
FEED						
Solids	165.5	150.2	4.00	165.5	37.5	10.4
Liquid	93.2	84.5	1.00	372.6	84.5	23.5
Slurry	258.7	234.7	1.92	538.1	122.0	33.9

Feed % Sol./Wt. ***64.0%*** *Feed % Sol./Vol.* ***30.8%***

297 CARYRAY DRIVE, WESTON, ONTARIO, CANADA, M9L 1P2 • TELEPHONE 416-749-3991 • FAX 416-749-9767
SUBSIDIARY OF FULLER-TRAYLOR INC.

SYNTEC
PROCESS EQUIPMENT LTD.
HEALEY ROAD, UNIT 1,
BOLTON, ONTARIO. L7E 5A4
TEL: (905) 951-8000
FAX: (905) 951-8002

QUOTATION

SYNTEC
VALVES
CONTROLS
INSTRUMENTATION

TO: Lou Bernard Engineering

S/Q-

DATE: 10/24/96

ATTENTION: Lou Bernard

SALES REP: Steve Stoll

WE ARE PLEASED TO SUBMIT THE FOLLOWING

REFERENCE: Feasibility - Europe

ITEM	QTY	DESCRIPTION	UNIT PRICE	TOTAL PRICE
A	10	E-188-201-115 Clarkson Reagent Feeder, 115/50Hz Voltage 20 Large Cup (100cc/min each), 304SS Construction (Max Flow Rate 32gal/hr.)	1562.00 ea	15,620.00
		For 20 Small cup Deduct ... (Max Flow Rate 3.2 gal/hr.)	61.00 ea	
		Adder for Variable Speed Drive Motor Includes 90 VDC Motor with manual control add ...	967.00 x5	4835.00

THIS QUOTATION FIRM FOR 30 DAYS

TERMS: (O.A.C.) Net 30 Days

F.O.B. Bolton

DELIVERY: 6 weeks

SALES TAXES: All Extra

IF APPLICABLE: ACCEPTANCE AND APPROVAL: IT IS AGREED THAT ANY ORDER RESULTING FROM THIS PROPOSAL SHALL BE IN ACCORD WITH SYNTEC PROCESS EQUIPMENT LTD.'S STANDARD TERMS AND CONDITIONS.

WE APPRECIATE THIS OPPORTUNITY TO QUOTE

BY: Steve Stoll

October 31, 1996

Norman Wade Company Limited
10 Brockley Drive
Hamilton, Ontario
L8E 3P1
Tel: (905) 561-9195
Fax: (905) 561-5979

Mr. Louis Bernard
331 Maplehurst Ave.
Oakville, Ontario
L6L 3Y3

Dear Mr. Louis Bernard:

We would like to thank you for considering **Norman Wade Company Limited**, your potential supplier of Total Station technology. According to your request, I have included for your review a quotation regarding our **Topcon GTS-210** total station system.

TOTAL STATION

Catalogue #	Description	Price
512 8671100	**GTS-211D** Total Station Includes: 5 second total station, 2 batteries, battery charger, tool kit, carrying case, silicon cloth, instrument cover, instrument manual, & carry straps.	$12,325.00
512 8671200	**GTS-212** Total Station Includes: 6 second total station, 2 batteries, battery charger, tool kit, carrying case, silicon cloth, instrument cover, instrument manual, & carry straps.	$10,465.00

DATA COLLECTOR

512 8492000	**FC-48GX** Data Collector Pkg.	$2,130.00

Includes the following

512 8488200	Cable For HP48 ENV CASE & FS-2	Inc.
512 8488600	480 ENVIR CABLE PC FS2 9-25P	Inc.
512 8490100	HP48 ENVIRONMENTAL CASE	Inc.

ACCESSORIES

Catalogue #	Description	Price
512 8615300	SINGLE TILT PRISM ASSEMBLY	$295.00
512 8609200	8FT/2.5M PRISM POLE(TILTING)	$216.00
513 8306000	WADE FIBERGLASS WOOD TRIPOD	$300.00

We will be in contact with you shortly to discuss our proposal. Should you have any immediate questions or concerns, please call us directly at (905) 561-9195.

Yours sincerely,

NORMAN WADE COMPANY LIMITED

Stephen Dawe
Survey Sales Consultant, Survey Division

FAX

Date: October 31, 1996 **Time:** 2:53:21 PM 3 **Pages**

To: MR. LOUIS BERNARD

Fax: 905 849-9622
Voice:

From: STEPHEN DAWE
NORMAN WADE COMPANY LIMITED

Fax: (416) 291-4219
Voice: (416) 291-4211

Comments:

Good Afternoon:

Please find enclosed a quotation regarding our Topcon Total Station system.

Regards,

S. Dawe

KENT AIR PRODUCTS
CANADA LIMITED

1072 WEBBWOOD DRIVE, SUDBURY, ONTARIO P3C 3B7
Telephone: (705) 673-1212 Fax: (705) 673-9563

Rammer

October 31st, 1996

LOUIS M. BERNARD
331 Maplehurst Ave.
OAKVILLE, ON
L6L 4Y3

Dear Sir:

Further to our phone conversation, Kent Air Products are pleased to quote on the following Booms and Hammers:

1 ONLY

KENT 16' HYDRAULIC HEAVY DUTY BOOM complete with 12"x12"x8' tubular sections, heavy duty base, hinge supports and mounting plates - with support pedestal & operators console.

POWER UNIT (WITH ELECTRICAL HYDRAULIC VALVE) including 40 HP motor, 60 gallon tank, "Load Sensing" hydraulic pump, flow regulators, complete wit hoses etc...

1 ONLY - RAMMER HYDRAULIC BREAKER, Model S-26 complete with mounting brackets and a point.

.... $ 66,500.00

1 ONLY

KENT 16' HEAVY DUTY BOOM & POWER UNIT (AS PER SPECIFICATIONS ABOVE) complete with -

1 ONLY - RAMMER HYDRAULIC BREAKER, Model S-29 CITY complete with mounting brackets and a point.

.... $ 69,000.00

1 ONLY

KENT 16' HEAVY DUTY BOOM & POWER UNIT (AS PER SPECIFICATIONS ABOVE) complete with -

1 ONLY - RAMMER HYDRAULIC BREAKER, Model E-64 complete with mounting brackets and a point.

.... $ 72,500.00

. . . . /Continued

KENT AIR PRODUCTS CANADA LIMITED

1072 WEBBWOOD DRIVE, SUDBURY, ONTARIO P3C 3B7
Telephone: (705) 673-1212 Fax: (705) 673-9563

Rammer

1 ONLY — KENT 16' HEAVY DUTY BOOM & POWER UNIT (AS PER SPECIFICATION ABOVE) complete with -

1 ONLY - RAMMER HYDRAULIC BREAKER, Model E-64 CITY complete with mounting brackets and a point.

.... $ 76,000.00

1 ONLY — ROCKBREAKER CAB 48" x 48" x 6' high, complete with 5/8" tempered glass and new console cover (#2167)

.... $ 4,950.00

<u>OPTIONS FOR CAB</u>:

#2167-005 INSULATION & INTERIOR LINING .... $ 2,145.00

#2167-008 AIR INTAKE FILTER & EXHAUST FAN ..... $ 1,072.50

<u>ALL PRICES ARE</u>:

F.O.B. : Toronto
DELIVERY: 5 to 6 Weeks
TERMS : Net 30 Days
TAXES : All Applicable Taxes Extra

I hope that this information is satisfactory! Please do not hesitate to contact our office with any questions or comments.

Kind Regards,
KENT AIR PRODUCTS CANADA LTD.

Robert J. Allen,
District Manager

RJA:kjk
Encls.