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Lökken Gruber.

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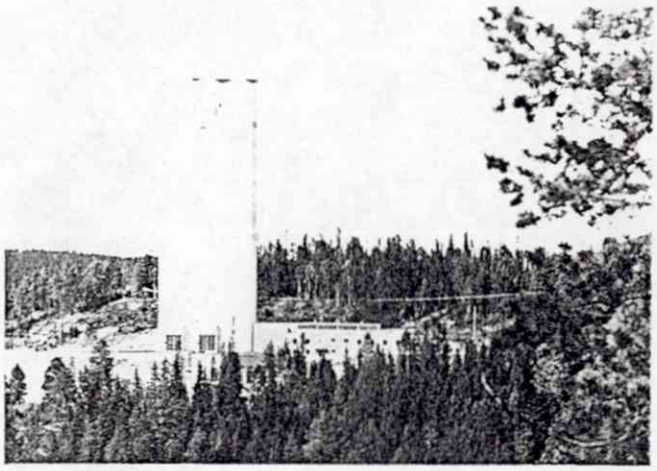
Sammendrag

Artikkel fra Mining Magazine, April 1985, med grundig presentasjon av Lökken Gruber, Meldal. Produksjon siden 1654, om lag 20 mill. t. er utdrevet. Forekomsten ligger i en invertert og folda bergartsenhet med metavulkanitter og yngre sediment. Massive, stratabundne sulfidmineraliseringer av vulkansk ekshalativ opprinnelse. Reserver pr. 1984 2-3 mill.t. malm med 2,5 % Cu og 2,1 % Zn. Skildring av transportsystem, ventilasjon, brytningsmetoder og oppredningsprosessen.

Løkken Gruber

By Kim Burridge*

Metalliferous ores have been mined in the Løkken district of Norway for over three hundred years. The present underground copper-zinc mining operations are fully mechanized and utilize up-to-date equipment and methods.



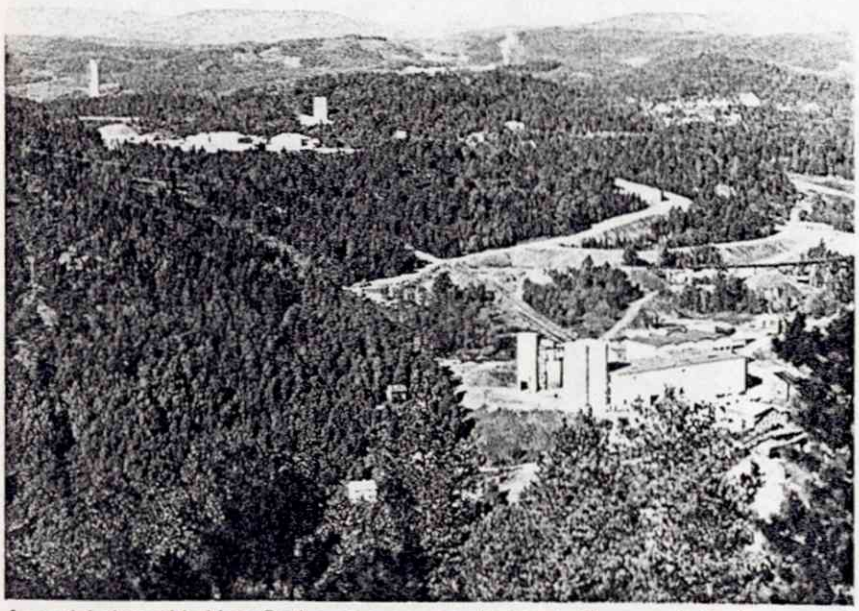
Astrup shaft headframe.

THE Løkken area of central Norway, situated 50 km southwest of Trondheim, has been mined since 1654, when the first copper-bearing pyrite was discovered. Located close to the village of Løkken Verk is the remnant of Norway's largest copper-zinc sulphide deposit, originally with about 25 Mt of reserves, from which 20 Mt have already been extracted. Today the only remaining operation in the district is the Løkken underground copper-zinc mine. This relatively small mine is worked by Løkken Gruber A/S & Co., which is jointly owned equally by Orkla Industrier A/S and Outokumpu Oy of Finland. The mine produces 21,000 t/y of copper concentrate and 5,400 t/y of zinc concentrate.

The present underground operations at the Løkken mine are based on the Astrup shaft, which was opened in 1972. Astrup was the third production shaft to be sunk at Løkken this century. In 1911 production centred on the Fearnley shaft, close to Løkken Verk village. However, by 1916 the Wallenburg shaft had been sunk to extract pyrites from a major new orebody discovered approximately 2 km to the west. During the 1920s the Wallenberg mine became one of the world's largest exporters of pyrites. Most of the output was transported on Norway's first electric railway which was built by the mine company to link Løkken Verk with Thamshavn, from which the ore was shipped.

Production switched to the Astrup shaft following exhaustion of reserves at Wallenberg. The latter now serves as a repository for mine drainage water from the present mining operation. The Astrup shaft with its ore silos, exploration office and workshops is located about 4 km west of the mine's processing plant and main administrative buildings at Løkken Verk village.

*Ms. Burridge, formerly Assistant Editor of Mining Magazine, is now Editor of Mine & Quarry.



An aerial view of Løkken Gruber. The process plant and offices are in the foreground. The Wallenberg shaft is in the middle distance and the Astrup shaft, serving current mining activities, lies beyond (top left).

Geology

The Løkken area lies within the Norwegian Caledonides and consists of an inverted and folded succession of metavolcanic rocks with younger sediments. The metavolcanics comprise two subgroups, an upper and lower group, which are mainly basic in composition. The lower subgroup is characterized by massive flows developed partly as pillow lavas. These occur stratigraphically above metagabbros which vary from coarse-grained dioritic varieties to fine grained gabbroic types.

The upper subgroup consists of generally thinner lava flows, often pillowed (greenstones), which are separated from each other by horizons of reddish jasper, black cherts with sulphides ("vasskis") or volcanic sediments.

The ore deposits occur in the lowest section of the upper subgroup stratigraphically below the black cherts, jaspers and volcanic sediments. Occurring as massive stratabound sulphide deposits they are considered to be of volcanic exhalative origin. The main deposits consist of several separate bodies which together form an elongated lens striking east-west over a total distance of 4 km. This mass, which has an average width of between 150-200 m and thickness of about 50 m, has been followed to a depth of 1,050 m. It is located on the southern limb of the Løkken synform and plunges gently to the west, dipping to the north.

Reserves

Present proven reserves at Løkken total 2-3 Mt grading on average 2.5% Cu and 2.1% Zn. These are only sufficient for a further 8-10 years mining. The present mine owners intend to continue mining at Løkken if possible and

are optimistic that new reserves will be located. The Norwegian Government is also anxious to maintain mining in the area which at present has one of the highest unemployment rates in Norway. Government aid for exploration is available and as the mine lies in one of Norway's specially designated "enterprise zones" incentives are also available to maintain the region's industry.

A three-year exploration programme is in progress, based around the immediate vicinity of the Astrup shaft, following the orebody to greater depths. A previous three-year programme exploring the areas surrounding the mine was carried out as a joint venture with Norwegian Gulf Exploration Co., a subsidiary of Gulf Oil. The NOK15 million programme (£1 = NOK10.5) began in 1980 and involved various airborne and ground geophysical surveys, geochemical surveys, and mapping. Many different anomalies were revealed but diamond drilling produced no significant results.

The company recently installed a Hewlett Packard 86B system for ore reserves calculation using software developed in the University of Trondheim.

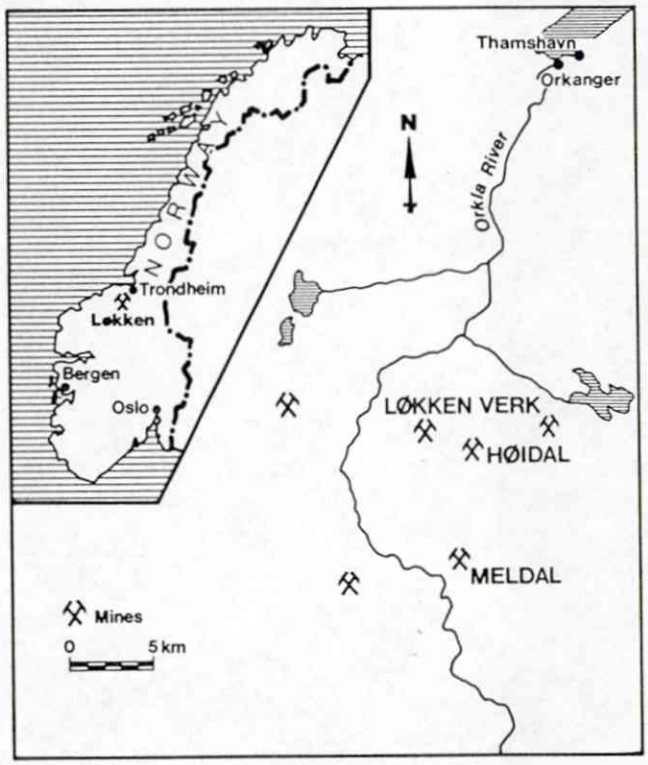


Fig. 1: Sketch map of Løkken district with (inset) a map of Norway showing Løkken's locations.

Shaft system

Access to the working area is by the 970 m deep Astrup shaft which has an elliptical cross section of 11 m^2. The shaft head, with ore and service hoist, 2,500 t ore silo and 1,400 t waste rock silo, are contained within a fully enclosed concrete tower next to the workshops and exploration office. An EPR 1,010 kW friction hoist with Siemens controls, used for ore hoisting and personnel, has a capacity to hoist 200 t/h of ore using a 15 t skip, and a 28-man cage. The 52 kW service hoist carries a 1 t skip and a cage for 5 persons.

Loading occurs at the 965 m level, and shaft stations are located at the 720, 750, 820, 930, 945 and 965 m levels. The primary crusher and silo is located on the 945 m level.

An inclined rampway connects levels 720 and 930 and is at present being extended downwards to the base of the known ore deposit.

The various levels connecting the Astrup shaft area with the Wallenburg shaft section of the mine were recently sealed to reduce inflows of mine drainage water from the old mined-out districts.

Ventilation

Total ventilation capacity is 85 m^3/s. The system consists of three shafts, with surface fan stations at the main Astrup shaft (No. I), the nearby shaft (Astrup II) and an exhaust station at the Moen shaft. All are fitted with Norsk Vifterfabrike fans: Astrup I has a 1.4m dia. 25 m^3/s unit; Astrup II has three 1.2 m dia. 20 m^3/s units, and Moen has two 1.4 m dia. 42.5 m^3/s units.

Mining methods

The Løkken deposit is worked by a combination of sub-level stoping with

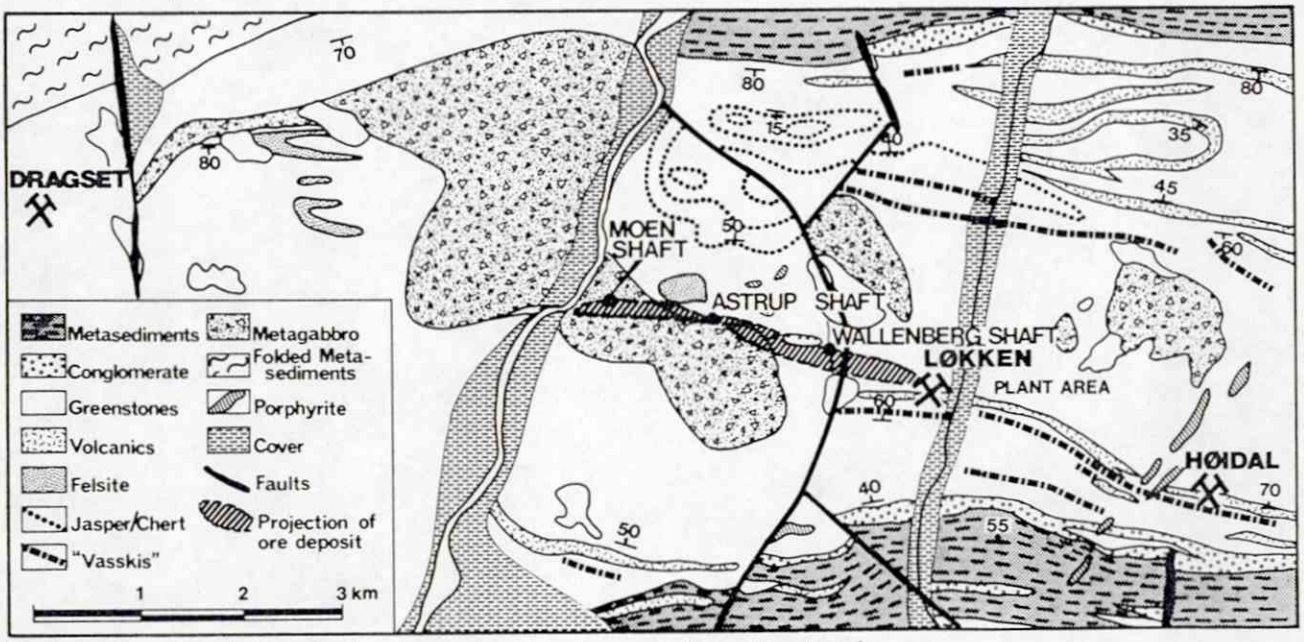


Fig. 2: Geology of the Løkken district, showing locations of mine shafts and the plant site.

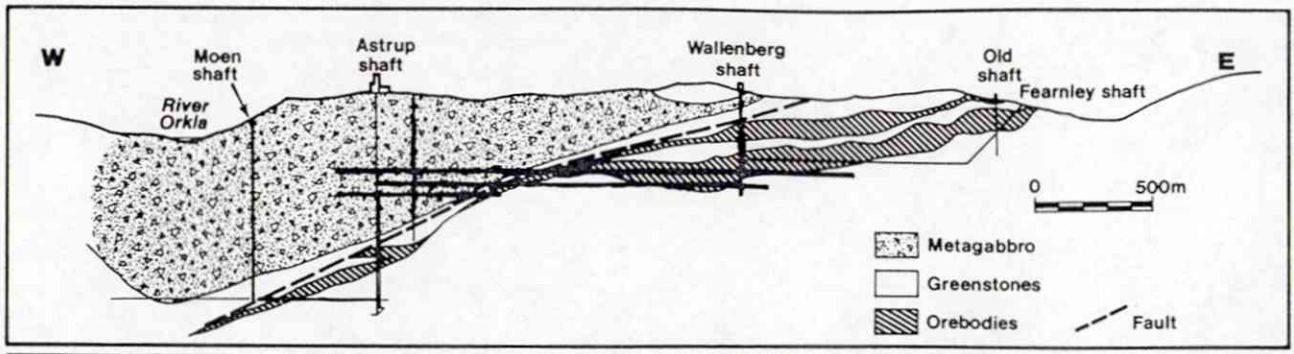


Fig. 3 (above): Vertical longitudinal section of the Løkken massive sulphide deposit.

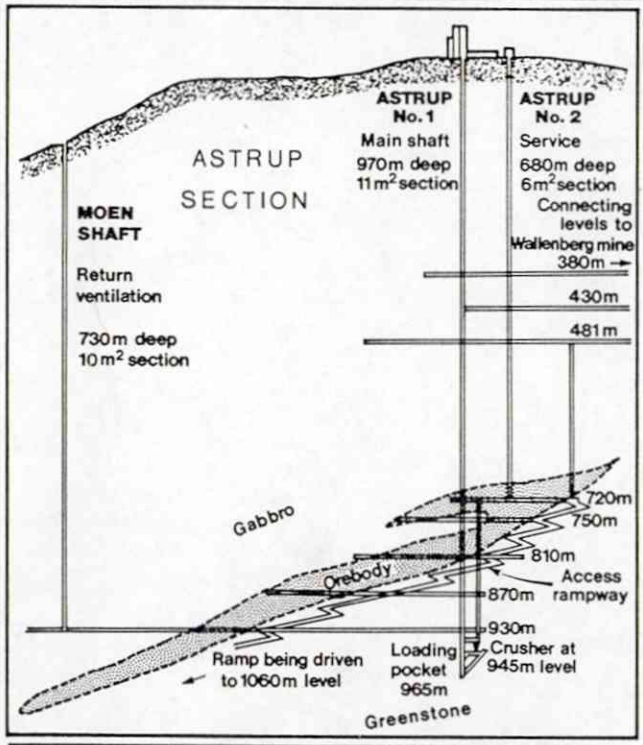


Fig. 4 (left): Details of the Astrup Section shafts and access rampway.

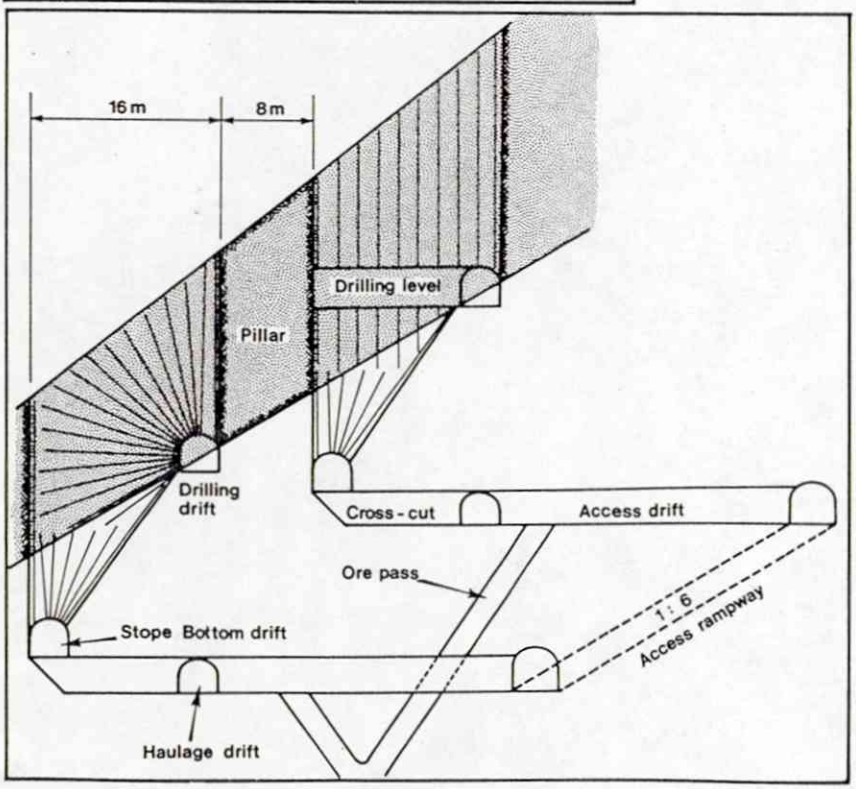


Fig. 5 (below): Diagram of the stoping method employed at Løkken.

longhole drilling, and room and pillar methods. At present 80% of output is produced by sub-level stoping and the remainder by room and pillar. However, over the next three years the proportion of room and pillar extraction will increase to 50%.

Løkken ore is a very hard, fine grained pyritic material which has produced problems of accelerated wear in equipment. Generally roof conditions are good and approximately 3,000 expansion bolts are used annually. Thin layers of jasper and black chert occur with bands of the pyrite within the footwall, while in the hanging wall, zones of disseminated and stringer mineralization are present.

Both pneumatic and electric hydraulic drill rigs are used. A Tamrock Solo 1500H rig with one HL 438L drifter, and an Atlas Copco Roc 601 with one Cop 130 drifter are used for production drilling. Two twin-boomed Minimatic H units with MR 600 booms and HL 438T drifters, plus one twin-boomed Atlas Copco Promec 472 with BUT booms and Cop 1038HD drifters are used for development work.

Fan drilling within the flat orebodies is carried out with the long hole rigs, giving an extraction rate of 10 t/m drilled. Parallel long hole drilling has an output of 16-18 t/m drilled. A hole diameter of 51 mm is used, and hole length is typically 25 m. Dynamite and ammonium nitrate are used for blasting in the stopes.

Diesel load-haul-dumpers were first used in 1977-78 when Schopf machines were introduced, and to date this equipment has proved so successful that no other manufacturer's units have been purchased. At present the mine operates seven Schopf LHD's, including five L110s with 3.5 yd^3 (2.68 m^3) capacity and two L92s of 1.5 yd^3 (1.15 m^3) capacity, as well as four Schopf T193 24 t capacity trucks.

Ore is dumped, via ore passes, down to the main haulage levels where it is transported in 2 m^3 Granby cars on a battery-locomotive operated rail system, to the crusher. After passing through the 1,200 mm by 900 mm

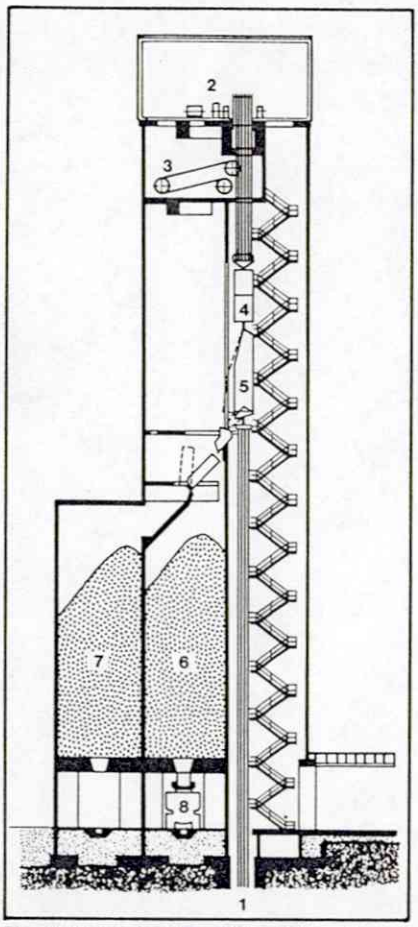


Fig. 6: Cross-section of the Astrup 1 shaft headframe facilities. (1) Shaft; (2) Main hoist; (3) Service hoist; (4) Cage for personnel; (5) Skip; (6) Ore bin; (7) Waste rock bin; (8) Truck collects ore for transfer to mill.

Morgardshammer jaw crusher (capacity 500 t/h) the ore collects in a silo at 965 m level, ready for loading to skips and hoisting to surface.

Waste material is dumped into empty stopes, using the LHDs and T193 trucks.

Development work is underway to extend the rampway to the base of the known orebody and to establish further room and pillar workings. For development work, holes of 45 m diameter are drilled 3.5 m deep. Blasting is by ANFO using Nitro Nobel's VA system and Tri Electronic initiation.

Underground equipment is serviced every 125 working hours and given major overhauls every 1,000 workings hours, the work being carried out in the 175 m^2 underground workshop.

The mine operates a two shift/day system with four managers, 14 dayworkers and 60 shift workers. Output is approximately 20 t/manshift.

Present mine production is 275,000 t/y of run-of-mine ore plus 50,000 t/y waste production. Output for 1985-86 is planned at 300,000 t/y r.o.m. ore plus 20,000 t/y waste. Grade is controlled using an EDB system based on block analysis, with sampling by diamond drilling in the corners of 16 m by 20 m blocks. Present exploration indicates that copper grades will increase with depth while zinc content will decrease.

Ore processing

Mine production is transported from the Astrup shaft silos to the processing plant, some 4 km to the east, in 40 t trucks owned by a private contractor. Ore is stored at the plant in a 15,000 t silo.

The present processing plant, completed in 1974, was designed to produce both copper and zinc concentrates. Up until the second half of the nineteenth century copper had been the only valuable element in the Løkken ore. However from the 1850s onwards, the sulphur content was the most important ore constituent. Cupriferous pyrites from Løkken were sold to roasters for more than a hundred years and Orkla also made elemental sulphur and a copper matte from lump pyrite using the Orkla process. By 1973 other sources of sulphur had become more competitive and the decision was taken to install modern flotation equipment for copper and zinc concentrates production. Pyrite is now dumped in the tailings area.

The Løkken ore comprises 70-75% pyrite, 6% chalcopyrite, 3% sphalerite plus some magnetite. Gangue minerals include 12-14% quartz and small quantities of calcite and chlorite. The extremely fine grained ore texture consists of pyrite grains of between 20-30 microns: about 5% of the chalcopyrite occurs as small particles of 1-2 micron size within the pyrite crystals. The sphalerite contains about 2-5% iron, bound in the lattice. On average the ore contains 41.4% sulphur, 37.5% iron, 2.1% copper, 1.9% zinc, 0.07% cobalt, 0.07% manganese, 0.04% arsenic, 0.02% lead, 0.01% cadmium, 0.0005% selenium, 0.0008% nickel, 19 g/t silver, 0.2 g/t gold and 13.7% silica.

Run-of-mine ore is fed by belt conveyor to a Blake 24 in by 36 in (610 mm × 910 mm) jaw crusher and crushed to −200 mm before passing onto a Symons 4¼ ft (1,300 mm) cone crusher. After screening, it is passed through a Wemco heavy media separation section operating with a medium density of 3 kg/l. The sink product from the HMS plant is screened into three fractions: 150-60 mm, 50-20 mm and −20 mm, after which it is stored in bins which feed the primary autogenous mill (Scanmec). Good control of the size distribution in the mill feed is essential in regulating the mill production, and the amount of 50-20 mm material is critical in controlling the mill power and correct grind. Feed to the mill is maintained at a constant overall rate of 60 t/h.

The autogenous mill consumes on average about 30 kWh/t. An optimal grind from this mill is 85% finer than 37 microns, with the average mill product being 45% finer than 10 microns. If mill power is decreasing and the product from the mill is becoming coarser, more pebbles are transferred from the screen to the coarse fraction. If mill power is still too low the screen plates on the resonance screen, in front of the Symons crusher, are changed from the usual 130 × 130 mm openings to 120 × 120 mm openings. These procedures regulate mill power and give a more constant mill product. The secondary mill (also Scanmec) is a pebble mill which consumes 5-10 kWh/t and produces material 90-95% finer than 37 microns.

Copper flotation

The copper flotation circuit (Fig. 7) begins with a unit flotation cell between the two grinding operations. The rougher concentrate from this cell is cleaned once to give a primary copper concentrate containing 25-28% Cu at 40-50% recovery. The main copper circuit consists of one bank of rougher cells and one of scavenger cells. Rougher concentrate is cleaned in four steps before passing to the copper concentrate thickener. The cleaner tails and the scavenger concentrate together with the tails from the unit cell are then recirculated to the pebble mill cyclone pump.

Combined, the 'unit cell' and main circuit copper concentrates have a grade of 22-23% copper. Higher grades are more difficult to attain because of the fine intergrowths between pyrite and chalcopyrite.

Zinc flotation

The original zinc flotation circuit at Løkken (Fig. 8) operated for four years. However, overall plant results were disappointing and did not match pilot tests. Planned recovery of 70% of the zinc in zinc concentrates was attained but grades reached a maximum of only 50% Zn. Pyrite and pyrrhotite were contaminating the zinc concentrate and the iron content reached 13-14%. Also, the resulting zinc concentrate was extremely finely ground with about 50% at −10 microns.

In 1977 reverse flotation was considered for upgrading sphalerite concentrate and a number of batch tests were carried out at the Løkken laboratory.[1] The present flowsheet evolved from these tests (Fig. 9). As in the earlier system, the feed is conditioned in three steps. The two rougher concentrates are combined and cleaned in three stages to produce a primary concentrate, which is thickened to 60% solids. The tails from the cleaning cells together with the first scavenger concentrate are returned to

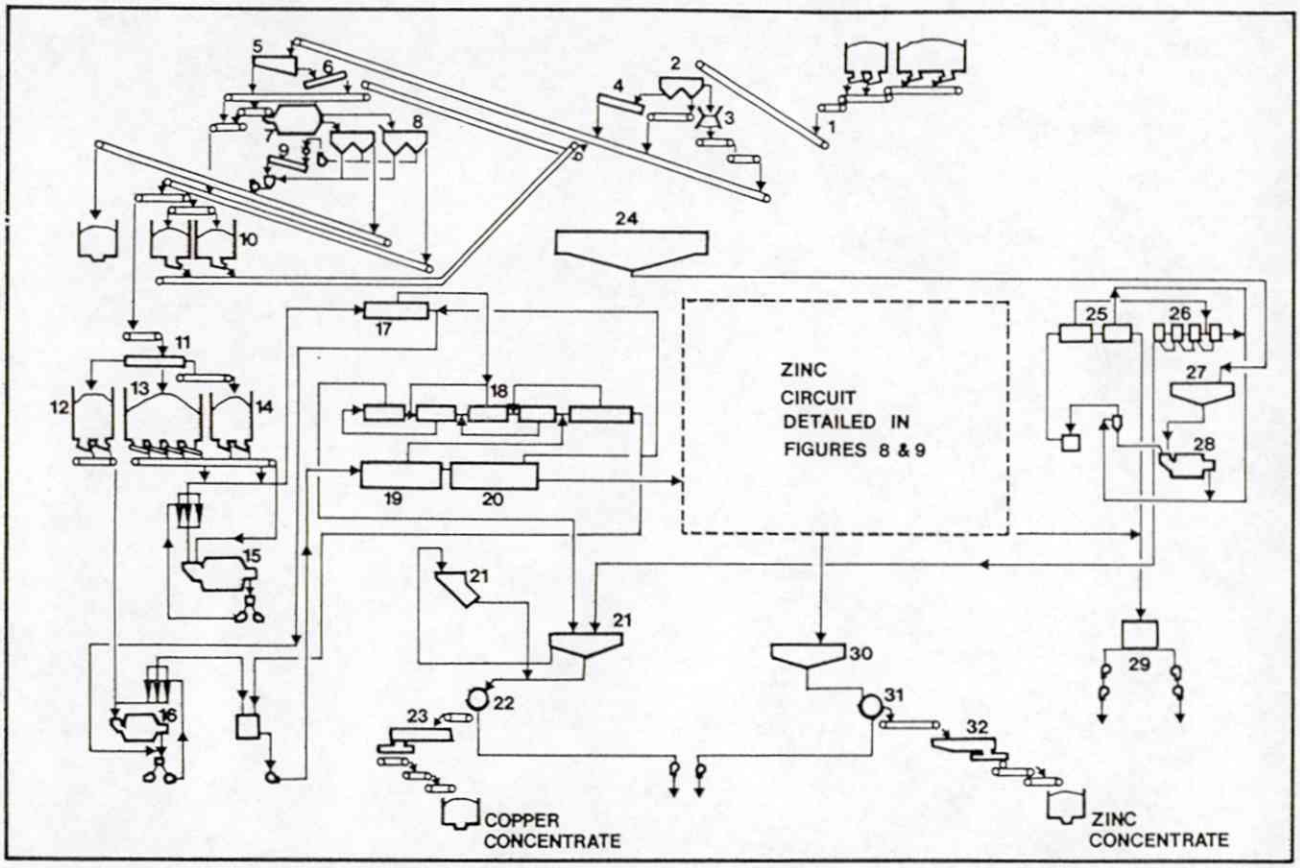


Fig. 7: Løkken concentration plant flowsheet. (1) Jaw crusher 610mm × 914mm; (2) Resonance wet screen 1.22mm + 4.58m; (3) Symons cone crusher, 4¼ft; (4) Screw classifier; (5) Screen, 1.22m × 4.88m; (6) Screw classifier; (7) Wemco H.M.S. drum separator; (8) Filters; (9) Heavy medium cleaning; (10) Double deck screen; (11) Reversible conveyor; (12) Pebbles bin : 50-20mm; (13) Coarse bin : 150-60mm; (14) Fines bin : −20mm; (15) Scanmec autogenous mill; (16) Scanmec pebble mill; (17) Unit flotation cell; (18) Copper cleaners; (19) Copper roughers; (20) Copper scavengers; (21) Copper thickeners; (22) Copper filter; (23) Copper drier; (24) Thickener, 37.5m dia.; (25) Tailings rougher; (26) Tailings cleaner; (27) Tailings thickener; (28) Tailings mill; (29) Tailings pumps; (30) Zinc thickener; (31) Zinc filter; (32) Zinc drier.

the conditioning tanks ahead of the rougher. The second scavenger concentrate is returned to the rougher feed.

Thickened primary concentrate, containing 40-45% Zn, passes on to the reverse flotation circuit. Here the first step is a specially-designed conditioning tank where the pulp is steam heated to 85-90°C and sodium bisulphite is added. The retention time is about 20 minutes. Pulp passing on from the conditioner is diluted by cold water to about 40% solids, giving a temperature of about 50°C in pulp fed to the reverse flotation circuit.

In these cells, pyrite, chalcopyrite, sphalerite/pyrite intergrowths and some small (but economically important) amounts of silver-bearing minerals are floated with xanthates away from the depressed sphalerite. The froth product is recirculated for a regrind in the pebble mill ahead of the main copper flotation circuit, where most of the contained copper and silver are recovered in the copper concentrate.

Since the introduction of the modified zinc circuit, zinc concentrate has been easily upgraded. Recovery of copper and silver have increased substantially. Copper in zinc concentrate was reduced from 0.6% Cu to 0.25% Cu, the formerly contaminating copper now being recovered in the copper concentrate. With the reverse flotation in operation the plant can be operated with higher copper in the copper tailings and the rest of the copper can be recovered from the zinc circuit. This results in higher copper in the copper rougher circuit and the copper concentrate grade has been increased from 21.3% to 22.4% Cu. The process also makes it possible to operate the zinc circuit at a pH of 10 instead of the previous pH of 11.8. As a consequence, lime consumption has been reduced by approximately 60%. In addition to this, greater variations in mill feed can now be handled.

Table 1: Typical metallurgical results in flotation

	Original flowsheet				Present flowsheet			
	Cu %	Zn %	Ag g/t	Fe %	Cu %	Zn %	Ag g/t	Fe %
Feed to flotation	2.1	1.8	16	—	2.1	1.8	16	—
Copper concentrate	21.3	1.2	70	—	22.4	1.0	110	—
Zinc concentrate	0.6	50.0	40	10.5	0.2	56.0	25	5.7
Recovery in the respective concentrate	89.8	69.0	40*	—	92.0	80.0	60*	—

* recovery of Ag into Cu-concentrate.

Table 2: Reagent consumption for the two process schemes in g/t milled (flotation feed)

	Original process		Present process	
	Copper	Zinc	Copper	Zinc
Hydrated lime	3,000	2,000	3,000	500
Z-200	25	10	25	10
Aerofloat 238	12	—	12	—
Amyl Xanthate	60	10	60	15
Ethyl Xanthate	—	20	—	20
Copper sulphate	—	430	—	430
Sodium bisulphite	—	—	—	300
MIBC	4	4	4	5
Fuel oil for steam (litres)	—	—	—	0.4

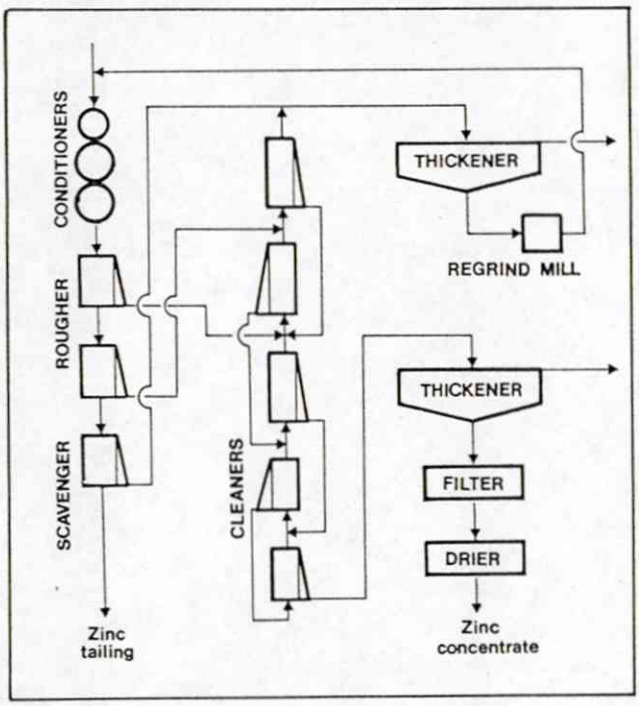


Fig. 8: Original zinc flotation flowsheet.

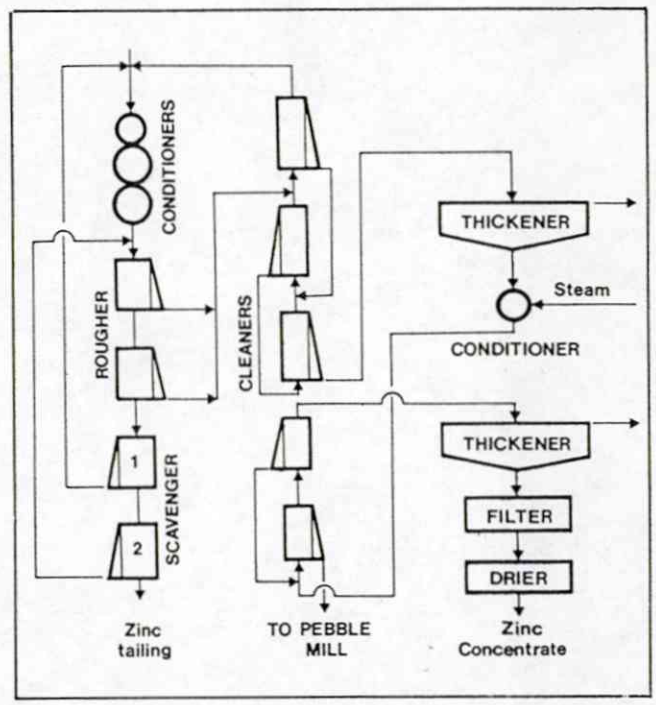


Fig. 9: The present zinc flotation flowsheet.

Plant operating results are given in Tables 1 and 2. The major costs in the reverse flotation process are those for the sodium bisulphite and the fuel oil for steam production. The savings in reduced lime consumption are of the same order as the cost of the sodium bisulphite. The overall improvements in metal recovery after using the new zinc circuit are 3.6% for copper and 7.7% for zinc. At the same time, the value of silver in the copper concentrate is roughly double that obtained previously. The total increase in revenue is of the order of 8%.

The plant treats 260,000 t/y of ore on a 5 d/week schedule, with a total staff of 32. Present production is 21,000 t/y copper concentrates, 5,400 t/y zinc concentrates and 2,000 kg/y of silver. The final products are taken by road to Orkanger for shipment.

In the flotation plant a new range of process control equipment is being installed. The new computerized system includes RXO2 and VT100 units which are at present being tested using Outokumpu-developed software. On-stream analysis of the copper concentrate and tailings is to be improved using a Courier 30 unit.

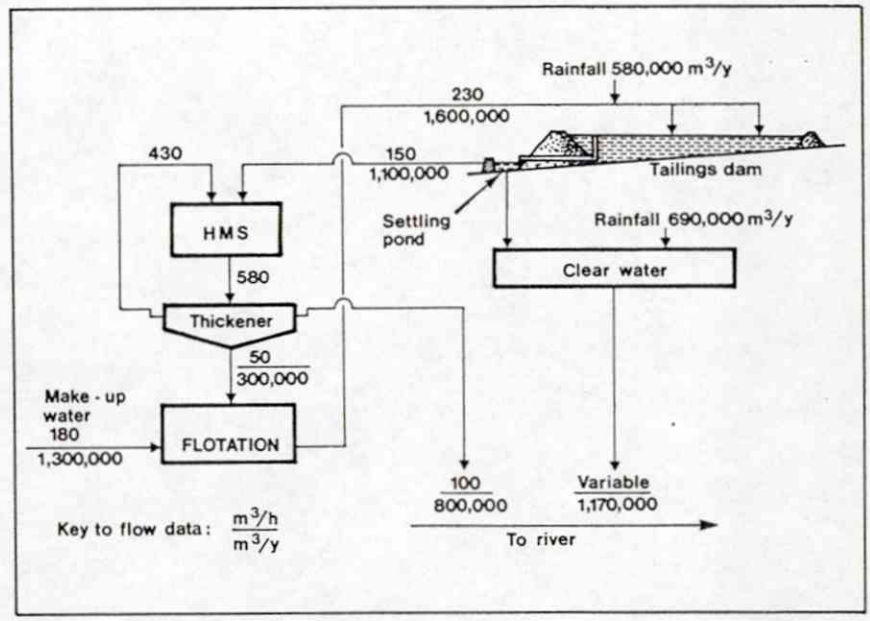


Fig. 10: Water balance diagram for Løkken.

Environment

Acid drainage from the old workings and dumps within the present mine area is the major environmental problem inherited by Løkken Gruber. Previously some 500,000 m^3/y of mine water was draining from the Wallenberg workings into the Astrup mine, but since the sealing of the levels this flow has stopped and the water from Wallenberg is pumped to the sea through a 25 km pipeline.

Tailings are pumped 2 km to the tailings dam area through a 207 mm diameter steel pipeline. Three dams with a total capacity to hold 6 Mt of material are available. Process water is controlled by pumping it to the tailings dams before it is discharged. Each month NIVA, the Norwegian Institute for Water Research, samples and monitors the pollutant content of water overflows from the tailings areas.

The company is undertaking revegetation of acidic wastes to stabilize the dumps and reduce acid drainage. Old banks of slimes, deposited in the Rødbekken ("Red River"), close to Løkken Verk, are to be reworked in the future and removed completely.

Acknowledgements

Mining Magazine thanks Mr. Knut Brøndbo, Managing Director, Mr. Arve Slordahl, Mine Superintendent and Mr. Harald Ese, Mill Superintendent for arranging a visit to the Løkken mine and providing the information on which the article is based.

Reference

1. Ese, H.: "Flotation of Løkken ore", XIV International Mineral Processing Cong., Oct. 1982, Toronto, Canada. ■