Mining and milling operations at Limni mine, Cyprus

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Synopsis
The history, general mining methods and current milling practice
of Cyprus Sulphur and Copper Co., Ltd., are outlined. More
detailed descriptions are given of production and exploration
drilling, servicing and labour in relation to the open-pit operation.
The present crushing and treatment plant layout is outlined,
information being given on some metallurgical and mechanical
problems involved in the economic treatment of the low-grade
waste copper orebody.

The Cyprus Sulphur and Copper Co., Ltd., a subsidiary
of Esperanza Trade and Transport, Ltd., holds a mining
lease of 30 square miles in the northwest of Cyprus,
Limni mine being situated two and a half miles inland
from the coast and three miles east of the village of
Polis (Fig. 1). Mining and milling operations at Limni
are detailed in this paper.

Roman times. The Limni deposit was one of the most
important mines worked during these periods, and it is
estimated, from the tonnage of slag surrounding the old
smelting sites, that some 70,000 tons of copper metal
was recovered. Evidence uncovered by present-day
operations shows that the Romans worked the out-
cropping Limni deposit by opencast methods, followed
by underground mining; although the underground
workings have collapsed over the centuries, sections of
timbering that have been preserved show the ancient
miners to have had a sound knowledge of ground
support (Fig. 2).

As the Romans were only interested in high-grade
copper ore, the lower grades were used as back-fill, and
this, together with the high-grade stope pillars that
remain, today provides a sizable tonnage of above-

average grade. With the break-up of the Roman
Empire, mining in Cyprus declined, activities eventually
ceasing in about A.D. 400. At Limni the underground
workings gradually collapsed and by 1882 a lake had
formed in a surface depression some 400 ft in diameter.

From 1882 to 1918 a succession of companies

Fig. 1 Sketch map showing Limni concession area, Cyprus

History
Copper mining and smelting in Cyprus date back to the
early Bronze Age and continued in Phoenician and

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investigated the property; a number of shafts were sunk and an adit was driven 2000 ft to drain the depression. During this period small tonnages of ore were extracted, but low metal prices, coupled with transport costs to European smelters and competition from Spanish producers, forced the abandonment of these ventures.

Fig. 2 Roman timber set (A.D. 150)

In 1918 the mining lease was taken over by The Esperanza Copper and Sulphur Co., Ltd., and it was decided to work the mine as an opencast operation, the ore to be treated by heap-leaching and cementation.

 bearing gossans and oxidized ore from the upper levels; but this project was abandoned in 1946 due to wartime shortages of spares and equipment.

Following the reorganization of the Company in 1947, a drilling programme was undertaken to determine the ore reserves, and, after sufficient tonnage had been proved, a concentrating plant was erected with a capacity of 500 ton/day (since increased to the present throughput of 1400 ton/day).

During the period 1951–59 two separate orebodies were worked—Kinousa, an underground operation, and Uncle Charles, which was opencast. Both of these were worked out and operations are now confined to the Limni orebody and the small adjacent deposit at Etoimeni.

Geology

From evidence now available the stratigraphic sequence in the area of Limni mine can be summarized as follows:*

\[
\begin{align*}
\text{Sedimentary rocks} & : \\
\text{Terra Reef Limestone} & \\
\text{Lapithos Chalk} & \\
\text{Lapithos Marl and Argillites} & \\
\text{Volcanic rocks} & : \\
\text{Upper Pillow Lavas} & \\
\text{Lower Pillow Lavas} & \\
\text{Basal Pillow Lavas} & \\
\text{Diabase and Gabbro} & \\
\end{align*}
\]

The orebody is emplaced in the Pillow Lava Series. The various phases (tectonic, eruptive and alteration) have had a direct bearing on mineralization and the

<table>
<thead>
<tr>
<th>Tectonic phases</th>
<th>Volcanic phases</th>
<th>Alteration phases</th>
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<tbody>
<tr>
<td>Post volcanic faulting</td>
<td><strong>Period of Upper Pillow Lava mineralization</strong></td>
<td>Final-stage deposition of zeolites (calcite and ancolite)</td>
</tr>
<tr>
<td>Third period of tensional stress</td>
<td>Fourth volcanic phase</td>
<td>Extrusion of Upper Pillow Lavas and addition of feeder dykes to gabbro, Lower Pillow Lavas and Basal Lavas</td>
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<tr>
<td>Second period of tensional stress</td>
<td>Third volcanic phase</td>
<td>Extrusion of Lower Pillow Lavas and addition of feeder dykes to gabbro and Basal Pillow Lavas</td>
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<tr>
<td>First period of tensional stress</td>
<td>Second volcanic phase</td>
<td>Commencement of formation of diabase by infilling of tensional cracks in gabbro. Extrusive phase produced the first recognizable Basal Pillow Lavas</td>
</tr>
<tr>
<td>Geosynclinal movements in Tethys geosyncline</td>
<td>First volcanic phase</td>
<td>Intrusion of gabbro into and through folded Mammelia sediments</td>
</tr>
</tbody>
</table>

Operations continued on a small scale until 1932, when the mine was shut down owing to the low price of copper.

Operations were resumed in 1937, and in 1939 a small cyanide mill was constructed to treat the gold-economic importance of the volcanic suits. These phases are summarized in Table 1; and a schematic sketch map of the opencast is given in Fig. 3.

The diabase dykes trend northwest–southeast and dip southwest. Faulting and fracturing within the diabase is often marked by alterations containing pyrite, chalcopyrite and sphalerite. Although this mineralization in the diabase is of no economic importance at present, it is thought that these mineralized zones were most probably the passageways through which the mineralizing agents reached the Pillow Lava Series.

Although the chemical and mineralogical characteristics of the lavas may have played an important part in aiding the deposition of ore minerals, other physical factors were involved. The numerous sediments reposing immediately above the orebody are assumed to have been deposited, at least in part, prior to mineralization. These deposits formed an impenetrable barrier to the ore-bearing fluids. In support of this assumption is the fact that the highest-grade ore is found immediately below this cap, and further, the thicker the cap, the higher is the grade of ore.
In Limni, umbers rest on Upper Pillow Lavas, with an overall covering of argillites, the umbers and argillites attaining a maximum thickness of 100 ft.

The orebody covers an area roughly 2200 ft long and 750 ft wide, with a maximum thickness of 500 ft. The principal copper ore is chalcopyrite, occurring in massive form with a superficial covering of covellite, giving the mass a bluish tinge. Chalcocite and bornite are also present. Grade in the pit varies between 2·5 and 0·5 per cent Cu. The aim of the mining operations is to maintain an average grade of 0·8 per cent Cu and 10 per cent S to the mill.

Figs. 4 and 5 show schematic block diagrams of the orebody before and after faulting, respectively.

Mining
The present pit lies between 840 and 370 ft above sea level, the planned final depth being 170 ft above sea level. A wall slope of 45°, with 30-ft bench heights, was planned, but major faulting and heavy water-bearing ground conditions have made it impossible to adhere rigidly to the intended slope. Slopes in the pit now vary between 55 and 30°. Argillite, pillow lava, primary ore and diabase will all stand well at 45° over a height of 500 ft, except where they are intersected by steeper faults, or where water-bearing clays underlie the mass, providing no adhesion between rock types.

On the basis of present planning, the ultimate dimensions of the mine will be 3000 ft x 1300 ft x 670 ft deep. A general view is given in Fig. 6.

The mining equipment used at present comprises one RB64 and six RB38 excavators, two LW30 and three LW25 Haulpaks, thirteen 15-ton Euclid rear-dump trucks, one D8 Caterpillar with rippers, one LW Tournatractor rubber-tyred dozer, two Aveling Barford TS250 front-end loaders, two Aveling Barford 99H motor graders, three 7-ton water trucks for spraying, and one Leyland four-wheel drive truck fitted up as a service vehicle with a Tecalenite unit and a 200-gal diesel refuelling tank with pressure pump.

Two LW30-ton Haulpaks carry 2200 tons daily from the open-pit to the crusher—a round trip of 5 miles. This has been made possible by the construction of a wide and well graded road (Fig. 8).

Two LW25-ton Haulpaks and eleven 15-ton Euclid dump trucks handle all the overburden removed. Time studies and the fitting of Servis recorders to all trucks have led to a 10 per cent increase in production. One Haulpak and two Euclid dump trucks are taken out of service daily for maintenance and/or overhaul. Strict maintenance schedules have had a marked effect on costs; an 87 per cent vehicle availability is maintained.

A D8 Caterpillar is used for all heavy ripping and dozing. A rubber-tyred dozer is used for clean-up around the machines and dumps, the mobility of the dozer allowing for hourly clean-up of all work areas—\[\text{with resultant savings on tyres and machines. A rubber-tyred dozer is considered an essential part of the clean-up fleet. Road maintenance is given a high priority, in order to reduce tyre costs, and two graders with three water trucks are used to ensure smooth rock-free surfaces.}\]

A TS250 fitted with a 2½-yd³ bucket is used (a) for loading ore when blending is required, (b) for clean-up of large rocks, drill areas, etc., and (c) for occasional loading of overburden for removal. Experience has shown that a 1½-yd³ excavator will give a superior
disadvantage of the excavator. Because of the many varied uses of a front-end loader, however, it is considered, correctly applied, an essential part of the opencast mining operations—especially when blending of ore from two ends of the pit is required.

One TS250 is employed at a small limestone quarry which produces 6000 tons of limestone annually for use in the mill.

Fig. 9 shows a RB38 loading a LW25 with pillow lava.

Servicing and maintenance
The Tecalemite service truck, with a crew of four, spends half an hour daily in refuelling and greasing each excavator (Fig. 10). The dump trucks working the excavator are greased at the same time by their drivers, and tyre pressures are checked by the Tecalemite crew. Minor adjustments—teeth changes, etc.—are done at the same time, thereby reducing down-time. Compressors and drilling rigs are refuelled and greased at the drill site, but are returned to the pit workshop for servicing and repairs.
All mining equipment is maintained on a preventive maintenance system at the pit workshop, which is adjacent to the workings. The system of unit replacement which has been built up allows for a very comprehensive service to be completed in eight hours. The worn or damaged units are sent to the main workshops at Mavrol, 2 1/2 miles away, for overhaul and/or rebuilding. Sufficient spares are carried at the pit for 24 hours' demand.

**Blast-hole drilling**

Four Halco Stenuick 7HR down-the-hole rigs, drilling 4 1/2- and 5 1/2-inch holes, are used for blast-hole drilling. The 5 1/2-inch drill bits have only recently been introduced at the mine, but results to date indicate definite advantages over the smaller holes. A monthly average of 235 ft per machine shift is maintained with either 4 1/2- or 5 1/2-inch holes. The average life for 4 1/2-inch bits is 4654 ft (5 1/2-inch bits, 6705 ft) and costs for 4- and 5-inch hammers are, respectively, 0.385 and 0.104 p./ft.

The Holman high-pressure 170 lb/in² rig was tested against the 100 lb/in² rig; it gave a 103 per cent better performance. Any new drill rigs will almost certainly be of the high-pressure type. The tests were carried out by drilling (a) hard primary ore, (b) broken ground and (c) wet sticky ground. Under all conditions the superiority of the high-pressure rig was obvious.

Compressed-air requirements are handled by four 600 ft³/min portable compressors.

Owing to the varying ground conditions encountered, and bearing in mind the size of break required, it was decided to carry out a series of drilling tests with the object of defining more clearly the optimum conditions for each type of ground (see Table 2). Ore is broken to about 10-inch particle size and overburden to about 24 in. All blast-holes are drilled 2 ft below the bench floor.

**Table 2 Drilling and loading**

<table>
<thead>
<tr>
<th>Pillow lava</th>
<th>Bench height, ft</th>
<th>Spacing, ft</th>
<th>Burden, ft</th>
<th>Angle, °</th>
<th>Tons</th>
<th>Ammonium nitrate, lb</th>
<th>Ton/lb</th>
</tr>
</thead>
<tbody>
<tr>
<td>4 1/2</td>
<td>25</td>
<td>10</td>
<td>10</td>
<td>15</td>
<td>140</td>
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<td>70</td>
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<td>12</td>
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<td>400</td>
<td>100</td>
<td>4.0</td>
</tr>
<tr>
<td>5 1/2</td>
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<td>18</td>
<td>15</td>
<td>15</td>
<td>450</td>
<td>118</td>
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<td>18</td>
<td>20</td>
<td>720</td>
<td>170</td>
<td>4.2</td>
</tr>
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</table>

18 ft³ in situ = 1 ton.

<table>
<thead>
<tr>
<th>Primary ore</th>
<th>Bench height, ft</th>
<th>Spacing, ft</th>
<th>Burden, ft</th>
<th>Angle, °</th>
<th>Tons</th>
<th>Ammonium nitrate, lb</th>
<th>Ton/lb</th>
</tr>
</thead>
<tbody>
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<td>10</td>
<td>10</td>
<td>15</td>
<td>179</td>
<td>60</td>
<td>3.6</td>
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<td>12</td>
<td>10</td>
<td>15</td>
<td>257</td>
<td>71</td>
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<td>10</td>
<td>20</td>
<td>300</td>
<td>81</td>
<td>3.7</td>
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<td>15</td>
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<td>402</td>
<td>111</td>
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<td>15</td>
<td>20</td>
<td>675</td>
<td>182</td>
<td>3.7</td>
</tr>
</tbody>
</table>

14 ft³ in situ = 1 ton.

Blasting

Ammonium nitrate accounts for 99 per cent of all explosive used in the pit. Although at least 15 per cent
of all holes drilled contain water, it would be uneconomic to switch to slurries, owing to the relatively small amounts involved.

Previously, Opencast Gelignite was used in all wet holes and also for horizontal holes. It has since been found that the use of plastic tubing, provided that average care is taken, enables satisfactory results to be achieved with ammonium nitrate in all vertical holes. A home-made AN–FO loader is used to load horizontal holes.

Extensive trial blasts were carried out in order to determine the best possible primer—a 12-in piece of 100 g/ft cordex, a 50 g/ft cordex loop, one stick of \( \frac{7}{4} \) in \( \times \) 6 in Belox and one stick of \( 3 \frac{1}{4} \) in \( \times \) 5 lb Opencast Gelignite being among those tested. All priming is now done with \( 1 \frac{1}{4} \) in \( \times \) 8 in Belox in dry holes and 2 sticks of \( 1 \frac{1}{2} \) in \( \times \) 8 in Belox spaced 6 ft apart in wet holes.

All blasting is by means of cordex detonating fuse connected up in open-blast pattern, with delay detonators between each row. An average blast would contain 150 holes and initiation is by one detonator and 10 ft of safety fuse.

Mechanical mixing of the ammonium nitrate is done at the magazine. To indicate complete mixing 'Red Tax' oil dye is added at the fuel oil reservoir. Fuel is pumped into an exact measure (5-7 per cent by weight) attached directly above the mixer. It has been found that by making the whole system 'foolproof' a constant mix is maintained. The AN–FO mix is loaded directly into wagons and transported ready for loading to the site.

Exploration drilling
Three main methods of exploration drilling have been used: churn drilling, diamond drilling and a down-the-hole rig linked to a Venturi and cyclone. Churn drilling gives the best results, but it is both costly (£4/ft) and slow (500 ft/month per machine working a 132-h week).

Ground conditions at Limni have proved to be unsuited to diamond drilling with good core recoveries. In a large number of the holes drilled core recoveries were so poor that the information received has not been used to calculate ore reserves. Similar poor results have been obtained in other areas of Cyprus.

Varying pressures, speeds, flushing agents, core barrels, bits, etc., were tried, but no noticeable improvement was obtained in the results achieved. The ore fractured and broke up along the lines of mineralization. A sludge splitter was used and a combination of core and sludge samples was used to determine grade, but the results obtained did not tally with subsequent drilling or with the grade of ore when mined.

The down-the-hole rig is rapid (120 ft/shift) and cheap (10p/ft). Unfortunately, with the present equipment this method is unsuitable for water-bearing ground and for depths greater than 140 ft. Samples are cut at every 10 ft, but as recoveries are virtually 100 per cent, a handling problem arises. To overcome this a sample splitter is used on site and all samples are broken down to 25 per cent bulk before being transported to the assay office. Fig. 11 shows the down-the-hole rig.

![Fig. 11 Halco Stenuick down-the-hole rig with sampling set-up](image)

It has been found that a combination of churn drilling and the down-the-hole rig is best suited to our requirements. A run-over of an area with the rig, followed by churn drilling where further information is required, has proved both fast and economical.

Pumping
The drainage adit driven between 1882 and 1918 is blocked, and at the present time all pit water has to be pumped. A hole into a crosscut off the main adit was effected and the old underground workings are being used as a sump.

It is considered too risky to attempt to open the adit from the portal end as there is no accurate knowledge of the amount of water backed up behind or of the length of the blockage. Once the pit floor is below the 350-ft elevation, an attempt will be made to reopen the adit from the pit end for drainage purposes. The adit was driven on a 1 per cent gradient from the portal.

The water pumped contains soluble copper, which is recovered in four banks of cementation cells. The cells are agitated with compressed air, and the 'baskets' containing the scrap metal are handled by a mobile crane. Two men are employed to clean out the cells.

Costs
Total costs (£) per ton of ore treated are as follows: mining, 52·7; milling, 60·0; and crushing, 8·1. Owing to a planned cutback of overburden removal from 1972, it is, despite soaring prices, hoped to maintain mining costs at around the present level. For the past five years ore and overburden have been mined at a 1:8 ratio, and the ore now remaining for removal will be mined at a 1:5 ratio.
A breakdown of mining costs (Table 3) clearly shows that the bigger equipment is more economical to operate. A direct cost comparison for maintenance and repairs between the 30-ton Haulpaks and 15-ton Euclids cannot be made, owing to the different conditions under which they operate. High speeds (35–40 mile/h), with the resultant heat build-up, scuffing on corners and impact damage, increase tyre costs by approximately 100 per cent on the ore run.

Labour
Expatriate staff are employed on a two-year contract basis, but salaried and daily-paid local employees are employed under separate agreements negotiated between the company and the three trade unions concerned.

Senior personnel are supplied with company housing. The rest of the employees travel to work from surrounding villages. The labour force (Table 4) is gradually changing into a smaller, more skilled group, as mechanization is introduced into more and more aspects of the mining operations. The policy of bigger equipment plus labour-saving devices has had a marked effect on increased production per man-shift over the past few years.

A sample of the wage rates for the daily-paid employees is set out in Table 5. In addition to these rates, a bonus is paid for attendance and production. The cost of living rate is variable: it is reviewed every four months. Free medical attention is provided by a company doctor for employees and their dependants. All employees are members of a provident fund, to which they and the company subscribe.

Milling

Mineralogy
The sulphide copper in the Limni ore occurs chiefly in the form of chalcocite, with small quantities of chalcocite and covellite. A further 5 per cent of the total copper appears as the mineral chalcanite.

Pyrite is the second major constituent of this ore, with quartz, silicates and chlorite as the primary gangue minerals. Small quantities of gold and silver are still found disseminated through the ore (Seattle, op. cit.).

The liberation of this ore appears to take place in two distinct stages. Approximately 90 per cent of the sulphide copper is liberated from pyrite and gangue with a grind of 30 per cent +52 mesh and 40 per cent –200 mesh. The remaining copper has been shown to be in the form of minute inclusions in the pyrite ranging in size up to 15 μm. This portion of the values would therefore not be liberated until all the ore was reduced to less than 10 μm.

In practice, this second stage of the liberation is not achieved and the inclusions are recovered in the pyrite section in the pyrite concentrate or lost to the tailings. With no provision for the recovery of oxide copper minerals and these fine inclusions, the highest recovery that can be expected is therefore in the region of 85 per cent.

The Roman workings have modified the mineralogy in a considerable area of the mine. Extensive oxidation over the years has produced acids and many soluble salts, including copper sulphate. This alteration has had a

<table>
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<th>Equipment</th>
<th>Operation</th>
<th>Maintenance and repairs</th>
<th>Total</th>
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</thead>
<tbody>
<tr>
<td>RB54 excavator, 2½ yd³</td>
<td>0.71</td>
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<tr>
<td>RB38 excavator, 1½ yd³</td>
<td>0.83</td>
<td>1.42</td>
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<tr>
<td>Haulpak, 30 ton</td>
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<td>Euclid, 15 ton</td>
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<tr>
<td>Drilling and blasting</td>
<td></td>
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<td>1.6</td>
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Table 4  Departmental strength at 1 July, 1971

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<th>Department</th>
<th>Operations</th>
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<th>Daily paid</th>
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<tr>
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<td>303</td>
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Table 5  Wage rates (p/h)

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<td>22.7</td>
<td>23.2</td>
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<td>27.3</td>
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<tr>
<td>Cost of living allowance</td>
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<td>3.4</td>
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<td>3.8</td>
</tr>
<tr>
<td>Total</td>
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<td>24.9</td>
<td>27.5</td>
<td>29.2</td>
<td>31.1</td>
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<td>1</td>
<td>A</td>
</tr>
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<td>4</td>
<td>3</td>
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A category is paid to employees with above-average ability.
fundamental effect on the design of the treatment plant and on its efficiency. The average analysis (per cent, unless otherwise stated) of the ore for the year ending 31 March 1971, was as follows: copper, 0.92; sulphur, 12.79; zinc, 0.29; gold, 0.33 dwt/ton; silver, 1.73 dwt/ton; soluble copper 0.014; and oxide copper, 0.045.

**Crushing**
The ore is received from the mine for two shifts per day by 25- or 30-ton Haulpaks, which discharge through a 24-in stationary grizzly into a surge bin (150 tons live load). A pan feeder withdraws the ore at a controlled rate onto a Ross two-roll grizzly, followed by a Kue Ken jaw crusher. This first stage of crushing reduces ore to 3 in prior to screening on a Symons rod deck screen with a 20-in opening. The screen oversize is treated in a 7-ft Symons Soverhead cone crusher set at 5/8-in. The secondary crusher discharge is returned to the screen. The screen aperture can be readily increased to 3/4 in to improve the crushing plant capacity when wet. Clayey ores are being treated during the winter. The screen undersize, being mill feed, is transferred to a 3000-ton capacity storage bunker. Additional undercone storage for a further 10,000 tons is provided in the storage bunker extension, which is filled and emptied by bulldozer. This capacity is used for long-term storage to maintain the mill feed against breakdowns and reduced capacity when wet and clayey ores are being crushed. The crushing plant, which has a capacity in excess of 2000 ton/day, is operated for three shifts per day, 5½ days per week. The seven days per week milling operation is maintained from the storage bunker.

**Washing and grinding section**
Prior to grinding, the mill feed is washed to remove the soluble copper salts and acid, which are detrimental to the subsequent processes. The final blending of the mill feed is achieved by the operation of any combination of the five vibrating feeders installed under the storage bunker. These feeders discharge on to a series of conveyor belts which carry the ore via a Pollock sampler, a 20-ton surge bin and a weighmeter to two 8 ft x 40 ft stainless steel Dorr-Oliver rake classifiers in series. The hourly feed rate averages 55 long dry tons. Fresh water is added with the feed and in the form of sprays above the pool. The sand discharge of the second classifier is the feed to the first stage in the grinding process. The slime overflow from both classifiers flows by gravity to the slime section.

The first stage of the grinding circuit is a 6 ft x 10 ft rod-mill, which operates in open circuit with 3-in diameter rods. The rod-mill discharge is pumped to a splitter box, where the pulp is divided between three secondary ball-mills, 2½-in cast alloy balls being used. The pumping operation, initially, presented a problem, as the mill discharge can contain as much as 50 per cent +25-mesh material with only 10 per cent −200 mesh. The problem was overcome by installing a pump driven by a variable-speed motor controlled by a level-sensing device situated in the pump sump. This effectively maintained the depth of pulp in the sump within relatively close limits and ensured that the pump suction was flooded at all times. Each mill has a cyclone to complete the closed-circuit operation. All mills, including the rod-mill, are fitted with Skaga rubber linings with the 'K'-type lifter bar. Lime is added to the rod-mill feed, and the first additions of the collector combination are added to each of the ball-mills.

**Copper sands flotation**
The overflow of the three ball-mill cyclones is conditioned for approximately 35 min in a 15 ft x 14 ft conditioner. The flotation feed is split between three parallel banks of Agitair flotation cells, half of each bank being a rougher stage, the remainder serving as a scavenger. A standard rougher/scavenger/cleaner circuit is employed, provision being made for an additional cleaning stage, if required, to maintain product grade. Lime added to the rod-mill is controlled to give a flotation pH of 10.5-11. The collector combination employed is a 50.50 mixture of Flotrol R438 and potassium amyl xanthate, Hoechst Flotanol G being used as the frother. Average reagent consumptions (lb/ton) for both sand and slime flotation are as follows: lime, 20.35; R438/potassium amyl xanthate, 0.06; and Flotanol G, 0.08.

The final copper concentrate is thickened in a 30-ft thickener and filtered in a 6 ft x 6 ft disc filter to a 10-11 per cent moisture. The filter cake is transferred by truck to a drying and storage area close to the company’s loading facility on the seashore.

**Slime circuit**
The combined classifier overflow is thickened, rediluted with fresh water, and thickened for a second time in a series of two 40-ft thickeners, diaphragm pumps handling each underflow—this process separates the dissolved salts and acid from the slime. The thickener overflow flows by gravity to cementation launders for the recovery of copper by use of scrap iron. The second underflow is conditioned with lime and the same collector combination as sand flotation, then being diluted to 12 per cent solids for flotation. The rougher and scavenger concentrates produced are combined with the equivalent products from the sand section for further processing.

**Pyrite flotation section**
The tailings from both sand and slime sections are combined for treatment in two spiral classifiers in parallel. The classifier overflows are cycloned through a bank of three 10-in cyclones to produce a −300-mesh overflow, which is rejected to tailings. The combined cyclone underflow and classifier sands product, when diluted to 30 per cent solids with fresh water, constitutes
Fig. 12  Mill flowsheet

(1) Amsco apron feeder
(2) Ross two-roll grizzly
(3) 42 in x 30 in Kue Ken jaw crushe, Armstrong Whitworth
(4) 30 in x 16 in Thomas and Foster stone breaker
(5) 12 ft x 6 ft Symons rod deck screen, Nordberg Manufacturing Co.
(6) 4DS Locker vibrating feeder
(7) 4DS Locker vibrating feeder
(8) 7 ft Symons Shorthead cone crushe, Nordberg Manufacturing Co.
(9) 1600 mm x 1800 mm Wedag impact crushe
(10) 24 in x 48 in 4DL Locker vibrating feeders
(11) Patlock sampler
(12) Electronic belt weigher, Craven Electronics, Ltd.
(13) 30 ft x 9 ft Stainless steel washing classifier, Dorr-Oliver Co., Ltd.
(14) 10 ft x 6 ft Head Wrightson rod-mill
(15) 4 in x 3 in Simonacco Warman pump
(16) 6 x 5 Denver SRL pump
(17) D10B Krebs cyclone
(18) 4 ft x 14 ft thickeners
(19) 6-in acid-proof Simplex Denver diaphragm pump
(20) 44 ft x 14 ft thickener
(21) 5 in acid-proof Simplex Denver diaphragm pump
(22) 3 x 3 Denver SRL pump
(23) 8 x 8 Denver conditioner
(24) 2 Banks of 10 no. 36 Agitair flotation cells
(25) 6 x 6 Marcy mill, Pajson, Ltd.
(26) 5 x 5 Denver SRL pump
(27) D15B Krebs cyclone
(28) 22 in x 7 ft Hardinge mill, Ernest Newall
(29) 3 x 3 Denver SRL pump
(30) D10B Krebs cyclone
(31) 10 ft x 6 ft Head Wrightson ball-mill
(32) 5 x 5 Denver SRL pump
(33) D15B Krebs cyclone
(34) 14 ft x 14 ft Denver conditioner

(35) 6-in type C Wilfley pump
(36) Two banks of 8 no. 48 Agitair flotation cells; one bank of 12 no. 36 Agitair flotation cells
(37) Three banks of 12 no. 36 Agitair flotation cells
(38) 5 x 5 Denver SRL pump
(39) D10B Krebs cyclone
(40) One bank of 6 no. 36 Agitair flotation cells
(41) 3 x 3 Denver SRL pump
(42) Two banks of 6 no. 36 Agitair flotation cells
(43) 3-in Vaceal pump
(44) 30-ft Denver thickener
(45) 3-in Duplex diaphragm pump
(46) 6 x 6 Denver disc filter
(47) 3-in Vaceal pump
(48) 1-6 m x 7 m Wedag spiral classifier
(49) 6 x 4 Simonacco Warman pump
(50) Three D10B Krebs cyclone
(51) 5 x 5 Denver SRL pump
(52) Two banks of 10 no. 36 Agitair flotation cells; one bank of 8 no. 48 Agitair flotation cells
(53) One bank of 6 no. 36 Agitair flotation cells
(54) 5 x 5 Denver SRL pump
(55) 10 x 8 Denver SRL pump

Cresylic acid as frother. Average consumptions of potassium amyl xanthate and cresylic acid are 0·04 and 0·05 lb/ton, respectively.

The final pyrite concentrate is pumped through a 3000-ft pipeline to the company's loading facility, where the concentrate is stockpiled, dewatered and dried in one of three dams prior to shipment.

The combined tailings flow by gravity to a 100-ft thickener, from which the thickened underflow is pumped to a tailings dam. The clear overflow is recycled to the mill for reuse.

A flowsheet of the milling operation is given in Fig. 12.

**Loading**

The company's loading facility consists of a drying and storage area for both copper and pyrite concentrates and a loading jetty (Fig. 13). The final copper concentrate is sun-dried to approximately 8 per cent moisture and then transferred to a covered storage area to await shipment. The loading jetty consists of a feed bin with a series of five belt conveyors (total length, 700 ft). The concentrates are transferred from the storage area by use of hydraulic excavators and tip trucks. From the feed bin the concentrates are carried by the conveyor system, which includes an automatic sampler and weighmeter, along the jetty to 50-ton barges. The barges are towed by tug alongside the cargo ship, which is anchored in the bay, to be unloaded by use of electrohydraulic grabs suspended from the ship's loading derricks.

**Water supply**

Water for milling operations and domestic use is pumped from a series of nine wells situated in an aquifer located up to two miles east of the property. Some of these wells deliver fresh water, and the remainder is brackish, with a chlorine content up to 7000 ppm. Some fresh water is supplied for domestic
use, the remaining fresh and brackish water being combined for mill use at approximately 3000 ppm chlorine.

In view of the scarcity of water in Cyprus, some attempts have been made to use sea water to replace the fresh water used to wash the mill feed in the stainless steel classifier, and it was shown on a laboratory scale that salt water had little effect on the subsequent cementation or flotation processes. The combination of salt water and acid from the ore was, however, found to increase the rate of corrosion on the stainless steel rake mechanism. Corrosion takes place on internal surfaces, such as bolt and nut threads, rather than on surfaces in direct contact with the pulp. Salt water is available on a standby basis for use in the event of a water shortage.

Acknowledgement
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