Mining and milling operations at Limni mine, Cyprus

T. O. Trenerry A.C.S.M., C.Eng., M.I.M.M.
B. G. Pocock A.R.S.M., B.Sc., C.Eng., A.M.I.M.M.

Report of discussion at April, 1972, general meeting (Chairman: Professor M. G. Fleming, President) and contributed remarks. Paper published in Transactions/Section A (Mining industry), vol. 81, January, 1972, pp. A1–12

T. H. B. Lawther said that it was disappointing that neither author was able to be present. Although he was delighted to introduce the paper for them, it was some time since he had had anything to do with Limni mine, and they knew the present position better than he did. He had, however, been able to collect some additional remarks from both the authors, as well as some other comments. Two alterations had to be made: on page A8 'chalcanthite' was incorrectly spelt, and in the north–south section in Fig. 3 (p. A3) the curved line of crosses did not indicate the boundary of the mineralized Basel Group but the approximate present profile of the open-cast workings.

Mr. Trenerry had advised him that all remnants of Roman workings, mentioned on page A1, were now exhausted, but had added that it might be of interest to know that the stairways used by those excellent engineers were always inclined at 50°, and that during current mining operations in the old Roman areas baskets, used for carrying ore, had been found. Pottery found in graves in the surrounding hills had been dated as belonging to the fifth century B.C. and, so far as was known, belonged to people who were then working in the mine.

On the use of the 5½-in diameter bits (p. A6), further work had shown that the latest results were not quite as good as had at first been thought. Bit costs per foot were high due to the tungsten carbide inserts falling out, but when the savings, i.e. one less drill and the use of less cordex because of the improved spacing of drill-holes, were taken into account, the overall cost was still on the right side in comparison with the 4½-in diameter bits.

Mr. Trenerry also wanted to make quite plain that his comments under Exploration drilling (p. A7) were not intended to suggest that the methods employed were better than those which could be obtained with a large rotary drill, but they did give satisfactory results when limited equipment was available.

He had some further comments to make on the servicing of the LW30 Haulpaks, and, for the record, engines and transmissions were changed every 6000–7000 h. Front tyres on the 40 miles/h run between opencast and mill lasted for 2000 h, and rear tyres for 4000 h, whereas in the pit (on overburden removal) the tyres lasted 3000–4000 h and 6000 h, respectively. For open-pit work relugging of tyres, costing one-third of a new tyre, gave approximately 60 per cent of the hours. Two relugs were possible, but further ones meant that tyres could only be used on slower equipment, such as the TS250 front-loader. Daily checking of tyre pressures allowed. It was estimated, at least one extra relug per tyre, as it was tyre well damage that caused the greatest damage.

Mr. Pocock had some comments on the milling side. He particularly mentioned the use of the rubber linings (p. A6), which had proved very superior to cast manganese steel linings. Initial costs of both types of linings were very similar, but the life of the rubber shell plates was two years and that for rubber lifter bars was one year (ten months for the complete steel lining). There was also a considerable saving in labour costs, downtime and power consumption, the rubber lining being lightweight and easy to handle. He had also written that the use of Skena rubber for the rod-mill lining at the time of the initial installation was not unique, but was quite unusual. Wear rates were approximately double those for the ball-mill, the shell plates and lifter bars having a life of one year and six months, respectively, compared with the complete steel lining at four to five months. A major problem related to the use of rubber linings in a rod-mill was created by the break-up of worn rods. The lining could be rapidly destroyed by
the small flattened lengths produced as thin rods broke up. It was found necessary to call the rod charge regularly and remove all rods which were approaching a size which would break. In practice, the rod charge was replaced every three months with new rods, the partly worn rods from the previous charge being recharged to maintain the grinding load. The size of partly worn rods determined the time of recharging during the three-month cycle. That practice undoubtedly resulted in a higher cost of grinding media per ton of feed than with steel lining. No loss in capacity was evident during the period when grinding was carried out in the mill containing all 3-in rods. On the whole, the rubber linings were more successful in the ball-mill than in the rod-mill.

Mr. K. J. Whiter, production manager at Limmi, had advised that there was insufficient information on the use of “K”-type (rubber) lifter bars in the ball-mills to ascertain their life, but he thought that they would last at least equal the old type. They had the advantage that they did not require reversing when half worn, as was the case with the previous lifter bars, which led to a saving on mill downtime. They would reverse the shell plates when installing the second set of “K”-type lifters as they believed that that would increase shell plate life.

In the rod-mill the life of the “K”-type lifter bars was identical with the previous type, but as downtime was avoided for reversal, they had that advantage.

The consumption of grinding media in the rod-mill was equal to that before the introduction of rubber linings. That had been accomplished by modifications to the feed to the rod-mill which allowed them to maintain rod-mill density at or above 75 per cent solids. That was done by returning part of the secondary ball-mill discharge to the rod-mill feed to act as a ‘lubricant’ in the rod-mill circuit, thereby eliminating the large quantity of water sometimes required to get the desired feed from the washing classifiers into the mill.

One other point briefly referred to in the paper (p. A9) concerned the installation of a pump driven by a variable-speed motor controlled by a level-sensing device situated in the pump sump.* The general manager of the mine, Mr. R. H. S. Ewins, had also provided some notes, which, it was felt, would be of interest. The mill (p. A2) was originally designed as a gravity mill with tables with a capacity of 500 ton/day. That was unsuccessful and it had been changed to all flotation, additional units being added over the years to raise the capacity to the present 1400 ton/day.

The crushing section had been redesigned and two new major units had been installed, including a Kue Ken and a 7-ft Symons shorthead crushe. Those units might appear to be oversized for the plant, but the idea was to reduce crushing time and save labour costs. Both units had given excellent service.

In the grinding section the layout of the mills was, frankly, a mess, as additional mills had to be fitted into the original building. The mills in the gravity plant were a 22 ft × 7 ft Hardinge and the 6 ft × 6 ft Marcy. When the mill was changed to flotation and the capacity raised, two secondhand 6 ft × 10 ft mills were obtained. One was installed as a ball-mill, as the primary mill, and the Hardinge and Marcy as secondary mills. When capacity was again increased, the second 6 ft × 10 ft mill was installed as a rod-mill and the others became secondary mills.

The rod-mill could not be sited to give a gravity flow of the discharge, so that had to be pumped. Variations in the feed caused fluctuations in the density of the mill discharge and pipeline blockages were frequent. The installation of the variable-speed pump already referred to had largely overcome those problems.

There were wide variations in ore types in the mine, and sudden changes required close attention to flotation. As was usual with most copper–pyrite separations, the main problem was efficient depression of the pyrite in the copper section. In certain types the pyrite was very ‘active’ and difficult to depress.

Copper tailings losses were mainly in composite chalcopyrite–pyrite particles. The problem was investigated at the Royal School of Mines, London, but results had shown that to liberate the chalcopyrite grinding would have to be carried out to such a degree of fineness as to be uneconomic.

Those were the extra comments by those who were, or had been, actively engaged at the mine. In his view the authors had produced a very competent paper and had given a very good description of what went on at a very efficiently run mine. His personal comment, which he made from his own experience of Limmi, was that a lot of interesting facts had been left out. Many were the improvements which had been made over the past eight or nine years to attain the present peak of efficiency. He supposed—and he knew that the authors realized that—that the paper would have been much too long if all the improvements had been faithfully catalogued.

Bryan Earl congratulated the authors on a particularly interesting paper on a mining operation, which discussed the integration of the various aspects of the undertaking. He hoped that there would be more papers of that type published in the Transactions. He would add that the use of such good illustrations enhanced the value of the paper.

There was a point he wished to comment on, as the paper dealt with a classical mining area. He was glad that trouble had been taken to draw attention to the historical importance of Limmi mines. If old workings, such as smelting furnaces, should be revealed during any work on the retreatment of old dump material, perhaps some of those could be preserved for investigation by such a body as the Historical Metallurgy Group, interested in the study of historical technology. There was an increasing general interest in the history of mining and metallurgy, and there might be a future for developing the inherent historical aspect of the site from the tourist industry point of view, as had been done, for example, in the west of England. That could be beneficial to the mining industry in drawing attention to the importance of mining to civilization, rather than emphasizing the worst ‘ecological’ side of metallurgical operations, as had become prevalent at the present time.

With regard to the drilling and blasting, he wondered if the authors had considered charging the wet holes with the ammonium nitrate–fuel oil explosive direct, omitting the plastic tubing, after the water had been blown out. That could simplify blasting work. The relatively large hole diameters employed at the mine could be useful in aiding the detonation propagation of such explosives under wet conditions, possibly with an increased priming charge to ensure initiation.

He would also suggest that attention be given to the angling of the shot holes. The authors adopted different inclinations at varying bench heights (Table 2, p. A6). It should be possible to use 20° inclination for all the shot holes, and so obtain a more standardized drill set-up. That might also be beneficial in the shallower benches in achieving a better distribution of explosive—ratio of stemming to charge length—as the hole length would be increased with a greater angle of inclination. The angle of 20° would help to avoid any problem of ‘toes’ being left, as a favourable vertical bursting angle would be available, as well as controlling the face overhang. He noticed that ‘delay detonators’ were mentioned: that was perhaps a misprint for ‘detonating relays’, as the firing was by fuse and cordite.

With regard to the prospect hole drilling, he wondered if the use of down-the-hole isotope fluorescent equipment had been considered in preference to attempting to sample by narrow-diameter diamond drilling and coring. The fluorescent analyser could enable small-diameter holes to be used and overcome the poor core recovery.

Finally, he would like to know about the methods used for assaying. His own experience in the last year had been that a 10 per cent or greater difference in results was often found between completely reputable assayers, and that could have a significant influence when marginal deposits were being

*See contributed remarks by K. J. Whiter (pp. A195–6).
evaluated. Perhaps the authors would say which assay techniques had been the most reliable in their own experience—that would be particularly useful in comparison with the recent paper by Lister and Gallagher.*

Dr. S. A. Wrobel said that the authors were to be complimented on a detailed and clear description of the mining and milling operations at Limni mine. There were a few points which he wished to raise, and these were also added to the paper.

It was stated (p. A1) that the Roman underground workings had collapsed over the centuries. In 1954, accompanied by the company's geologist, Dr. Gordon-Smith, he had been fortunate enough to visit those underground workings. The Roman shaft through which they had entered the old workings was inclined at approximately 55° and, although partly inaccessible, some of the workings were still intact. At one end were heaps of human bones, and it was explained that slaves and condemned men were used to work underground, never coming out—that was in accordance with the history of Roman mining. In the part of the workings still accessible the supports were in place, and he and Mr. Smith had been able to talk to each other quite loudly.

On page A2 it was stated that the mining lease was taken in 1922 by The Esperanza Copper and Sulphur Co., Ltd. It should be of interest to observe that the mine was, in fact, purchased as a gold mine, and for many years afterwards shareholders would ask what had happened to the gold. Gold and silver were present in the Limni mine, partly in free metallic form, partly locked with copper sulphide minerals and partly locked with jaspinite. The pyrite appeared to be completely free of both gold and silver.

On the same page it was stated that in 1947 the company was reorganized and built a concentrating plant for 500 ton/day. That point required some enlargement, as it was interesting to note that the plant built was a complete gravity plant, with trommels, jigs, classifiers, and shaking tables. That fully equipped plant did not produce a single ounce of directly salable product, and it was fortunate for everyone that it was changed to an all-flotation plant, after its first visit. The whole changeover had been carried out with local craftsmen and labour under his direction when the company was almost penniless; in approximately 18 months the revamped concentrator was essentially as now described by the authors. The present general manager, Mr. Ewins, was at that time assistant general manager, and was the pillar of strength and support in the revamping stage of the mill. The company, as history had so clearly shown, had gained strength from the new flotation plant and the subsequent sale of salable products, the benefit of which was finally reaped after so many dormant years.

He would like to suggest, for the benefit of budding mining engineers, that they learn the wisdom of other people's errors as well as their good deeds, and for mining companies to ensure that they employed qualified engineers for a plant design which would produce the desired products of a salable nature.

Professor R. N. Pryor asked if more information could be given regarding exploration. The paper had only referred to churn drilling and the use of the down-the-hole drill, depths of 140 ft being mentioned. He understood that the pit had been deepened to 200 ft, so it would seem that drilling must have been carried out to greater depths. It would also be interesting to know more about the hole-spacing, and how they had set about determining the ideal intervals.

The Chairman said that he had been interested in the additional information that had been given about the rubber linings, but had been somewhat surprised that there was no loss of capacity on changing over to these. In at least two operations in the United Kingdom capacity had been very much lower with the rubber linings than had been expected. In one case the change from steel to rubber had resulted in a direct reduction in capacity of the order of 20 per cent.

He would be interested in any further comments that the authors might like to make on that point and also on the comparative life of the ball-mill liners.

J. E. Denyer stated that a rubber lining had been tried in the rod-mill at South Crofty, but when the rods were thin and broke, the shot lengths penetrated not only the rubber but also the shell of the mill, so they had had to go back to steel liners. As had been suggested, the difficulty might have been overcome by removing worn rods at frequent intervals, provided that that was always done thoroughly.

T. H. B. Lawther said that he would not attempt to answer the very interesting question raised, but he would comment on the query by Professor Pryor about the drilling. It was deeper because the tonnages found by churn drilling, all completed before the present managers and engineers were at the mine, and on which the whole operation was based, were more or less evenly spaced and went down some 300-500 ft, right through the mineral zone. But the exploratory drilling, done by down-the-hole drills, was only down towards about 100 ft, and generally was done only to check if the churn-drill results were correct. There had been occasions when they had come to mine ore, which churn drilling had shown to contain 1-2 per cent copper, but had found little or no copper. Therefore, the down-the-hole drilling was carried out to find out whether there was any copper there; if there were not, the rock had to be mined and put on the waste heap.

**Contributed remarks**

K. J. Whiter* The rod-mill discharge pump and automatic controls consist of the following.

1. (4/3) Simonaccio-Warman CAM slurry pump operating through a speed range of 1000-1210 rev/min and pumping 75-200 gal/min of pulp at 65 per cent solids.

2. (4/3) Laurence Scott and Electromotors, Ltd., N-S variable-speed motor designed to produce 15 hp at 1900 rev/min infinitely variable down to 1570 rev/min. Complete with tacho generator mounted at the non-drive end for supplying a signal to the automatic control scheme.


4. See L.S.E., Ltd., starter incorporating Arsec automatic speed control control to control motor speed over a varying level when receiving a 0-10 mV signal into 2000 VΩ, the signal corresponding to the full-speed range of the equipment.

5. Transducer for variable-speed control, comprising (5.1) Fielden telster type 62 A capacity to microamp converter; (5.2) Fielden stainless steel electrode type 50/B1/2/C rod, fully covered PTFE, length 3 ft; (5.3) Fielden telster type 62 C power supply, ranging and amplifier unit to supply the telster 62A with 12 V dc from ac mains and incorporating the 'set full scale' and 'set zero' controls; (5.4) Fielden type 'A' display unit and three-term process controller; and (5.5) Fielden matching amplifier to work with item (5.4) to give 0-10 mA output in 2000 VΩ.

The equipment given under item (5) is for control of the speed of the 15-hp type N-S motor driving the pump. The electrode is installed in the pump sump and the object of the variable-speed control is to match the output of the pump to the input of the sump so that a flooded suction at the pump inlet is always maintained and there is no overflow of slurry from the pump. The electrode forms a capacitor between itself and the sump and variation in the value of the capacitance is achieved as the pulp level alters (capacitance change is proportional to level).


* Production Manager, Limni Mines, Cyprus.
This supplies the signal for the level indicator of the type ‘A’ display unit (item 5.4).

The three-term process controller receives an output from the display unit depending on the difference between the set value and the measured value from the electrode in the sump and, from this signal, it generates the necessary proportional, integral and derivative functions to give a current used to operate the controlling unit. In this case the Arsec automatic control system controlling the pump speed. This control system then accelerates or retards the motor depending on whether the pulp level in the pump is above or below the set point.

Owing to the characteristics of the rod-mill discharge, the small size of the pump sump and the pump characteristics, there is a tendency for the control to ‘hunt.’ This had been reduced to acceptable limits by setting the process controller proportional band to 100 per cent integral time to a maximum of 30 min, and derivative time to a minimum of 0 sec. Integral and derivative action has been reduced to a minimum, and, hence, the sensitivity of the system has likewise been reduced.

The meter scale of the type ‘A’ display unit shows the ‘variable indicator’, ‘set point indicator’ and the ‘valve position indicator’.

For operation the set point indicator is in mid position, corresponding to the pulp level in the sump being midway between upper and lower limits, the variable indicator shows the actual pulp level, and the valve position indicator shows the pump. Upper and lower level limiters in the sump are close and only allow for a level variation of 25 in.

The rod-mill discharge is 75 per cent solids, and normal operating procedure is to add sufficient water at the pump sump for the slurry to be pumped at 65 per cent solids. This has proved to be the density for maximum operating efficiency.

To improve the operating efficiency of the controls the following modifications have been made. (1) The accelerate and retard contactors of the Arsec control system required attention to the contacts and springs at two-weekly intervals and replacement yearly. The original contactors were replaced by Allen West type UCO open-type contactors. This type of contactor was installed recently and, to date, has performed for six months without the need for attention. (2) The polarized discriminating relay of the Arsec control system actuating the accelerate and retard relays was found to be oversensitive and was adjusted to operate at a slower rate. (3) The pilot motor on the single induction regulator was fitted with a brake to prevent over-run. Initially, wear on the brake was rapid and it was removed until satisfactory conditions of control were established. No reason has been found to refit the brake, and it was discarded.

The unit operates reasonably troublefree and requires very little maintenance. The motor is inspected at three-monthly intervals and commutator brush life is averaging six months.

This is the only item to require regular replacement.

Three main advantages arise from the use of the automatic pump compared to a fixed-speed pump. (1) The throughput of the rod-mill is maintained at 25 per cent more than was possible with a fixed-speed pump. (2) The slurry density and feed rate to the secondary ball-mill is more constant, and this has improved grinding efficiencies. (3) With a fixed-speed pump it was necessary to have an operator permanently at the pump site, but pumpline blocks were frequent. This is no longer necessary and pipeline blockages are no longer a major problem.

The pump is fitted with rubber linings and impeller and the rate of wear is severe, but this must be expected. The impeller life varies from 250 h for hard, highly abrasive ore types to 700 h for the softer, less abrasive ore. The casing liners average 1700 h.

C. Harvey Richards The authors are to be congratulated on their very interesting paper. The following remarks are intended to fill in a little of the background to the workings at Limni. First, the name ‘Limni’ is locally said to be derived from the Greek ‘limnoi’ which was to over the old Roman workings for many hundreds years. It may be of interest to mention that when the lake was drained, towards the end of the nineteenth century, no attempt was made to precipitate the copper content—a fact regretted by many later operators.

Kinouza and Uncle Charles were worked mainly as pyrite deposits; Uncle Charles was a small opencast and, Kinouza was worked from underground stopes. As the stoping operations approached the surface at Kinouza, the natural subidence of the surface area disclosed an ancient burial ground from which many specimens of ancient pottery were salvaged. Rumour had it that some ancient gold ornaments were also found.

It is noted that it is the intention to open up the old low-level adit after the 370-ft bench has been reached. In 1980 the question of opening up this old adit to save pumping costs was considered, but postponed due to the then low price obtainable for copper. One wonders whether drilling a large borehole, horizontally, through the concrete plug, towards the old ‘Goat’ shaft had been considered. This might have been done by drilling through a control valve, thereby draining the deposit ahead of mining.

The use of ammonium nitrate for blasting is interesting, as in 1890, due to a shortage of explosives, experiments were carried out by our geologist, John Gordon-Smith, with ammonium nitrate fertilizer and失信 mixture, packed in cellophane bags. Holes were drilled 6 in in diameter by churn drills and the holes were charged by use of a stick of blasting gelignite at the bottom of the hole and another stick at 4-ft intervals along the hole. The holes were detonated by cordex instantaneous fuse. The results were encouraging, but were halted on receipt of an order from the newly formed Mines Department, which informed us that our action constituted illegal manufacture of explosives; when a licence was applied for we were told that the Government did not wish to become interested by the local population that ordinary fertilizer might be used in the manufacture of explosive! At the time of Independence it was ruled that employees should be in the ratio of seven Greek to three Turkish Cypriots, and, fortunately, that ratio prevailed, the mining being almost entirely carried out by Turkish employees and most of the remaining employees being Greek. Operations during the past few years must have been very difficult and one wonders whether the ratio has been maintained.

In 1960 a four-shift basis for continuous working of the mill was introduced by Mr. R. H. S. Ewins, the present General Manager. This was resisted, at first, by the Communist-controlled Trade Union, but after a short trial period, when it was seen that it allowed the men a long weekend in Nicosia every three weeks, it was accepted. This system cut out a lot of overtime and gave extra men for three days a week for repairs and general cleaning-up jobs. It may be worth placing on record, as it is not generally known.

In the section on milling no mention is made of the average pH of the mill feed. It used to be just over 5.0—necessitating some 20 lb of lime per ton of ore milled. Has this improved? Another interesting figure missing is the power consumption in the mill in units per ton milled. In this connection it is understood that power is supplied from a national grid, and it would be interesting to know how this compares with the cost per ton milled with the power formerly supplied by the mine’s own power station.

A point of interest is that the mine boreholes which supply fresh water lie close to the sea shore and, in some cases, about 100 yd from the sea. The aquifer is fed by underground water from the Troodos Mountains and its weight keeps the sea water back, but the wells have to be constantly checked for salinity as any over-pumping would allow the sea water to encroach and it would be difficult, if not impossible, to push it back. This has been seen in the coastal area in the east of Cyprus near Salamis, where once fertile areas have become completely barren due to encroachment of salt water into over-pumped wells.

Dr. D. L. Searle Having been interested in the geology of Limni mine since 1984, I am very pleased to add a few comments upon the geological nature of the orebody.

The larger copper-producing mines in Cyprus obtain their
Mining and milling operations at Limni mine, Cyprus

T. O. Trenerry A.C.S.M., C.Eng., F.I.M.M.
B. G. Pocock A.R.S.M., B.Sc., C.Eng., M.I.M.M.

Further contributed remarks and authors' reply to discussion* on paper published in Transactions/Section A (Mining industry), vol. 81, January, 1972, pp. A1–12

P. G. Petropoulos The authors are to be congratulated on their very comprehensive and interesting paper.

Anybody who is particularly interested in ancient mining may wish to refer to a brief account published in 1967.1 This was an historical paper and hardly touched upon the technical side of the subject.

Various publications have appeared on ancient mining in Cyprus, and these are supplemented or revised in the light of new discoveries as and when new evidence is brought to light. There is a very enlightening chapter on the antiquities in the mines of ancient Cyprus by J. F. H. Druce, in collaboration with C. P. Manglis and D. M. Creveling, which forms Appendix V of volume III of The Swedish Cyprus expedition.2 This publication covers a long historical account of the ancient copper industry in Cyprus and makes references, in particular, to writings by ancient historians who visited Cyprus, and contains much interesting technical information, such as photographs of timbers and tools used by the ancients relating to Mr. Bruce's findings at the mines worked by Cyprus Mines Corporation (Soli), of which Mr. Bruce was managing director at the time.

I give below a list of publications for those who are interested in this fascinating subject. Recently, several people from European universities and other bodies have shown an interest in the slags which are lying about the various sites of ancient mining activity, as well as in the probable method or methods of smelting and refining. I am sure that all those who are concerned here in Cyprus will afford every help to anyone who wishes to carry out research on this or any other subject concerned with mining.

Bibliography


Authors' reply

T. O. Trenerry and B. G. Pocock We would like to thank all who contributed to our paper and for the interesting comments and suggestions made. Particular thanks are due to Mr. T. H. B. Lawther for the invitation of the paper on our behalf.

On the question of drilling and blasting, raised by Mr. Bryan Earl (pp. A184–5), once the water has been blown out, direct charging of wet holes is possible provided that the water is (a) drill water or (b) rain water. If groundwater is present in any quantity, it has been found that in spite of increased priming, initiation of the ammonium nitrate is not possible.

Although a standardized drill set-up of, say, 20” is possible, the blast-hole drilling equipment available requires manual rod changing, and the nearer the rig is to vertical the easier it is for one man to handle the drill rods (35 in × 9 ft × 35 lb).

The advantages of a favourable vertical burst angle are fully appreciated, but when all the factors had been considered, it was decided to use 15° as a minimum angle on all benches up to 30 ft. This gives a satisfactory face angle for the excavators to work without creating any problem with toes.

The reason for changing to 20° for benches in excess of 30 ft was to provide a safer work angle for the excavators. A 20° angle has been used on the few occasions when 50-ft holes were drilled.

'Detonating relays' is indeed correct.

Down-the-hole iso-kinetic equipment has not been considered, but the relative merits of this system may be explored.

The standard method of precipitation and titration with sodium thiosulphate is used for all copper assays.

We consider this method the most accurate volumetric assay for copper, especially in low-grade samples. In this method titration must commence immediately the potassium iodide is added, as iodine in solution is appreciably volatile and low results will be obtained if titration is delayed.

For copper concentrate sales, samples are exchanged with the buyer and the splitting limit is 0.3 per cent copper. Over a number of years our results, on average, will vary from those of the buyer from between 0.0 per cent Cu and 0.2 per cent Cu.

Sulphur is determined by precipitation as barium sulphate, air-drying and weighing. In this case the splitting limit with the buyer is 0.5 per cent S. This assay method has proved to be sufficiently accurate and to be well within these limits.

Early (1947) exploration drilling, queried by Professor R. N. Pryor (p. A195), was carried out with churn drills and a 100 ft × 100 ft grid pattern. The intention was to drill through the mineralized zone. As Mr. Lawther indicated (p. A195), most of these holes were drilled to depths of between 300 and 500 ft; however, many did stop at approximately 150 ft, still in good mineralized ground. It is understood that due to the old underground workings drilling conditions were difficult.

When mining out the orebody, it was found that the ore reserves bore little relation to the actual ore, and this may be attributed to a number of reasons. (1) Some holes stopped in good mineralized ground and were just not deep enough. This was normally due to excessive caving or the hole penetrating underground workings and being forced off line. (2) Holes passing through high-grade bands were stopped low mineralized zones, giving false reserves. (3) Owing to the complex nature of the orebody, the 100-ft grid was just not close enough.

The last exploration drilling done was on a 50 ft × 50 ft grid, and in certain conditions, where interpolation between holes was not possible, the grid was closed up to 25 ft × 25 ft.

Mining for the past year has shown that the ore-reserve limits are accurate, but that the values obtained from Halco drilling samples are approximately 5 per cent lower than those obtained for the mill feed.

Professor M. G. Fleming (p. A195) raised the question of the capacity of the primary rubber-lined mill. This was the main factor to limit the throughput of the mill at Limni, and, as such, it was studied very closely. There were, however, a number of variables which complicated the comparison between steel and rubber linings on this operation. The grading characteristics of the ore changed to a predominantly harder type of primary ore from a more soft, secondary type, which had been affected by Roman excavations. Equipment changes in the crushing plant undoubtedly changed the size distribution of the rod/mill feed. The proportion of slimes in the mill feed also varied, affecting the tonnage fed to the rod/mill. These variables tended to complicate any accurate estimation of change in mill capacity. On balance, however, there appeared to be little or no reduction in capacity due to the introduction of rubber linings.

We were interested in Mr. J. E. Denevan's comments (p. A195) on broken rods being driven not only through the rubber lining but through the shell of the mill. This difficulty could have been overcome by removing worn rods, as we did at Limni. The only way to ensure the complete removal of this size of rod was to remove the charge entirely from the mill and replace it with new rods.

With regard to reopening of the lower adit for drainage purposes, raised by Mr. C. Harvey Richards and Mr. W. Brynmor Davies (pp. A196–8), two pilot holes were drilled through the concrete plug, and as these only showed a trickle of water, it was considered safe to remove the plug by stages. The plug was completely removed and the adit was found to be full of mud for a distance of at least 50 ft to within 1 ft of the back. As virtually no water is escaping, it is thought that the blockage could of a considerable length. The adit is 2000 ft long and the plug was situated at 570 ft from the portal. A borehole from the surface to check the extent of the blockage would in our case be difficult, as the area is well covered by waste dumps.

On the other points raised by Mr. Richards, labour problems did arise due to political unrest, and work had to be curtailed at times. The years 1964–65 were the worst, as is clearly shown by the production figures. The past few years have gone smoothly, working a mixed labour force of 8 Greeks to 3 Turks, and although the majority of Turks are employed in the mine, it is a much more integrated labour force today.

It may be of interest to note that since the paper was published a few days, 40–h week is now worked, and in addition there has been a 25 per cent increase in the figures given for wage rates (Table 5, p. A8).

During 1971 the power consumption and cost per ton averaged 19·79 kWh per ton milled and 1·90 kWh per ton crushed—at a total cost of 17·0 p/ton. Perhaps the fairest comparison of cost per ton treated is shown at the time of changeover.

1965—17·2 p/ton
1966—16·4 p/ton (50 per cent National Grid)
1967—14·2 p/ton (100 per cent National Grid)

In spite of yearly increases, the present cost per ton of 17·0 p compares favourably with the 1965 figure. With our own power station as standby, mill downtime has been virtually eliminated.

The natural pH of the ore is still around 5, and the lime consumption is still relatively high at 15 lb/ton. There were occasions, in treating material badly affected by oxidation, particularly in areas associated with old Roman workings, when lime additions of up to 60 lb/ton were normal.

The shearing resistance of the deposit (Mr. Davies, p. A197) was noticeably increased as drainage and pumping lowered the water-table. Unfortunately, no accurate measure of this was obtained, but it may be said that faces which were moving on a 25–30° slope are now stable at 40–45°.

Fig. 1 shows the type of notch commonly used by the Romans in Limni. The set appears to be unnecessarily elaborate by today’s standards and must have involved considerable work. Even though poles of varying size were used, the notches generally fit together surprisingly well, suggesting the use of a template. No timber heavier than 6-in diameter has been encountered by us.

The figures requested by Dr. D. L. Searle (pp. A196–7) are tabulated below.

<table>
<thead>
<tr>
<th>Year</th>
<th>Ore mined, Grade, %</th>
<th>Overburden removed, yd³</th>
<th>Overburden/ore ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>1961</td>
<td>277 247</td>
<td>2·03 23·0</td>
<td>807 922</td>
</tr>
<tr>
<td>1962</td>
<td>320 316</td>
<td>2·00 23·6</td>
<td>1 019 854</td>
</tr>
<tr>
<td>1963</td>
<td>400 822</td>
<td>1·59 19·9</td>
<td>1 058 642</td>
</tr>
<tr>
<td>1964</td>
<td>253 088</td>
<td>1·56 19·94</td>
<td>800 485</td>
</tr>
<tr>
<td>1965</td>
<td>380 081</td>
<td>1·27 17·25</td>
<td>1 290 180</td>
</tr>
<tr>
<td>1966</td>
<td>413 018</td>
<td>1·12 15·20</td>
<td>1 661 826</td>
</tr>
<tr>
<td>1967</td>
<td>430 640</td>
<td>1·11 14·98</td>
<td>1 523 046</td>
</tr>
<tr>
<td>1968</td>
<td>455 825</td>
<td>1·09 16·35</td>
<td>1 982 268</td>
</tr>
<tr>
<td>1969</td>
<td>438 487</td>
<td>0·98 13·06</td>
<td>2 153 123</td>
</tr>
<tr>
<td>1970</td>
<td>452 488</td>
<td>0·96 12·78</td>
<td>1 941 489</td>
</tr>
<tr>
<td>1971</td>
<td>462 484</td>
<td>0·89 11·97</td>
<td>1 862 493</td>
</tr>
</tbody>
</table>

Finally, we would like to thank Mr. P. G. Petropoulos for his interesting comments.