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REVIEW OF COPPER ~~SMELTING~~ OPERATIONS
Extraction

by

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SUMMARY

Background

In 1965 the South Australian Government Department of Mines authorised Amdel to carry out a review of lead and copper smelting operations. This decision was undoubtedly prompted by the proving of considerable reserves of lead ore at Ediacara, and the exploration work for copper proceeding in South Australia.

Objective

It was desired to have readily available technical and cost data for the smelting and refining of ores and concentrates of these metals. This information would assist the Department of Mines in assessing newly-discovered lead and copper occurrences.

Summary of Work Done

This report is the concluding report of a series of three, the two previous being:

Amdel Report 398	"Review of Lead and Copper-Smelting Operations — Sale of Concentrates"
Amdel Report 451	"Review of Lead Smelting Operations".

In this present report dealing with copper extraction the published literature has been searched and a bibliography of over 400 references is given in Appendix B. The extraction of copper from both sulphide and oxidised ores has been reviewed and the fixed capital investment and the direct treatment cost estimated for most of the processes.

Conclusions

It will be appreciated that there is no presently-known copper ore body in South Australia which can be used as an example. The information should prove useful when considering the treatment in South Australia of ore from another State, and in evaluating new discoveries in South Australia.

1. INTRODUCTION

Copper ores can be broadly classified as either sulphide or oxidised, according to the copper minerals present. Sulphide ores contain the minerals chalcocite (Cu_2S), chalcopyrite ($CuFeS_2$), bornite ($Cu_2S.CuS.FeS$), tetrahedrite ($4Cu_2S.Sb_2S_3$), enargite ($3Cu_2S.As_2S_5$), covellite (CuS), and copper-bearing iron sulphide. Oxidised ores contain malachite ($CuCO_3.Cu(OH)_2$), azurite ($2CuCO_3.Cu(OH)_2$), brochanite ($CuSO_4.3Cu(OH)_2$), chrysocolla ($CuO.SiO_2.2H_2O$), and cuprite (Cu_2O) (Dennis, 1961).

Both sulphide and oxidised ores have been mined in South Australia, and mining companies are actively engaged in the search for copper at the present time. There is however, no major deposit which can be taken as an example for the purpose of this report. Among the prospects and areas of past activity are Yudnamutana, Burra, Wallaroo-Moonta, Kapunda and Callington. These are shown in Figure 1.

While this report deals primarily with Copper Smelting, all methods of extracting copper from its ores are included in the review of processes, whether pyrometallurgical or hydrometallurgical. Emphasis is placed on new developments.

In order to be able to estimate the cost of raw materials and utilities, it is necessary to decide on a location for the smelter or chemical treatment plant. Wallaroo and Port Adelaide will be selected. Wallaroo would serve the Wallaroo-Moonta district, having shipping and rail facilities and superphosphate works requiring sulphuric acid. The Port Adelaide-Birkenhead area would be a logical point for concentrates from Kapunda and Callington, and has rail and shipping facitilities as well as sulphuric acid manufacturers and users.

It is clear that, at its present status, the extractive metallurgy of copper can no longer be reviewed as a limited number of well-defined routes each proceeding in well-tried stages from ore to metal. Consecutive stages are being integrated and techniques from different routes married together to suit the particular applications. Automation is finding its way into most plants. Mathematical models have been developed by Kennecott Copper Corporation for a reverberatory furnace and for a converter, thus paving the way for computer control.

2. AVAILABLE METHODS OF PRODUCING CRUDE METAL

2.1 Smelting of Sulphide Concentrates

2.1.1 Roasting, Smelting and Converting

The roasting, smelting and converting sequence for the production of blister copper from sulphide concentrates is a continuous process of oxidation which can be represented diagrammatically as illustrated in Figure 2.

Roasting. The object of roasting is to eliminate sulphur so that iron is converted into an oxide which will be slagged by silica. Increased elimination of sulphur will increase the copper content of the matte, but will also lead to magnetite formation and increased copper loss in slag, so that a compromise is necessary.

Roasting is generally carried out in mechanical hearth furnaces whose design is based on the original McDougal furnace. Twenty-five feet diameter, 7-hearth modified McDougal furnaces are used at Anaconda (Day, 1965) to roast a portion of the sulphide concentrate which is then combined with wet raw concentrate for charging to the reverberatory furnaces. This practice eliminates the dust problem associated with roasting of the total flow. Asano and Nojima (1962) describe the operation of Herreshoff roasters in Japan.

Fluid-bed roasting is practised at the Tennessee Copper Co. (Anon, 1962(2); Anon, 1964(12); Blair, 1965). There it is stated to have increased the capacity of the reverberatory furnace and converter and made possible the recovery of a large portion of the sulphur lost in direct smelting. Wet concentrate is mixed with water and sand to produce a slurry containing 78% solids which is sprayed into the roaster operating at 620^{0}C. A 15% SO_2 gas is produced for acid manufacture and the partially-roasted calcine is charged to the reverberatory furnace by means of a Wagstaff gun.

As the copper content of concentrates increases, and the iron content decreases, roasting becomes less necessary, and a number of new plants have eliminated roasters and charge wet concentrate directly to reverberatory furnaces (Morenci, Arizona; White Pine Copper, Michigan; Chile Copper Co., Chuquicamata; Kennecott Copper Corporations' modernisation of the original Garfield Smelter, Utah (Anon, 1965); Andes Copper Mining Co., Potrerillos (Anon, 1965(2)).

Smelting – Blast Furnace. While oxidised ore can be smelted with coke and flux in a blast furnace to yield metallic copper, sulphide ores require roasting and sintering. "Pyritic" smelting, in which pyrite acts as the fuel and matte is tapped from the furnace, was practised at Mt Lyell for many years. In the Orkla process coke is added and the top of the furnace is sealed by a bell so that sulphur oxidation is prevented. Approximately 80% of the sulphur content of a pyritic ore is recovered as elemental sulphur.

Generally speaking, blast furnace processes for copper smelting have lost favour because most ores are now concentrated by flotation and the fine concentrate would require agglomeration. Mt Lyell do in fact charge wet flotation concentrates to a blast furnace and lose approximately 15% from the furnace as dust. However, Momoda (1959) has described a side-flue blast furnace in operation in Japan which makes use of the heat of oxidation of the iron and sulphur in the concentrates and yields a gas suitable for acid manufacture. It is understood that there is a possibility of such a furnace being built in Europe.

Smelting – Electric. Barth (1961, 1962) has reviewed the electric smelting of sulphide copper concentrates. This is practised in Norway, Finland and Sweden, with one plant in Austria and one in Uganda. There are many similarities to reverberatory smelting but electric smelting differs in that heat is developed in the slag layer and is transferred downwards to the matte and upwards to the floating charge and thence to the gas. The matte in the electric furnace is thus hotter than in the reverberatory furnace. Based on the operating furnaces, which with one exception, smelt 26-28% Cu concentrates to 30-40% Cu mattes, Barth quotes the following approximate energy requirements:

Size of Furnace	Kind of Charge	kWh/metric ton of Charge
Small	Unroasted, wet, cold	700
Large	Unroasted, wet, cold	600
Large	Roasted and unroasted wet, cold	500
Large	Roasted, hot	400

The capital costs of an electric furnace and a reverberatory furnace are comparable, but the ancillary equipment for the former is more expensive. Kershanskii et al (1961) have described Russian electric-smelting trials.

Smelting - Reverberatory. The roaster calcine or wet raw feed is charged to reverberatory furnaces, commonly over 100 ft long by 30 ft wide. Converter slag is also poured in through a spout. The furnace roof is generally a sprung arch of silica bricks or a suspended arch of magnesite bricks. At the charging end the furnace is fired with oil or powdered coal and at the tapping end matte and slag are drawn off through separate tapholes.

Reverberatory furnace practice has been reviewed by(inter alia) Johnston (1959), Evans (1960), Verney (1960), Anderson (1961), Fowler (1961), Gibson (1962), Bogert (1964)(1)(3), and Day (1965). The use of oxygen in the reverberatory furnace has been discussed by Maklovsky et al (1961), Davidson (1962) and Kupryakov and Miller (1964), who are strongly in favour of oxygen enrichment of the blast. Beals et al (1965) have patented the injection of oxygen into the matte in the reverberatory furnace. This is simply a partial converting operation carried out in the reverberatory furnace.

Converting. Matte is charged to the converter where it is blown with air to oxidise ferrous sulphide to oxide, which is slagged by added silica. This is the first stage of converting and is strongly exothermic, the cuprous sulphide remaining after pouring off the slag being known as "white metal". In practice this stage is repeated a number of times – more matte and flux are charged and blowing continued – until the converter is full of white metal. The whole charge is then blown to blister copper in the second stage of the process.

The squat upright "great Falls" converters were obviously derived from the Bessemer converter, but with tuyeres in the side instead of the bottom. Perettà (1948) has given a physico-chemical explanation of the failure of bottom-blowing applied to copper converting. While there are isolated stationary converters, the horizontal Peirce-Smith converter is now almost universal, ranging in size up to 35 ft long by 13 ft diameter. Automatic tuyere-punchers are reducing labour costs at a number of plants (Nicholson et al, 1961; Anon, 1965(4)). Converter practice has been reviewed by (inter alia)Lathe and Hodnett (1958, 1959), Iolko (1959), Shalygin and Meierovich (1960), Anderson (1961), Turnbull (1962), and Day (1965).

Oxygen Enrichment of the Blast. A number of workers have investigated oxygen enrichment of the blast, e.g. Kurzinski (1958), Iolko (1959), Maklovsky et al (1961), Tsurumoto (1959, 1960, 1961) and Ramachandran (1965). Arentzen (1962) reports tests using 13 ft by 30 ft Peirce-Smith converters at Anaconda, Montana in which air was enriched to as much as 44.2% oxygen. The aim was to determine whether with oxygen enrichment three converters could be made to do the work of four blown with air. Experimental difficulties make it impossible to draw conclusions apart from the fact that much more cold scrap was required. Pobedonostsev et al (1960) carried out trials with 24.9% oxygen in the blast. The blast consumption was lowered, the converter capacity increased, and less magnetite was formed in the slag. This facilitated retreatment of the slag in the reverberatory furnace. Yakushin et al (1961) at the Irtysh copper smelter raised the oxygen content to 25.3%. They found that the life of a chrome-magnesite lining was reduced and the grade of the blister was lowered. Diev et al (1958, 1959, 1960) obtained an improvement in output and efficiency when using oxygen-enriched blast. They claim that the use of oxygen reduces the cost of building a copper smelter 25-40%, and that with an oxygen content of 40% in the blast the operating costs in converting are "4 to 5 times lower" than when using air.

It has been the practice, at the Hitachi Smelter in Japan, to charge raw concentrate to a converter which then carried out a combined smelting and converting operation using oxygen-enriched air (Tsurumoto, 1959, 1960, 1961). Trials along these lines have also been carried out in the USSR (Kolarov et al, 1964).

Flotation of Converter Matte. The International Nickel Co. of Canada have pioneered a mineral-dressing interlude to separate copper and nickel in converter matte. The copper-nickel matte is blown to oxidise as much iron as possible while still retaining cobalt and sufficient sulphur to avoid more than 10% metallics in the solid matte. Slow cooling produces a coarse structure and grinding and flotation yields separate copper and nickel concentrates (Sproule et al, 1961).

2.1.2 Suspension Smelting

Autogenous Flash Smelting. The exothermicity of the oxidation of chalcopyrite and associated sulphides is an incentive to not only making the roasting self-supporting heatwise, but also utilising the heat of oxidation to carry out the smelting. Outokumpu Oy in Finland and the International Nickel Co. of Canada both solved this problem, slightly differently.

Outokumpu Process. Dried concentrates and flux,and air preheated to 500^{0}C are injected down a vertical shaft (Fig. 3). The showering particles of concentrate partially oxidise extremely rapidly producing sufficient heat to melt into droplets which collect in a settler at the base of the shaft while the hot gases are removed by an uptake. The gases pass through a waste-heat boiler, heat exchanger and electrostatic precipitator before being used for sulphuric acid production. In the settler matte and slag separate. The matte is tapped and converted conventionally. The furnace at Harjavalta treats 500 tons of concentrate per day (Bryk et al, 1958). The Outokumpu-type furnace built at Ashio, Japan (Okazoe, 1956) has since been rebuilt and modified (Okazoe et al, 1965) and is at present smelting approximately 500 tons per day of concentrate containing 20-23% Cu, 24-28% Fe, 23-26% S. Concentrates with Cu:S of 0.6-1.2 can be smelted, sulphur recovery has reached 91% and the copper content of granulated slag has been as low as 0.47%.

Inco Process. Practically pure oxygen is used at Copper Cliff to smelt 1000 tons of concentrate per day in the furnace shown in Figure 3. Concentrate and flux, metered by gravimetric feeders, are injected into the furnace by oxygen through two burners at one end of the furance. At the other end, where the products are tapped, the slag is continuously cleaned by burning a pyrrhotite concentrate. The matte assays about 45% copper-nickel and is blown in converters. The gas averages 75% sulphur dioxide, the balance being essentially nitrogen. The gas is passed through scrubbers and a precipitator before being dried and compressed to produce liquid sulphur dioxide which is transported to sulphite pulp mills within a 400 mile radius. (Inco staff, 1955).

Russian workers, notably Penzimonzh et al, (1960, 1961); Deev et al (1960), Kochnev et al (1960), Bochkarev et al (1962), and Chalov and Lipin (1962) have discussed the use of oxygen in the flash smelting of copper concentrates.

Further designs of flash smelting furnaces have been tried by Lange and Barthel at Freiberg (Lange,1960), and Penzimonzh et al (1960, 1961) at the Balkhash copper smelter in Kazakhstan. Penzimonzh et al injected the concentrate upwards at 45^0 in preheated or oxygen-enriched air. Kochnev et al (1960) operated an experimental furnace in which concentrate was injected in a vortex of oxygen-enriched air through a burner directed slightly downwards and advocated the construction of a 200-500 tpd semi-industrial unit in the Urals or in Kazakhstan. This furnace might be better classified as a cyclone furnace.

Cyclone Smelting. The concentrate is introduced into a cyclone chamber where it is rapidly smelted in a whirling blast of preheated air. The molten droplets of matte and slag collect in a settler below the cyclone chamber. A large amount of experimental work has been done in Russia (Tonkonogii, 1956, 1957, 1960; Schulzinger and Tsyganov, 1957; Karlyshev, 1964; Onaev et al, 1964, 1965) and also by Lange and Barthel at Freiberg (Lange, 1960). Holy (1964) has published a design of an industrial cyclone furnace based on pilot-plant data.

In spite of the experimental work, it does not appear that any commercial-scale cyclone furnaces have yet been built for copper-smelting.

2.1.3 Continuous Processes

There are many disadvantages associated with the batchwise operation of copper converters and a continuous operation would be expected to have a higher throughput, lower labour requirement, longer equipment and refractory life, constant gas tenor, and be more amenable to accurate control. Based on the work of Diomidovskii et al (1959) an experimental continuous converter was built at Kosice, Czechoslovakia (Holeczy et al, 1962(1)(2), 1963(1)(2)(3), Sehnalek et al, 1964). It was shown theoretically that it is quite feasible to have three phases in the converter (slag, white metal and blister copper) and to feed matte and air into the white metal phase to constitute a continuous process. Difficulty was experienced with the build-up of solidified splashings on the tip of the lance, but this was overcome and the process operated apparently satisfactorily on the pilot scale. Shalygin and

Diomidovskii (1963) have also described continuous converting trials using top-blowing with oxygen-enriched air.

Worner (1964) has patented a continuous smelting process which is stated to be applicable to copper. Brittingham (1965) describes in general terms a continuous smelting process based on the Outokumpu flash smelting concept. His design consists of two flash smelting shafts superimposed on a settler, with an offtake for gases between the two shafts. Copper sulphide concentrate is smelted in one shaft and the matte tapped continuously from this end of the settler to a converter. At the other end of the settler the secondary reaction shaft smelts a pyrite concentrate which decopperizes the slag which is tapped from that end of the furnace to waste or an iron-recovery process. This principle of slag cleaning is that used in the Inco process.

2.2 Hydrometallurgical Extraction from Sulphide Concentrates

2.2.1 Roasting, Leaching, Electrowinning

Sulphide concentrates can be oxidised to convert copper to sulphate or oxide. Control of temperature and atmosphere are necessary, and over-heating results in the formation of difficulty-soluble copper ferrite. A fluidised-bed reactor with its accurate temperature control is the best unit for this operation. The calcine is then leached with water or dilute sulphuric acid (which may be spent electrolyte) and the solution purified and electrolysed using inert anodes.

A 14 ft diameter 80 tons per day fluid-bed roaster was started up at the Jadotville Shituru plant of Union Miniere du Haut Katanga in 1955 to roast chalcocite-carrollite concentrate. Two more reactors, 16 ft diameter, were installed at the Kolwezi Luilu plant in 1962 (Grothe and McLeod, 1957, Theys and Lee 1958, Theys 1958, Thoumsin and Coussement, 1964). The last authors discuss the chemistry of the roasting reactions.

At the Dowa Mining Co's Kosaka smelter a fluid-bed roaster treats 80 tons per day of chalcopyrite-covellite-pyrite-sphalerite-marmatite concentrate from which copper and zinc are recovered electrolytically (Kurushima and Tsunoda, 1955; Grothe and MeLeod, 1957). A similar roasting operation is carried out at Macalder Nyanza in Kenya Colony but the copper was initially recovered by cementation on scrap iron (Grothe and McLeod, 1957).

In 1964 a fluidised-bed roaster with a capacity of 15,000 tons per month of sulphide concentrate was started up at Nchanga Consolidated Copper Mines. An additional (No. 6) tankhouse was also commissioned to electrowin cathode copper from the pregnant solution (Anon, 1964(6)).

A fluid-bed roaster with a capacity of 100 tons per day should by now be complete at Chambishi in the Rhodesian Copperbelt (Anon, 1965(3)). The calcine will be leached and the pregnant solution electrolysed.

Pilot plant studies have been carried out at CSIRO Melbourne (Urie and Walkley, 1958, 1959; Whitehead and Urie, 1961) and at Batelle (Stephens, 1953). Work has also been done with a mixed copper-zinc-lead concentrate by Mechenov and Sotirov (1960). A fully integrated pilot plant was set up by the Bagdad Copper Corporation in Arizona to prove the whole process as applied to a chalcopyrite concentrate and a cement copper precipitate, (Howell et al, 1957; Anon, 1957(1); Grothe and McLeod, 1957, Anon, 1959 (1)). The results appeared good but the company has apparently abandoned the project. The cost of electricity for electrowinning in that location may have been the deciding factor.

Fursman (1962) describes sulphate roasting and water leaching of cupriferous and nickeliferous pyrrhotite from Funter Bay. Complex mattes may also be roasted and leached (Hsiao and Smirnov, 1961). This is done at Falconloridge Nickel Mines in Canada.

Sulphite Reduction. Okabe and Ito (1964) have recently reviewed the chemistry of copper precipitation by reduction of cupric sulphate with ammonium sulphite. They propose roasting a copper sulphide ore to sulphate, leaching, reduction of cupric sulphate in solution with ammonium sulphite, separation of metallic copper and recovery of ammonium sulphate. The advantages of the process are:

1. sulphur dioxide is available from the roasting step
2. pure copper is produced without electrolytic refining
3. the sulphur in the ore is almost completely recovered as ammonium sulphate.

2.2.2 Direct Leaching, Hydrogen Precipitation

A variety of reagents have been used to attack copper sulphide minerals.

Acid Leaching. Warren (1958) has investigated the pressure-leaching of chalcopyrite, chalcocite and covellite. Kuperman (1964) leached copper sulphide concentrates at 160^{0}C under oxygen pressure, extracting 97-99% of the copper and oxidising 90% of the sulphur to sulphates. Vasil'ev et al (1962) leached 200-mesh chalcocite with a mixture of 2% ferric sulphate solution and 5% sulphuric acid and found that ultrasonic vibrations gave a tenfold increase in rate of copper extraction compared with mechanical agitation.

Alkali Leaching. In the Sill process (Anon, 1957(5)) lean sulphide-arsenide ores are leached with caustic soda under oxidising conditions.

Ammonia Leaching. A large amount of work has been done, particularly by Sherritt Gordon Mines Ltd in Canada, on ammonia leaching of sulphide ores at elevated temperature and pressure. The process is used to extract copper from the Lynn Lake copper concentrate treated at Fort Saskatchewan and should by now be in operation at the plant of Marinduque Iron Mines at Mindanao in the Philippines. The concentrate, in which chalcopyrite is the predominant copper mineral, is leached with water, ammonia and air in horizontal agitated autoclaves. After thickening, the residue at the Marinduque plant was to be subjected to flotation to recover unleached pyrite for sulphuric acid manufacture and gold and silver extraction. The pregnant solution passes to distillation columns where excess ammonia is removed. Sulphur compounds in solution are converted to sulphate and ammonium sulphate is precipitated.

Hydrogen Reduction. The copper-bearing solution resulting from the process above is reduced with hydrogen in autoclaves precipitating metallic copper which is dried, pulverised and screened. The powder may be sold or further processed into strip. The Sherritt Gordon and Marinduque operations have been extensively described (Forward and Machiw, 1955; Brenthel, 1955; Anon, 1961 (numerous references), Argall, 1961, Starratt, 1961, Evans et al 1961, 1964).

The hydrogen reduction process is to be applied at Bagdad in Arizona where a $4 million plant, owned jointly by Bagdad Copper Corp. and Chemetals Corp. (which holds the process rights) is to come into production in 1966. Cement copper will be redissolved in sulphuric acid and precipitated with hydrogen at elevated temperature and pressure (see Section 3.3).

2.3 Oxidised Ores

2.3.1 Blast Furnace Smelting

As mentioned earlier blast furnace smelting to metal has lost favour with the exhaustion of reserves of coarse high-grade oxidised ores. Artamonov (1961) obtained considerable improvement in shaft-smelting of copper ore with coke by enriching the blast with oxygen.

2.3.2 Leaching and Cementation or Electrowinning

The lixiviants used for extraction of copper from oxidised ores are dilute sulphuric acid, ferric sulphate solution (which also dissolves sulphides) ammonia and ammoniacal ammonium carbonate solution. Sodium hydroxide has recently been introduced, dissolving oxidised copper minerals to form sodium cuprate.

Leaching can be carried out in situ by allowing lixiviant to percolate through unmined ore, e.g. ore remaining in a worked-out mine. In heap leaching lixiviant is applied to the top of the dumped ore and collected from below. Leaching can also be carried out in tanks or vats, either by allowing the lixiviant to percolate through the ore, or by agitating the suspension.

Leach-Precipitation-Flotation Process (L-P-F). Before listing in detail the various mines and concentrators where leaching is carried out, reference should be made to the L-P-F process, in Russia referred to as the Mostovich process. Sulphuric acid is added to the ground pulp containing oxidised copper minerals, which are thereby dissolved. Iron powder is added, causing the copper to precipitate as finely-divided metal which is recovered by flotation. If copper sulphide minerals are present these are recovered along with the metallic copper. This process is used at the Ray Mines Division of Kennecott Copper Corporation where the sulphuric acid and sponge iron are both produced from pyrite concentrated from the ore (Last et al, 1957; Anon, 1959(2), Franz, 1959). L-P-F is also used at the Morenci concentrator of Phelps Dodge Corporation (Beall, 1965). Bean (1960, 1961) has described L-P-F results obtained at Miami, but a recent review of the operations of Arizona copper producers (Beall, 1965) only refers to concentration of sulphide ore and leaching in situ at Miami. Mijoshi et al, (1960) has described the application of the L-P-F process to the Tsuchihata deposit. Sutulov (1963, 1964) has also discussed the process and refers to its use at Butte in the USA and at Almalyk in Russia.

Agitation Leaching and Electrowinning. Leaching and electrowinning is practised at Nchanga Consolidated Copper Mines, Northern Rhodesia (Chapman and Page, 1961). A low-grade oxide concentrate is agitated with spent electrolyte, the pulp thickened and the thickener overflow purified and fed to electrolytic cells. Cathode production in 1960 was approximately 6000 tons per month with a purity of 99.85%.

A new low-grade leach plant was built in 1962 to treat 60,000 tons per month of low-grade oxide concentrate, and the high-grade leach plant was extended. A fluid-bed roaster to render sulphide concentrates amenable to leaching was commissioned in 1964 (Anon, 1964(6)).

Agitation leaching is also carried out by Union Miniere du Haut Katanga in the Congo. The capacity of the plant is given as 8400 tons per month (Dennis, 1961).

Percolation Leaching and Electrowinning. Percolation leaching in vats is employed at the Chuquicamata plant of the Chile Copper Co. which was stated to have a capacity of 40,000 tpd of 2% copper ore, (Dennis, 1961). The grade has since fallen to 1% Cu (McArthur and Leaphart, 1961). The ore charged to the vats has the following size distribution:

	%
+ 1/2 in.	6
- 1/2 + 3/8 in.	15
- 3/8 in. + 6 mesh	38
- 6 +48 mesh	29
-48 mesh	12

At the new Chambishi Mines Ltd plant in the Rhodesian Copper belt (Anon, 1965) oxide ore is sized at 100-mesh, the minus 3/8in. plus 100 mesh material being leached by percolation vats, and the fines by agitation.

Heap or Dump Leaching and Cementation. Beall, (1965) has surveyed the copper mines and concentrators in Arizona and gives data for the mines using dump leaching, viz. Cananea, Copper Queen, Ray, Chino, Inspiration, Bagdad and Esperanza. Published figures for the percentage copper in the dumps range from 0.15-0.75% the pregnant solutions range from 0.2-3.6 g per litre and the iron consumption from 1.5-4 tons per ton of copper precipitated.

The lixiviant is a mixture of sulphuric acid and ferric sulphate. If there is sufficient pyrite in the dump, natural water can be used for leaching, the acid being formed by reaction with the pyrite. In other cases sulphuric acid must be supplied. Certain bacteria, Ferrobacillus ferro-oxidans and Thiobacillus ferro-oxidans, have the ability to markedly increase the rate of dissolution of copper.

The pregnant solution flows over scrap iron in precipitation tanks and the cement copper is sent to a smelter.

2.3.3 Segregation Process

This process is applicable to oxidised and mixed oxide-sulphide copper ores which are associated with a carbonate gangue, or ore difficult to concentrate by flotation. The moist crushed ore is mixed with a carbonaceous reducing agent and salt or some other Group IA or IIA halide and heated to 700-800^{0}C. Copper is presumably volatilised as chloride and reduced to deposit as metal on the carbon. Silver and gold report with the copper and the metals are recovered by flotation or ammonia leaching.

Pollandt and Pease (1959, 1960) and Pease (1963) have described the process as applied to silver-copper ores at Santa Lucia in Peru. Recoveries of 90% of the copper and slightly less of the silver from 1-2% Cu ores is reported (Ek and Mason, 1964).

The process is used at Akjoujt in Mauretania where an ore containing 3.1% Cu and 2 g Au per ton is upgraded to yield a 65% Cu, 60 g Au per ton concentrate recovering 85% of the copper and 72% of the gold (Rey, 1959, Ek and Mason, 1964).

Rampacek (1959, 1964) of the US Bureau of Mines has done considerable research on the process and the Bureau of Mines has developed a direct-fired kiln for the segregation stage (McKinney and Evans, 1963).

The process is used at the Lake Shore Mine in Arizona to treat a 1.8% Cu oxide-sulphide ore yielding a 50% Cu concentrate with 85% recovery of copper (Freeman et al, 1961; Ek and Mason, 1964). Arizona copper silicate ores are also amenable to treatment (Anon, 1960(1)).

Perlov et al, (1962) have also discussed the segregation process and Sutulov (1962) has assessed its applicability to Chilean ores. He quotes 1.8% Cu as the break-even grade for treatment under Chilean conditions and stated that a plant was to be built near Antofagasta.

3. REFINING OF CRUDE METAL

3.1 Fire Refining

Fire refining of copper consists in common practice of oxidising the molten blister by bubbling air through it to remove remaining sulphur, iron and zinc. After removal of the slag, the metal which at this stage has an oxygen content of approximately 0.9% is reduced by "poling". This has generally been done using green logs or "poles" pushed into the molten bath, reduction being accomplished by the hydrocarbon gases produced.

It has been shown that, while natural gas (predominantly methane) is not very effective as a reductant, the products of reforming natural gas are very satisfactory (Kuzell et al, 1959; Anon, 1961(2); Huttl, 1961(2); Klein, 1961, 1962; Komlev, 1964). The use of reformed natural gas greatly reduces the labour requirement for the poling operation.

Brantley and Schack (1962), suggested the use of a packed-column for contacting metal and reducing gas.

The use of gaseous ammonia for "poling" was advocated by Henych et al (1965). While technically feasible the cost of ammonia may eliminate it as a commercial process.

Several new developments in fire-refining have been investigated on a laboratory scale. Stolarczyk and Ruddle (1957, (1)(2)) used various slags to remove lead and tin from molten copper, and Ward and Hoar (1961), carried out electrolysis in which molten copper, under a layer of molten barium chloride, was the cathode. Oxygen, sulphur, selenium and tellurium in the copper were reduced to less than 0.001%.

Conventional fire-refining does not remove bismuth, nickel, selenium and tellurium, nor recover precious metals, and where these occur electrolytic refining must be used.

3.2 Electrolytic Refining

Blister copper is fire-refined and cast into anodes which are suspended in a copper sulphate-sulphuric acid solution (approx. 35 g per litre Cu, 150 g per litre H_2SO_4, temp. 50-60°C). The cathode is generally a specially-prepared sheet of electrolytic copper. Current densities range from 12-20 amp. per square foot and one ampere-day of current deposits only 1 oz of copper. Refineries are therefore very large and the amount of copper tied up is considerable. The 10,000 short ton per month refinery at Copper Cliff, Canada has 1216 depositing tanks and covers an area of 60 acres (Dennis, 1961). Jenkins and Saint-Smith (1961) have described the Townsville copper refinery. A new 3000-tons-per-month refinery has been built at Potrerillos in Chile (Anon, 1965(2)).

There are two systems in use for arrangement of the electrodes. The more commonly-used is the "Multiple" system. Cells are connected in series with all the anodes in a cell in parallel and all the cathodes (which begin as starting sheets) in parallel. In the "Series" system only the first and last electrodes in a cell are connected to the current. Between them are suspended the plates of copper to be refired, copper being removed from one side of each plate and deposited on the facing side of the next plate. A coating of resin separates the deposit from the crude metal. The series system is a low current-high voltage system, the copper output per kilowatt day being about twice that of the Multiple system. However, due to the greater tendency to current leakage, the necessity for special cell construction and less flexibility in operation there are only one or two refineries which use the Series system.

Besides descriptions by Liddell (1945), Butts (1954), and Dennis (1961) an informative survey by Schloen and Forbes (1961) and reviews by Jenkins (1962) and Levin and Nomberg (1964), specific plants have been described by Bauld et al (1956), Lockyer and Neller (1960) (Mufulira); Forbes (1956) (Canadian Copper Refiners Ltd); Franklin and Eigo (1960) (Kennecott, Baltimore); McArthur and Leaphart (1961) (Chuquicamata); Jenkins and Saint-Smith, (1961) (Townsville); Anon (1962)(1) and Lapee (1962) (Anaconda).

Henriksson et al (1959) have carried out experiments at Bolidens using channel cells which permit current densities up to 500 amperes per square metre (Levin and Mukhin, 1964). Despite lower building costs and inventory, Schloen and Forbes (1961) consider that this method will not be adopted immediately due to the demand for quality cathodes.

3.3 Chemical Refining

Bagdad Copper Corporation is building a $4m plant, jointly owned with Chemetals Corporation, to refine cement copper. The cement copper is redissolved in sulphuric acid and precipitated with hydrogen at 300°F and 450 psig pressure as high purity (99.9%) copper powder (Anon, 1964(2), Beall, 1965). The process is thus an offshoot of the Sherritt-Gordon process for which Chemetals Corporation are licensees. After sintering to remove traces of sulphur and oxygen the purity exceeds 99.95% copper.

4. ECONOMICS OF PROCESS

4.1 Producing Crude Metal from Sulphide Ores

4.1.1 Smelting Processes

In considering the pyrometallurgical processes applicable to smelting a sulphide copper concentrate in South Australia, the first governing factor is tonnage. The flash-smelting processes would probably only be justified by a high-volume operation. At Harjavalta and at Ashio the treatment rate is 500 tons per day of concentrate. The Inco process treats 1000 tons per day at Copper Cliff. Use of this latter process would involve construction of an oxygen plant and would call for a market for sulphur dioxide.

Since copper smelting in South Australia is rather hypothetical at the present time, two cases will be chosen for cost estimation:

1. large-volume smelting by the Outokumpu process,
2. small volume processing in a rotary reverberatory furnace.

Outokumpu Process. A treatment rate of 500 tpd of chalcopyrite concentrate containing 25% Cu will be assumed.

Fixed Capital Investment[a]
A flash-smelter of the same size as the plant at Harjavalta, with converting facilities is known to cost of the order of \$A7 million to build.

Direct Smelting Cost[a]
The smelting cost per ton of copper metal for a smelter at Port Adelaide or Wallaroo has been estimated as follows:

	Amount	Unit Cost, \$	Total Cost, \$
Raw materials: siliceous flux	2 tons	2	4
Labour: (assume 40 men on each of 4 shifts)	10 man hr	1.8	18
Supervision: (assume 10% of labour cost)			2
Maintenance: (assume 6% annually of fixed capital investment)	-	-	9
Plant Supplies: (assume 15% of maintenance)	-	-	1
Royalties and Patents (assume 1% of price of copper)	-	-	8

(a) For definitions of these terms see Appendix C.

	Amount	Unit Cost, $	Total Cost, $
Utilities:			
Fuel oil	0.1 ton	16	2
(the small cost of fresh and salt water will be compensated for by a small credit for power produced in excess of requirements)			
Direct Smelting Cost per ton of copper			44

It would be a pity to build such a plant without fully investigating continuous converting to make the whole operation continuous.

Reverberatory Smelting. The rotary reverberatory furnace ("kurztrommelofen" in Germany) is establishing itself in many braches of non-ferrous smelting, and can be used to smelt roasted or unroasted sulphide copper concentrates to matte (Schwartz, 1956). It will be assumed that the concentrates supplied are of sufficiently high grade to be charged direct to the rotary reverberatory.

It seems feasible that after smelting and tapping of slag the burner could be swung back, an air lance inserted, and the matte blown to white metal as is done at BHAS, Port Pirie, or possibly even to blister copper. It may well be best to use two furnaces, one for smelting and the other for converting but for a small throughput the following cycle is suggested, using only one furnace:

Smelting:

1. Charge: raw concentrate
   "converter" slag from previous converting stage
   flue dust
   fluxes.
2. Smelt
3. Remove reverb. slag and discard
   Charge: more concentrate
   fluxes.

   (Repeat until furnace full of matte)

Converting:

4. Remove "reverb" slag and discard
5. Charge fluxes
   Retract burner, insert air lance and commence blowing
6. Remove converter slag
   Charge: more concentrate
   fluxes

   (Repeat until furnace full of white metal)
7. Remove converter slag
8. Blow to blister copper and cast.

Schwartz quotes a daily throughput of 25-30 tons for a 3 metre diameter by 3 metre long rotary furnace smelting copper concentrates to matte. If blowing is carried out in the same furnace the capacity might be reduced to 10-15 tons per day, i.e. 3 tons blister per day, 1000 tons copper per year.

We will thus assume a treatment rate of 12 tons of concentrate (3 tons blister) per day.

Fixed Capital Investment

We will assume that the plant consists of:

i. fairly simple ore and flux handling equipment, e.g. a tractor-shovel for unloading railway trucks and filling a charging hopper,

ii. a 3 m x 3 m rotary reverberatory furnace with oil-storage and heating facilities,

iii. air blower,

iv. a baghouse, fan, and stack to disperse sulphur dioxide,

v. moulds for casting blister cakes,

vi. slag granulation and disposal facilities,

vii. a small laboratory, maintenance workshop and administrative office.

The purchased equipment cost is estimated very approximately as follows:

	$
Tractor-shovel	5,000
Charging hopper	2,000
Rotary reverberatory furnace with ancillary equipment	60,000
Electric crane	20,000
Oil storage	5,000
Air blower	8,000
Baghouse, fan and stack	20,000
Blister casting and slag granulation and disposal equipment	20,000
	140,000

Using Lang's factor of 3.63[a] the

Fixed Capital Investment becomes $0.5 million

(a) See Appendix C.

Direct Smelting Cost

We will assume a tractor driver, 3 furnace attendants, 1 spare man, a maintenance fitter and a foreman on each shift, with a manager, 2 clerks, 1 analyst and 2 tradesmen on day-shift. An estimate of the direct smelting cost per ton of copper is set out below:

		Amount	Unit Cost, $	Total Cost, $
Raw Materials:				
silica sand flux		2 tons	2	4
Labour:				
operating	(4 men/shift)	32 man-hr	1.8	58
supervisory	(1 man/shift)	8 man-hr	2	16
maintenance		12 man-hr	1.8	22
Materials (maintenance)		-	-	5
Plant supplies (15% of maintenance)		-	-	4
Utilities:				
fuel oil (14% of conc. wt.)		0.6 tons	16	10
electricity and water		(say)	-	3
			Direct Smelting Cost	$122

4.1.2 Roasting, Leaching, Electrowinning

Fixed Capital Investment. Based on the Bagdad pilot plant which treated 5 tons of concentrate per day, budget estimates were made for a plant producing 17,000 tons of copper per year. The cost of each item of equipment was calculated and the purchased equipment cost found to be $1,700,000 from which the fixed capital investment was estimated at $6.9m.

Direct Treatment Cost. It will be assumed that this plant would employ 4 shifts of 24 men each. The power requirement for electrowinning is based on Nchanga data (1.05 kWh ac per lb of copper).

Direct Treatment Cost per Ton of Copper

	Amount	Unit Cost, $	Total Cost, $
Labour	32 man hr	1.8	58
Supervision (assume 10% of labour)	-	-	6
Maintenance (assume 6% annually of fixed capital investment)	-	-	24
Plant supplies (assume 15% of maintenance)	-	-	4
Royalties and Patents (assume 1% of sales price of copper)	-	-	8
Utilities:			
Power-roasting	200 kWh	0.016	41
leaching	25 kWh	0.016	
electrowinning	2350 kWh	0.016	
Water	1900 gal	-	-
Fuel: roasting, heating electrolyte	0.08 tons	16	1
			142

It must be borne in mind that the product is electrolytic copper and would not need further refining.

4.1.3 Ammonia Leaching, Hydrogen Precipitation

Capital and operating costs will simply be quoted for the plant designed for Marinduque Iron Mines. This plant was designed to treat the following concentrates:

	Sipalay 35,000 tpy	Bagacay 40,000 tpy
Cu	25	14.5
Fe	30	29.0
S	35	41.0
Zn	-	15.5
Insoluble	10	-

and to produce:

semi-fabricated copper products	14,000 tons
ammonium sulphate	100,000 tons
zinc as oxide or metal, approx.	5,000 tons.

Taking zinc as equivalent to copper, the output of the plant can be taken as 19,000 tpy of copper.

Fixed Capital Investment. Published articles gave the cost of the Marinduque plant as $US 23 m or £stg 8 m, so that an Australian cost of $A20 m will be assumed. This is an expensive plant for an output of 19,000 tpy of copper but the actual copper-processing plant will be costly due to the necessity for high-pressure stainless steel equipment, and this investment will also include:

i. an electrolytic hydrogen plant

ii. an ammonia plant

iii. a pyrite-burning sulphuric acid plant

iv. a loading pier.

Direct Treatment Cost. If ammonia (27,000 tpy) were purchased at $200 per ton and ammonium sulphate (100,000 tpy) sold at $50 per ton the net cost to the process would be $21 per ton of copper. However, the Australian market might not absorb this quantity of sulphate in which case it would be necessary to regenerate ammonia for re-use by means of lime. The use of 64,000 tpy of hydrated lime at $20 per ton would add $65 per ton to the direct treatment cost of copper.

No further analysis of the economics of this process will be given, since while it produces copper as premium-grade powder, the capital investment and treatment cost would only be warranted if the ore were complex and there were favourable markets for the products.

4.2 Treatment of Oxidised Ores

4.2.1 Leaching

The capital investment and treatment cost will depend greatly on the type of operation — whether agitation leaching percolation leaching or heap leaching. Costs for agitation leaching and electrowinning could be estimated from data for Nchanga (Chapman and Page, 1961). This plant produced 6245 tons of cathode copper in July 1959 (say, 75,000 tons per year) employing a direct labour force of 311 men. This leads to an estimate of 9 man hours per ton of copper for operating labour.

Percolation leaching in vats is employed at Chuquicamata which in 1961 had a capacity of 41,000 short tons per day of 1% copper ore.

There are a number of dump-leaching operations in Arizona from which approximate costs could be estimated.

4.2.2 Segregation Process

Pollandt and Pease (1960, Discussion) itemised treatment costs for the segregation process as applied in Peru. At 100 tons per day the direct treatment cost per metric ton of ore in September, 1960 was £stg 2-4-10 ($A5.60); for 300 tpd the cost was £stg 2-0-10 ($A5.10). The ore contained 1.5% Cu and 5-20 oz Ag per ton and required 2-3% coal addition because of its manganese dioxide content, whereas an ore with inert gangue would require only 1% coal per ton of copper produced. This treatment cost would be of the order of $A400 with a credit of about $A10 for the silver content. The major costs were oil fuel, coal ($A31 per ton) and electric power, all of which would be fairly expensive in that location.

4.3 Electrolytic Refining

Fixed Capital Investment. Jenkins (1962) discusses the capital cost of electrolytic refineries and quotes a figure of £500 ($1000) capital investment per square yard of operating tank area. Operating intensity varies up to a maximum of about 30 tpy copper per square yard in large plants the average being 24.27 tpy per square yard. Hence a figure of $40 of fixed capital investment per annual ton of copper will be assumed.

Direct Treatment Cost. Jenkins also estimates the operating labour requirement in a tankhouse as 0.03 man per square yard of tank area. This gives an operating labour cost of say, 0.03 by 3600 equals $108 per square yard per year or $4 per ton of copper.

The copper produced per kilowatt-day at a current density of 18 amperes per square foot ranges from 165-180 lb in the multiple system and from 340-380 lb in the series system (Liddell, 1945). The cost of power at 1.6c per kilowatt is thus $5.0 per ton for the multiple system and $2.5 per ton for the series system.

Levin and Nomberg (1964) in their review of world electrolytic plants give some interesting analyses of costs. Experience at Montreal indicated that labour costs increased with increasing current density (2.26 man hours per ton at 211 amp/m^2, 2.64 man hours per ton at 285 amp/m^2). However gradual increase of current efficiency (combined, admittedly, with increased mechanisation) has decreased labour costs at the Pyshma plant in the USSR.

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APPENDIX A

SOME REAGENT COSTS

Ammonia

The price of "liquid ammonia" (25% NH_3) in bulk tankers in quantities greater than 200 tons is quoted by ICIANZ as $200 per ton of NH_3 ex Deer Park. Anhydrous ammonia in quantities greater than 200 tons per year is quoted at $180 per ton ex Deer Park.

Ammonium Sulphate

ICIANZ quote $65-75 for grade SA1 in bags in 10 ton lots ex wharf Australian ports.

Sulphuric Acid

Adelaide Chemical and Fertilizer Co. Ltd quote $26.40 per ton for quantities of the order of 500 tons per year FOR Port Adelaide.

APPENDIX B

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ZUBAREV, V.I., (1959), Some Problems of Development of Copper Smelting Industry of Central Kazakhstan, Tsvetnye Metally 32, (3), 1-4, March. Performance of Balkhash Mining-metallurgical combine: Dzhezkazgan combine and Boshchekul ore treatment plant.

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ANON, (1960)(1), Arizona Copper Silicates Respond to Segregation; Transarizona Resources, Eng. and Min. J. 161, 86-7, Nov.

ANON, (1960)(2), Continuous Casting Machine for Copper Billet and Slab Production, Metallurgia 61, 101-2, March.

ANON, (1960)(3), Continuous Casting Machine for Copper Billet and Slab Automation 7, 99-100, Feb.

ANON, (1960)(4), Kennecott Copper Plans Smelting Experiments, Eastern Metals Review, 13, 879, Aug. 21.

ANON, (1960)(5), New Continuous Casting Machine for Copper has Improved Features, Iron and Steel Eng. 37, 147-8.

ANON, (1960)(6), New Copper Refinery with New Ideas, J. Metals 12, 470-1, June.

ANON, (1960)(7), Smelter Highlights, Eng. and Min. J. 161, (4), 108-11. Principal features of Toquepala Plant in Peru, including right-angle design of plants to promote efficient flow-through of materials in process, and extensive use of materials-handling equipment; capacity of two 34 x 115 ft reverbs is 650 tpd of concentrates.

ANON, (1960)(8), Southern Peru Copper Corp. Starts Production at Toquepala, flowsheets, Eng. and Min. J. 161, 81-116, April.

BASINA, I.P., VDOVENKO, M.I., and KURMANGALIEV, M.R., (1960), Basic Results of the Study of Cyclone Processes of Smelting and Sublimation, Trudy. Inst. Energet., Akad. Nauk. Kazakh, SSR 2, 261-73. (CA 54 22238 c)

BEAN, J.J., (1960), Leach-precipitation-flotation at Miami: Analysis of Latest Operations, Min. Eng. 12, 1265-70, Dec.

BRIDGESTOCK, G., ELKIN, E.M., and FORBES, S.S., (1960), Operations of Canadian Copper Refiners Ltd, Can. Min. and Met. Bul. 53, (582), 773-87, Oct.
Plant at Montreal East, Ont., refines copper, silver, gold, selenium and tellurium; copper refining capacity is 236,000 tons/year.

BRITTINGHAM, G. J., (1960), Rehabilitation of Chang Hang Copper Smelter, South Korea, Instn. Min. and Met. Trans. 70, (1), 1-17 (discussion).

CARPENTER, S., et al, (1960), El Salvador Plant, Min. Eng. 12, (4), 356-70, 370D, 371-3, April.
Following group of papers includes Plant Construction and Procurement of Supplies, S. CARPENTER, 357-8; Introduction to Metallurgical Operations, L.O. FINES, 359; Crushing Plants, E.F. RAFFO, 361-4; Concentrator, L.O. FINES, A.H., SKOUBOE, H.G. DWYER, 365-370; Molybdenum Plant, J.S. OPKINS, I. HAUSER, 370A-B; Concentrate Handling, J.P. MANNING, 370C, 370D, 371; Smelter, P.K. ALDOUS, 372; Metallurgical Operations, L.O. FINES, 373.

DEEV, V.I., KOCHNEV, M.I., OKUNEV, A.I., and SERGIN, B.I., (1960), Distribution of Cadmium, Rare and Uncommon Elements Among Products of Oxygen Flash-Smelting of Copper-Zinc Concentrates, Tsvetnye Metally 33, No. 4, 19-24 (CA 55 1337b)

DIEV, N.P., et al, (1960), Use of Oxygen in Copper Smelting, Primenenie Kisloroda na Met. Predpriyatiyakh Urala, Sverdlovsk, Sbornik, 117-28 (CA 55 26921h)

EVANS, G.L., (1960), Copper Matte Smelting; Influence of Charge Composition Upon Heat Requirement, Inst. Min. and Met. Trans. 69, (5), 201-9, Feb.

FRANKLIN, J.W., and EIGO, D.P., (1960), Kennecott Opens Baltimore Refinery.
E Electrolytic copper refinery dedicated in Anne Arundel Country, Md. is designed to process 16,500 tons of refined copper per mo.

HEDLEY, N., and TABACHNICK, H., (1960), Cyanide Leaching to Extract Copper from Zinc Concentrate, Min. Eng. 12, 158-60, Feb.

KOCHNEV, M.I., et al, (1960), Fire-refining of Copper Using Oxygen-Enriched Air, (Primenenie Kisloroda na Met. Predpriyatiy KH Ural, Vserdlouk, Sbornik).

KOCHNEV, M.I., et al, (1960), Suspension Smelting of Ural Copper-Zinc Concentrates Using Oxygen Blasts, Tsvetnye Metally (Eng. Trans.) 1, 19-22, Oct. (CA 55 15267 d)

KUZELL, C.R., (1960), Development of Modern Copper Smelting, Extractive Metallurgy Division Lecture, Met. Soc. of AIME - Trans. 218(4), 578-84, Aug.
Ancient metallurgy of copper, Renaissance period in copper smelting; 19th century developments; advances in last 60 yr; future prospects.

LABINE, R.A., (1960), New Layout Streamlines Copper Refining, Chem. Eng. 67, (6), 134-7, March.
Basic electrorefining process as used by Kennecott Refining Corp., Baltimore, Md; modern materials handling equipment; use of straddle carriers, conveyors and automatic machinery to eliminate many traditional hand operations, and cut labour requirements drastically; raw material is 99.4% fire-refined blister copper.

LANGE, A., (1960), Suspension Smelting and Other Rapid Processes, Erzmetall 13, No. 4, 151-7, April.
ibid No. 5, 216-23, May.

LOCKYER, P.C., and NELLER, R.R., (1960), Electrolytic Refining of Copper, Metal Industry 96, (5), 88-92, Jan; (6) 110-14, Feb. 5; (7) 129-30, Feb. 12.
Plant and processes at Mufulira, Northern Rhodesia.

LYDICK, H.W., and RISLER, C.B., (1960), Magamp Improves Anode Casting at Inspiration, Eng. and Min. J. 161, (5), 102-4, May.
New and entirely static regulating system insures smooth surfaced anodes in casting wheel.

MECHENOV, P., and SOTIROV, B., (1960), The complex Extraction of Copper, Zinc and Lead from a Mixed Copper-Zinc-Lead Concentrate, Minno Delo i Met. (Sofia) 15, No. 1, 30-5.

MIYOSHI, S., IKEDA, S., TSURU, S., and FURUTA, E., (1960), The "L-P-F" Process of the Tsuchihata Deposit, Nippon Kogyo Kaishi 75, 455-62.
Until now the sludge flotation of raw ores was disturbed by the presence of sol. salts. The new leach-pptn-flotation (L-P-F) process yielded almost equally satisfactory results.

MOSHER, M.A., (1960), Some Practical Aspects of Copper Refining in a Multiple System Tank House, Electrochem Soc. J. 107, sup 7C-12C Jan.

NUSSBAUM, A.I., (1960), New Continuous Casting Process for Copper Wire Bars, Wire and Wire Prod. 35, 472, April.

PEARCE, R.F., et al, (1960), New Method of Matte Refining by Pressure Leaching and Hydrogen Reduction, J. Metals 12, 28-32, Jan.

PENZIMONZH, I.I., (1960), Smelting of Concentrates in a Fluidised Layer, Trudy Inst. Met i Obogashcheniya, Akad. Nauk Kazakh SSR 3, 105-23. (CA 55 15259 d)

POLLANDT, F., and PEASE, M.E., (1960), Extraction of Copper and Silver by Segregation Process in Peru, Inst. Min. and Met. Trans. 70 (3), 143-56, Dec.
Discussion of paper 69, (12), 1959-60 issue.

POBEDONOSTEV, Yu. K., POLYAKOV, A.N., REVAZASHVILI, M.G., YAKUSHEV, E.A., KLUSHIN, D.N., TARKHOV, N.G., KOCHNEV, M.I., (1960), Industrial Scale Experiments on Using Oxygen-enriched Air in Copper Matte Bessemerizing at Balkhashsk Combine, Tsvetnye Metally 33, (2), 29-36, Feb.
Design of control apparatus; tabular data on converter behaviour and output.

ROBERTSON, D. J., (1960), Filtration of Copper Smelter Gases at Hudson Bay Mining and Smelting Co., Can. Min. and Met. Bul. 53, 326-35, May.

SHALYGIN, L.M., and MEIEROVICH, V. B., (1960), Means for Intensification of Converter Efficiency in Nonferrous Metallurgy, Tsvetnye Metally 33 (7), 16-19, July.
Critical examination of article by I. M. Iolko; rising pressure in air blowing gives positive results for tuyere behaviour, slag properties and matte reaction; oxygen-enriched blowing can be rendered efficient only by top blowing as previously proposed by author et al.

SPROULE, K. et al (1960), Treatment of Nickel-Copper Matte, J. Metals 12, 214-19, March.
Discussion 12, 442, June.

TONKONOGII, A. V., ONAEV, I. A., BASINA, I. P., VDOVENKO, M. I., POBEDONOSTSEV, Yu. K., POLYAKOV. A. N., MEEROVICH, V. B., REVAZASHVILI, M. G., DURNOVO, I. G., and TSYGANOV, V. V., (1960), Cyclone smelting of Copper Sulphide Concentrate, Tsvetnye Metally 33 (3), 20-8, March.
Design of furnace operating at Balkhashsk metallurgical combine; charge is same as in reverberatory furnace and copper matte contains satisfactory quantity of Cu; advantages of cyclone smelting; furnace behaviour and new improvements.

TSUROMOTO, T., (1960), Smelting of Copper Ores and Concentrates by Converter Using Oxygen-Enriched Air at Hitachi Mine, Min. and Met. Inst. Japan - j 76 (861), 179-95.
Construction program at mine; wet blending is employed as preliminary treatment for converter feed ore; air separation unit produces oxygen; converter slag is treated by ordinary ore dressing process; expansion of two sulphuric acid plants.

VERMENICHEV, S. A., DEEV, V. I., and KOCHNEV, M. I. (1960), Roasting copper-zinc concentrates in a current of oxygen, Zhur. Priklad.. Khim. 33, 1036-42 (C A 54 17178 c)

VERNEY, L. R. (1960), Factors affecting Copper Reverberatory Furnace Performance and Their Influence on Choice of Various Smelting Methods, Instn. Min. and Met. Trans 69 (5), 211-36, February.
Discussion (8), 467-83.
At smelter of Mufulira Copper Mines Ltd improvements in smelting result from lowering both moisture and gangue contents of concentrate; there is tendancy for less efficient smelting to be associated with treatment of chalcopyritic concentrates as compared with bornitic and chalcocitic concentrates, although to some extent this effect can be overcome by adjusting composition of nonsulphide portion of furnace charge.

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ANON (1961)(1), Copper Refining, Metal Industry, February 12.

ANON, (1961)(2), Gas Reduction - New Route to Copper Refining, Phelps Dodge Corp. Chem. Eng. 68, 100+, June 12.

ANON, (1961)(3), New Copper Route Pioneered in Canada - Chemical Route to Copper, Can. Chem. Process 45, 72, March.

ANON, (1961)(4), Operations at Weed Heights, Min. Cong. J. 47, 60-1, May.

ANON, (1961)(5), Philippines to get Unique Copper Plant, Eng. and Min. J. 162, 91 April.

ANON, (1961)(6), Philippines gets New Copper Plant, Chem. and Eng. N. 39, 46-47 February 13.

ANON, (1961)(7), Unique Integrated Copper Operation, Can. Min. J. 82, No. 7 46-51, July.

ANON, (1961)(8), Yugoslavs Prove Out Copper Process, Steel 148, 132 March 13.

ANDERSON, J.N. (1961), Reverberatory Furnace and Converter Practice at the Noranda and Gaspe Smelters, "The Extractive Metallurgy of Copper, Nickel and Cobalt", (P. Queneau ed), Interscience.

ARGALL, G. O., (1961), Copper Concentrates to Metal Shapes in Marinduque's New Philippine Plant, World Mining 14 (3), March.

ARTAMONOV, K. I., LEBEDEV, N. I., ERGALIEV, E. E., LESECHKO, A. K., YAKUSHIN, M. V., KAZAKOV, V. N., BRYUKHANOV, N. G., NIKITINA, L. I., KHVESYUK, F. I. (1961), Smelting of Copper Concentrate in Blast Furnace with Oxygen-enriched air, Tsvetnye Metally 34 (3), 32-9 March.
Practice at Irtysh copper smelter; comparative data on smelting with air blast and oxygen enriched blast; optimum results obtained with addition of 27.3% of oxygen.

ARTAMONOV, K. I. (1961), Use of Oxygen in the Shaft Smelting of Copper Charges, Met. i Khim. Prom. Kazakhstana, Nauchn.-Tekhn. Sb. (2), 34-7.
To determine the effectiveness of an O-enriched blast, smelting was conducted with an air and with an O-enriched blast. Compared with an air blast the sp. smelting rate of the charge increased. The amount of air supplied to the furnace was reduced. The av. temp. of the exhaust gases decreased. The amount of incrustations in the blast zone and the hearth of the furnace decreased, the layer of slag hardened on the walls was thinner and the yield of cyclone dust decreased.

AVSARAGOV, B. G., et al (1961), Means for Improving Complex Treatment of Ural Copper and Copper-Zinc-Pyrites. Tsvetnye Metally 34, (4), 1-3.

BARTH, O., (1961), Electric Smelting of Sulphide Ores, "The Extractive Metallurgy of Copper, Nickel and Cobalt", (P. Queneau ed), Interscience.

BEAN, J. J., (1961), The Leach, Precipitation, Flotation Concentration Method at Miami Copper, Quart. Colo. School of Mines 56 (3), 263-81.

CHAPMAN, F. H., and PAGE, E. W., (1961), Leaching and Electrowinning of Copper from Nchanga Oxide Concentrate, "The Extractive Metallurgy of Copper, Nickel and Cobalt", (P. Queneau ed), Interscience.

EVANS, D. J. I., ROMANCHUK, S., and MACKIW, V. N., (1961), Production of Copper Powder by Hydrogen Reduction Techniques, Can. Min. and Met. Bull. 54, (591), 530-9, July.

CHIH TS'AI HSIAO and SMIRNOV, V. I., (1961), Fluid-bed Sulphatising Roasting of Cobalt-bearing Mattes of Nickel and Copper Smelters, Tsvetnye Metally 34 No. 1 35-9, (CA 55 18485f).

FORBES, S. S., (1961), Handling of Anode Slime at Canadian Copper Refiners Ltd. Can. Min. and Met. Bull. 54 (589), 381-2, May. Anode Slime is treated for economic recovery of copper, silver, gold, selenium and tellerium after electrolytic refining of copper.

FOWLER, M. G., (1961), Smelting Practices of Phelpds Dodge in Arizona, "The Extractive Metallurgy of Copper, Nickel and Cobalt," (P. Queneau ed), Interscience.

FREEMAN, G. A., RAMPACEK, C., and EVANS, L. G., (1961), Copper Segregation Process Shows Promise at Lake Shore Mine, Min. Eng. 13, 1152-5, October.

GAZARYAN, L. M., (1961), Suspension State Smelting of Copper Concentrates on Industrial Units, Tsvetnye Metally, English Translation 2 No. 3, 40-6 April.

HENRICH, G., HOFFMANN, W., and JUNGHANSS, H. J., (1961), Recovery of Copper as Cu_2O, Erzbergbau Metallhuettenwesen 14 (5), 223-8, May. New method for recovery of copper at Duisburg copper smelting plant; Cu_2O is produced by continuous reaction of CuCl suspension in hot water with lime milk slurry, is collected in thickener, and dewatered in revolving filter.

HUTTL, J. G., (1961), Cement Copper Begins at Bagdad, Eng. and Min. J. 162 (6), 86-8, June. Arizona project recovers copper by normal leach-precipitation from low grade oxide stockpiles and from oxide ore overburden; plant includes auxiliary sulphuric acid contact unit; capacity is raised by 20 tpd of copper; plant features deep cells to provide optimum contact between pregnant solution flowing through cells and detinned can load; semi-automatic handling of cans from stockpile to cells.

HUTTL, J. B., (1961), (2), "Poling" becomes Ancient History at Phelps Dodge Smelters, Eng. and Min. J. 162 (7), 82-5, July.
Use of reformed natural gas in place of green poles during reduction step; natural gas is reformed to give carbon monoxide and hydrogen which are active reducing agents.

JANICIJEVIC, D., (1961), Modern Plant Produces Oxygen-free Copper. Metal Prog. 79, 112-14+, February.
Report from Yogoslavia.

JENKINS, J. C., and SAINT-SMITH, J. C., (1961), Townsville Copper Refinery, Australasian Inst. Min. and Met. - Proc. (197), 239-60, March.
Anode, tank house and wire bar plant and operations with particular reference to those features of design and methods which are characteristic of refinery at its inception; organization of construction and origin of equipment.

KERSHANSKII, I. I., VORONIN, I. S., SAVRAEVA, K. E., GNATYSHENKO, G. I., SHUROVSKII, V. G., and SHOKOVAEV, Sh. D., (1961), Pilot Plant Results of Direct Electric Smelting of High Silica Copper Concentrates Reported, Tsvetnye Metally 34 (9), 24-34, September.
Experiments of smelting in 3-phase electric furnace of non-roasted concentrates with addition of limestone or slag; matte with high content of non-ferrous metals (gold, silver, etc.) obtained degree of desulphurization is somewhat less than in reverberatory furnace.

KLEIN, L., (1961), Gaseous Reduction of Oxygen-containing Copper, J. of Metals 13 (8), 545-7, August.
Description of process developed at Douglas (Arizona) plant of Phelps Dodge Corp. Process uses reformed natural gas.

LILLEY, J. N., CHAMBERS, R. W., and YOUNG, C. E., (1961), Handling of Acid Slurries at the Flash Smelting Plant of the International Nickel Co. of Canada Ltd. Can. Min. and Met. Bull 54 (589), 389-91, May.

MCARTHUR, J. A., and LEAPHART, C., (1961), Leaching of Chuquicamata Oxide Copper Ores, "The Extractive Metallurgy of Copper, Nickel and Cobalt, (P. Queneau ed), Interscience.

MCARTHUR, J. A., and LEDEBOER, B. J., (1961), The Electrolytic Tank House Operation and Cell Room Practice at Chuquicamata, Chile.

MAKLOVSKY, J., SCHMIEDL, J., HOLECZY, J. and SEHNALEK, F., (1961), Use of Oxygen in Some Metallurgical Processes), (Hutnicka Fak. Vst, Kosice, Czech.), Hutnicke Listy 16, 573-81.
Studies were made on roasting of sulphide ores and concentrates, roasting and melting of sulphide concentrate, metallurgical redn. treatment, converter treatment, design of exptl. equipment, calcn. of blowing time and effect of O on blowing time.

NELMES, W. S., et al (1961), Oxygen Jetting Secondary Copper, J. Metals 13, 216-20, March.

NICHOLSON, M. K., LOCKRIDGE, P. L., and BEALS, G. C., (1961), Copper Converting Practice at Chino Mines Division of Kennecott Copper, "The Extractive Metallurgy of Copper, Nickel and Cobalt," (P. Queneau ed), Interscience.

PAGE, E. W., (1961), Leaching of Oxide Concentrates at Nchanga, Seventh Commonwealth Mining and Metallurgical Congress, May.

PENZIMONZH, I. I., SCHUROVSKII, V. G., and KOZHAKHMETOV, S., (1961), Test of a New Variation of Flash Smelting of Copper Concentrates, Tsvetnye Metally 34 (6), 39-44.
A furnace for flash smelting of Cu concentrates is described. The concentrate, mixed with powd. coal, is blown with air or O through burners arranged in opposite pairs at 45-50^0 to the horizontal. The particles burn intensely, melt rapidly, coalesce, and fall into the bath, where they separate into matte and slag.

PRAX, Y., (1961), Olen Copper, Cobalt and Germanium Refineries, Ind. Chem. 37, 250-1, May.

QUENEAU, P., (ed)(1961), Extractive Metallurgy of Copper, Nickel and Cobalt, Interscience Publishers for AIME Metallurgical Society, 647 pages.

ROBB, K. G., (1961), Bibliography on the Extractive Metallurgy of Copper, Nickel and Cobalt, "The Extractive Metallurgy of Copper, Nickel and Cobalt", (P. Queneau ed), Interscience.

SADDINGTON, R. R., CURLOOK, W. and ROORDA, H. J., (1961), Roasting Practices at International Nickel, Trans. Can. Inst. Min. Met. 64, 359-66.

SCHLOEN, J. H., and FORBES, S. S., (1961), Industry Report on Modern Copper Tank House Practice, "The Extractive Metallurgy of Copper, Nickel and Cobalt", (P. Queneau ed), Interscience.

SMIRNOV, V. I., LEBED, B. V., TIKHONOV, A. I., and YABLONSKII, Yu. A. (1961), Complex Treatment of Slags from Copper Smelting. Tsvetnye Metally 34 (10), 46-50.
Liquid slag from Cu smelting is dezinced by fuming and decoppered with limestone, pyrite and C in an elec. or reverberatory furnace. The treated slag is smelted in an elec. furnace to produce Cu-free iron.

SPROULE, K., HARCOURT, G. A., and RENZONI, L. S., (1961), Treatment of Nickel-Copper Matte, "Extractive Metallurgy of Copper, Nickel and Cobalt". (P. Queneau ed), Interscience Publishers for AIME Metallurgical Society, N. Y.

STARRATT, F.W., (1961), Philippines' First Copper, J. Metals 13, 221-2, March.

THOMPSON, R.B., and ROESNER, G., (1961), Fluid Bed Roasting -- Principles and Practice, "Extractive Metallurgy of Copper, Nickel and Cobalt" (P. Queneau ed), Interscience Publishers for AIME Metallurgical Society, N.Y.

TSURUMOTO, T., (1961), Copper Smelting in Converter, J. of Metals 13, (11), 820-4, November.
Description of pilot plant and commercial scale tests of converter smelting of copper concentrates with oxygen-enriched air at Hitachi Smelter of Nippon Mining Co; advantages over blast furnace smelting are said to be; simple, flow sheet, fuel and flux economy, production of blister copper in single step, and economical recovery of H_2SO_4 from high grade SO_2 gas.

WARD, R.G., and HOAR, T.P., (McMaster University, Hamilton, Can.), (1961), The Electrolytic Removal of Oxygen, Sulphur, Selenium and Tellurium from Molten Copper, J. Inst. Metals 90, 6-12, (Paper No. 2085).
Cathodic treatment of molten Cu under Molten $BaCl_2$ reduced O, S, Se, and Te contents to less than 0.001%. Current efficiency with lab. equipment was 60-80%. Application of the process for blister Cu direct from the converter is discussed.

WHITEHEAD, A.B., and URIE, R.W., (1961), Fluidized-bed Roasting of Copper Concentrates, Australasian Inst. Mining and Met. Proc. (199), 51-85.
High-grade Cu Concentrates from the major Australian mines were roasted in a fluidized bed reactor to produce calcines with desirable leaching properties.

YAKUSHIN, M.V., et al, (1961), Converting Multimetallic mattes with an Oxygen-enriched Blast, Tsvetnye Metally 34 (10), 34-9.
The Cu-converting practice using O-enriched air at the Irtysh Smelter is described.

1962

ANON, (1962)(1), Every Economy Counts in Electrolytic Refining at Anaconda's Raritan Copper Works, Power 106, 69-71, May.

ANON, (1962)(2), Fluosolids System Reduces Costs, Increases Smelter Capacity at Tennessee Copper, Can. Min. Met. Bull., 4, 5 (Dec).

ANON, (1962)(3), Nippon Proves Oxygen Converters Cut Copper Smelting Costs, Eng. and Min. J. 163, 192-3, June.

ARENTZEN, C., (1962), Oxygen-enriched Air for Converting Copper Matte, J. of Metals, 14 (9), 641-3, September.

Air enriched with 25.5-44.2% O was used in tests; no final answer is given; secondary effects of using enriched air on operation of converter plant are discussed.

ASANO, N., and NOJIMA, S., (1962), Roasting of Copper Concentrates in the Herreshoff Roaster, Nippon Kogyo Kaishi 78, 397-402.

BARTH, O., (1962), The Smelting of Copper and Nickel Sulphide Ores and Concentrates in the Electric Furnace, Freiberger Forschungsh, B67, 35-55.

Heat balances and furnace efficiencies are tabulated. The metallurgical principles are the same for furnaces with gas or electric heating.

BOCHKAREV, L. M., BYKHOVSKII, Yu. A., and SHUMILOVA, O. P., (1962), Smelting of Copper and Copper-zinc Sulphide Concentrates in the Suspended State in an Oxygen Blast, Sb. Tr., Gos. Nauchn-Issled. Inst. Tsvetn. Metal (19), 397-410.

Large-scale lab. app. with a production of 2.5 tons/day was used for planning a pilot-plant furnace for the production of 75 tons/day. The experiments confirmed the basic advantages of the method.

BOCHKAREV, L. M., and BYKHOVSKII, Yu. A., (1962), The Status and Perspectives of the Use of Oxygen in Certain Nonferrous Metallurgical Plants, Tsvetnye Metally 35 (11), 38-43.

The use of O-enriched air in the shaft-furnace smelting of Pb, Cu and Ni suspending smelting of Cu, converting of Cu and Cu-Pb mattes, slag fuming.

BODI, D., (1962), Processing of Lead-Copper Matte, Kohasz. Lapok 95, 264-8.

BOGART, R. C., (1962), Profits from Waste Pit, Instrumentation 15 (1) 4-7.

Use of Honeywell control system in sulphuric acid plant at Bagdad Copper Mine in Arizona.

BRANTLEY, F. E., and SCHACK, C. H., (1962), Deoxidation of Blister Copper by Gaseous Reduction, U.S. Bur. of Mines, Rept. Invest. 6113, 12 pp.

Lab-scale tests suggest that the gas-redn. method has important tech. and economic advantages over wooden poling. Almost continuous deoxidn. tests indicate the possible use of a packed-column, vertical-tube furnace as a fast, efficient, continuous means of deoxidising molten Cu.

CHALOV, V. I., and LIPIN, B. V., (1962), Flash Smelting of Copper-zinc Concentrates, Tsvetnye Metally 35 (7), 29-31.

The flash smelting of Cu-Zn concentrate with the addition of 10-15% sand by wt. is proposed. The products are Cu-rich matte, slag, and fume contg. $\sim$ 85% of the Zn.

DAVIDSON, A.M., (1962), Furnace Lining Replacement in Oxygen-enriched Blast, Izvestiya Vysshikh Uchevnykh Zavedenii, Tsvetnaya Metallurgiya (6), 122-5.
Reverberatory copper-smelting furnace using oxygen-enriched air blast given as example; formulas are derived for gas stream temperature gradient along combustion zone for determining required frequency of lining replacement in different spots.

DAY, F.H., (1962), Anaconda Reduction Works in Retrospect, J. Metals 14, 824-7, November.

FURSMAN, O.C, (1962), Recovery of Mineral Values in Cupriferous and Nickeliferous Pyrrhotite, U.S. Bur. Mines, Rept. Invest. No. 6043, 24 pp.
Samples of Funter Bay cupriferous and nickeliferous pyrrhotite ore were treated by beneficiation, roasting, elec. smelting of roasted ore for matte production, chlorination, leaching and sulfatizing roasting with subsequent leaching. Sulphatizing roasting followed by H_2O leaching was the only treatment that gave recoveries of Cu and Ni which may have application.

GIBSON, N., (1962), Smelting Developments at Rhokana Corporation Ltd, Commonwealth Mining and Met. Congress, 7th - Tech. Proc. Northern Rhodesia, 301-21.
Salient features in intensive program of investigations and development which has been carried out in smelter reverberatory furnace section.

HOLECZY, J., SCHMIEDL, J., and SEHNALEK, F., (1962), Continuous Production of Converter Copper from Copper Sulphide Concentrate in a Smelting furnace, Czech, 104,449, July 15 (appl. January 25, 1961), 3 pp.
The conventional method of convertor Cu production requires three independent operations and is not continuous. This method of continuous production has various advantages.

HOLECZY, J., SCHMIEDL, J., SEHNALEK, F., KUNHALMI, G., and LUX, J., (1962), Continuous Conversion of Copper Mattes, Sb. Ved. Prac Vysokej Skoly Tech. Kosiciach (1), 161-73.
A new pilot plant installation was devised, and the technology of continuous conversion of Cu mattes was studied. The Cu produced in this installation was of good quality and the slag contained little Cu.

HUTTLE, J.B., (1962), Kennecott Adds Unique Lime Plant to Hayden Reduction Works, Eng. and Min. J. 163, 94-6+, November.

JENKINS, J.C., (1962), Plant Design for Electrolytic Copper Refining, Australasian Engr. 54, 42-6, April.
Examination of influence of original design on major cost factors such as operating labor and supervision, electrolyzing power, supplies, maintenance, and annual charges resulting from capital invested and value of copper in process; characteristics of tank houses; calculation of production per unit area of tank.

KLEIN, L., (1962), Gaseous Reduction of Oxygen-Containing Copper, Met. Soc. of AIME - Trans. 224 (1), 121-9, February.
Use of reformed natural gas by Phelps Dodge to replace poling.

KNOBLER, R.R., and JOSEPH, W., (1962), Copper Extraction and Refining at Mantos Blancos, J. of Metals 14 (1), 51-6, January.
Description of process specially developed to extract copper from copper chloride ores found at Mantos Blancos, Chile.

KNOBLER, R.R., and JOSEPH, W., (1962), Mantos Blancos Operation - Chile's New Integrated Copper Producer, Min. Eng. 14 (1), 40-5, January.
Presently, only largest of five known orebodies is being mined; reserves are 8 million tons of 1.9% Cu, about two-thirds of which occurs in atacamite and remainder in chrysocolla.

LAPEE, R.J., (1962), Copper Refining at Great Falls Reduction Department of Anaconda Co. Met. Soc. of AIME - Trans. 224 (2), 228-35, April.
History of progress of refining in Montana; casting furnaces and newly rebuilt electrolytic refinery and their operation described.

LOCKYER, P.C., and NELLER, R.R., (1962), Refined Copper Casting at Mufulira, Commonwealth Min. and Met. Congress, 7th - Tech. Proc. Northern Rhodesia.

MALOUF, E.E., and PRATER, J.D., (1962), New Technology of Leaching Waste Dumps, Min. Cong. J. 48, 82-5, November.

PERLOV, P.M., ESKIN, A.I., and MYAGKOVA, T.M., (1962), Extraction of copper from difficult ores by a combination of methods, Tr. Vses. Nauchn. -Issled. i. Proektn. Inst. Mekhan. Obrabotki Polezn. Iskop (131), 162-76.
A high degree of extn. of Cu from oxidised Cu ores difficult to conc. can be achieved by combined methods, including preroasting of the ore with small amts of NaCl and coke followed by flotation or leaching (ammonia solns.) of the cinders.

SUTULOV, A., (1962), Beneficiation of Chilean Oxidised Copper Ores by the Segregation Process, World Mining, 15, 28-9, August.

TRIANDAF, A., MAVROMATI, V., and OLARU, Fl., (1962), Modern Methods of Roasting Copper Concentrates, II, Rev. Chim. (Bucharest) 13, 389-95.
Fluidized bed roasting of cupriferous concentrates in mixture with auriferous pyrite and fluxes.

TSYGODA, I.M., KAZAKOV, V.N., SEREGIN, Yu.I., and KORNEEV, V.F., (1962), Pilot Tests of Sinter Roasting of Copper Concentrate Charge with Bottom Blast, Tsvetnye Metally (3), 23-30, March.
Operation of linear sintering furnace equipped with air blowing from bottom; design of furnace; result obtained at Irtyshsk Copper Smelting Plant with copper-lead-zinc concentrates.

TURNBULL, D.C., (1962), Converter Practice at Mufulira, C/wealth. Min. and Met. Congress, 7th - Tech. Proc. Northern Rhodesia Sec., 409-30. When expansion project is completed smelter production will be 195,000 short tons per year using 2 reverberatory furnaces, 3 converters and 3 anode furnaces.

VASIL'EV, V.V., KAN, T.D., and MURATOVA, N.E., (1962), Acceleration of the Selective Extraction of Copper Minerals by Ultrasonic Vibrations, Fazovyi Khim. Analiz Rud i Mineralov, Leningr. Gos. Univ. 50-5.
Application of ultrasonic vibrations, 540 kc/sec. at 2 kW for 1 hr, has greatly accelerated extn. of Cu from ores. In a mixture of 2 solns., 2% $Fe_2(SO_4)_3$ and 5% H_2SO_4, containing chalcocite powd. to 200 mesh, with the ratio soln.: ore 140, Cu extn. under ultrasonic vibrations was 10 times as fast as that with mech. stirring only. The rate of Cu extn. from the oxide ore containing Cu_2S impurities was increased in the ratio 20:1, in the same solution.

1963

ANON, (1963)(1), Concentration Plant in Northern Rhodesia, Min. J. 261, (6685), 315, Oct. 4.
Bancroft Mine in Northern Rhodesia treats 150,000 tons/mo; considerable degree of automatic control is used in plant; copper content is estimated by x-ray spectrograph; flotation circuit is being altered to permit production of separate low-grade oxide concentrate for leaching and electrowinning, which permits treatment of material carrying as little as 3% Cu.

E

ANON, (1963)(2), Continuous Casting of Copper Rod Announced, Iron and Steel Eng. 40, 171+, Nov.

ANON, (1963)(3), Greater Purity for Oxygen-free Copper, Metal Prog. 84, 124, Dec.

ANON, (1963)(4), Leach Plant for Low-grade Ore at Nchanga, Min. Mag. 109, (4), 201-4, Oct.
Since completion of low-grade oxide section of leach plant at Southern Rhodesian mine, oxide copper recovery has increased from 83.9 to over 90%.

ANON, (1963)(5), Miami Copper Mechanizes Can Handling, Eng. and Min. J. 164, (11), 97.
Handling of shredded cans has been mechanized from stockpile to point of distribution and loading at precipitation cells at Miami, Ariz. Copper mine.

ANON, (1963)(6), Nchanga's New Low-Grade Copper Leaching Plant, South African Min. and Eng. J. 74, (3684), 801-2, 804, 814, Sept. 13, see also Metal Min. and Processing 1, (11), Nov., 1964, p 22-5.
Two leach plants at Nchanga - high-grade and low-grade - have nominal production capacity of 7600 tons/mo; leach plant operates basically leaching-electro-winning process.

E

ANON, (1963)(7), New Process Relies on Unused Cu Quality; Arizona's Banner Mining Co., Eng. and Min. J. 164, 92, July.

ARGALL, G. O., (1963)(1), Leaching Dumps to Recover More Southwest Copper at Lower Cost, . Min. World 25 (11), 22-5, 27, Oct.
As a result of experimental work, it was found that leaching efficiency could be raised by dump leaching at 8 copper mines in western United States.

ARGALL, G. O., (1963)(2), How Leaching Recovers Copper from Waste and Leach Dumps in Southwest, . . . Min. World 25, (12), 20-4, Nov.
Practices employed at various southwestern mines for recovery of copper by leaching from copper oxide dumps.

BABENKO, A. R. et al, (1963), Roasting a Copper Batch in a Multiple-Hearth Furnace at the Krasnoural'sk Copper Smelting Works, Tr. Uralsk. Nauchn.-Issled. i Proektn. Inst. Medn. Prom. (7), 312-25.
An analysis was made of the furnace performance and processes were recommended by which the relative reaction rate of the oxidizing gas in relation to the particle is much greater than in a multiple-hearth furnace (the combined calcine in a fluidized condition; the roasting of the concentrate in a cyclone chamber).

BAUTISTA, E.N., PASILIAO, E.V., LOZANO, R.M., (1963), Leaching-Flotation of Balabac, Palawan Mixed Oxide-Sulphide Copper Ore, Philippine Islands. Bur. Mines - Report Investigation 51, 11 p. Dec.
Small scale laboratory leaching tests showed technical feasibility of producing cement copper from leach liquors, and copper sulphide concentrate from leached residue by flotation.

CARREKER, R. P., (1963), Dip-Forming — Continuous Casting Process, Jr. J. of Metals 15, (10), 774-80, Oct.
Laboratory development at General Electric Co. that has led to Dip Forming process of producing copper wire rod is described.

deCLEENE, P.B.,(1963), Physionomie Actuelle des Techniques d'Extraction et de Refinnage du Cuivre, Metallurgi 4, (1), 19-27.
Present Day Characteristic Features of Copper Extraction and Refining Techniques; review covers treatment by ignition, hydrometallurgy, electrolytic refining, dry charge, refractory materials for converters, and transformation, technology and specifications.

CRABTREE, E.H., (1963), Expert Compares Soviet Bloc and Western Copper-Molybdenum Recovery Practices, Eng. and Min. J. 164, 81-3, Dec.

HOLECZY, J., SCHMIEDL, J., SEHNALEK, F., (1963), Theory for Continuous Converter Process for Copper Matte, Hutnicke Listy 18, (2), 102-9, Feb.
Theoretical possibilities for obtaining blister copper from matte in one continuous automated operation are discussed on basis of $FeS-Cu_2S$ and $Cu-Cu_2S$ diagrams; necessary equipment was suggested, and feasbility of operation proved in model tests.

HOLECZY, J., SCHMIEDL, J., and SEHNALEK, F., (1963), Theory of the Triple-Layer Continuous Conversion of Copper Matte, Izv. Vysshikh. Uchebn. Zavedenii, Tsvetn. Met. 6, (4), 76-81.
Theoretical analysis of the pre-requisites for the adaptation of continuous conversion of Cu matte into converter Cu in one assembly and the technology of such a process are given.

HOLECZY, J., SCHMIEDL, J., and SEHNALEK, F., (1963), Theory and Pilot-plant Tests on Continuous Treatment of Copper Mattes in a Converter, Hutnicke Listy 18, (2), 102-9.
A brief thermodynamic study of converter treatment of Cu mattes is given. The disadvantages of conventional treatment, esp. unapplicability of automation, are stressed. On the basis of pilot-plant expts., a new technology is offered.

JACOBI, J.S., (1963), Recovery of Copper from Dilute Process Streams, Min. Eng. 15, 56-62, Sept.

KAMEDA, M., and YAZAWA, A., (1963), Refining of Crude Copper by Vacuum Melting, Tohoku Daigaku Senko Seiren Kenkyusho Iho, 19, (1), 57-68
Removal of impurities by vacuum melting was studied thermodynamically and some fusion expts. were carried out on a lab. scale. Bi and Pb were removed easily. Sb and As were reduced to $\sim$ 1/2 of their initial contents. Ni, Se and Ag remained in the melt. S and O were removed as SO_2 gas and the element with larger content remained in the melt. The d. of the Cu thus obtained was $\sim$ 8.8, while that of crude Cu was $\sim$ 7.

KERSHANSKII, I.I., (1963), Refining Dzhezkazgan Copper Concentrates by Electrosmelting, Bol'shoi Dzhezkazgan. Dobychai Pererabotka Rud (Alma-Ata: Akad. Nauk. Kaz SSR), Sb 399-417.
Results of pilot-plant expts. on refining Dzhezkazgan Cu concentrates by electrosmelting were cited.

KHLYNOV, V.V., ESIN, O.A., and NIKITIN, Yu. P., (1963), Separation of Metal Beads from Copper Smelting Slag by Means of Electric Current, Izv. Vysshikh Uchebn. Zavedenii, Tsvetn. Met. 6, (2), 58-62.
Addn. of 0.5-1.0% FeS to the Cu smelting slag, which is considerably smaller than usually practices at plant level (15% pyrite), produced a slag practically free of metal beads after treatment with a 6-amp current.

KOCHNEV, M.I., OKUNEV, A.I., MYASNIKOV, P.A., VERMENICHEV, S.A., SERGIN, B.I., and BAZHANOV, L.N., (1963), Smelting of Sulphide Materials in an Oxygen Flame without the use of Carbonaceous Fuels, Tr. Inst. Met., Akad. Nauk SSSR, Ural'sk. Filial (8), 33-42, Ural, Cu-Zn Concentrate, Noril'sk Cu Concentrate, Balkhash ore, Norilsk Cu-Ni Concentrate were studied.

KOCHNEV, M.I., OKUNEV, A.I., MYASNIKOV, P.A., VERMENICHEV, S.A, SERGIN, B.I., and STRIZHOV, G.F., (1963), Smelting of Ural Copper-Zinc Concentrates in a Fluidized Bed with an Oxygen Blast, Tr. Inst. Met. Akad. Nauk. SSSR, Ural'sk. Filial (8), 17-31.
The purpose of this work was to det. the feasibliity of smelting Cu-Zn concentrates (Cu 9.1, Zn 9.0, Fe 31.6, S 41.0, SiO_2 1.3 and the rest 8.0% in a vortex O flame with the production of matte contg. high Cu concn. and gases rich in SO_2. Another purpose was to improve the construction of the burner and det. the optimum conditions for the process. The app. for smelting was described by M., et al (ibid. 6-15).

KVYATKOVSKII, A.N., et al, (1963), Electrochemical Extraction of Copper from Fuming Process Slags, Tr. Altaisk. Gorno-Met. Nauchn.-Issled. Inst. Akad. Nauk Kaz SSR 14, 52-9.
E Previous studies of E. and Shurigin (CA 49, 2219 i; Khlynov and E., CA 52, 17011 i) have detd. the transfer in an elec. field of liquid droplets of Cu and matte using synthetic slags; the present work repeated the same expt. at 1200° with industrial slag contg. Cu 0.70, Pb 0.22, Zn 3.6, Fe (total) 33.4, S 1.58, SiO_2 25, CaO 9, and Fe^{III} 5.2%. The matte contained Cu 23.5, S 26.6, Pb 0.5, Fe (total) 43.4%. When liquid matte serves as an anode and a graphite cathode is introduced from the top into the slag, it is possible to ext. $\leq$80% Cu.

LASTRA, F., (1963), Copper Mining and its Future Development as Means to Increase National Wealth; Revista Chilena de Ingenieria y Anales del Institute de Ingenieros 76, (1), 7-10, Jan-Feb.
Statistics on copper production and data on investments; more revenue is expected when copper is refined in Chile; costs of copper smelting and refining.

McKINNEY, W.A., and EVANS, L.G., Segregation of Copper Ores by Direct-Firing Methods, (U.S. Bur. Mines, Tucson, Ariz.), U.S. Bur. Mines, Rept. Invest. No. 6215, 15 pp.
It is feasible to segregate oxidized Cu ores by use of a direct-fired refactory-lined rotary kiln. The preliminary direct-firing segregation studies were complemented by continuous runs on the quartzite ore, contg. 1.4% Cu in an integrated segregation-flotation pilot plant having a 60-lb/hr capacity. In a typical test, 84% of the Cu was recovered in a cleaner flotation product contg. 25% Cu.

MONNINGER, F.M., (1963), Precipitation of Copper on Iron, Min. Congr. J. 49, (10), 48-51, Oct.
Precipitation techniques as practiced at Weed Heights, Nev. operation, treating 11,800 tpd of 0.70% oxide.

PEASE, M.E., (1963), Copper-silver Smelter in Peru, Min. Mag. 109, (2), 73-9, Aug.
Smelter at Santa Lucia has output of 10 to 12 tpd of matte containing 43% copper and 120 oz per ton of silver.

OLEA, J., (1963), Thermal Refining of Copper, Tec. Met. (Barcelona) 19, (150), 205-8.
The effects of various impurities on Cu are briefly examined; Sb, As, S, Sn, Pb, Fe, Ni, Zn, O, and how their content is reduced. A new thermal refining process was developed, using strong oxidizing chemicals. Details of a 12-ton furnace and its operation are described. Some of the above elements are recovered in the fumes.

ORLOV, A.I., and KOPYLOV, G.A., (1963), Effect of Preliminary Sulphatizing Roasting on Copper Recovery from Oxidized and Mixed Copper Ores. Tr. Irkutskogo Politekhn. Inst. (18), 48-55.
Sulphatizing of oxidized and mixed Cu ores proceeds well in a rigid detd., temp. range of 480-500^0 for 1 hour with $\geq$6% S in the charge.

SHALYGIN, L.M., and DIOMIDOVSKII, D.A.(1963), Converting Nickel Matte with Top Blowing and Continuous Overflow of Slag, Tsvetnye Met. 36 (8), 20-30.

SHELUDYAKOV, L.M., TSOKALO, V.M., MEIEROVICH, V.B., and RANSKII, B.N., (1963), Extraction of Copper from Slags Produced in Copper Refining, Bolshoi Dzhezkazgan, Dobycha i Pererabotka Rud (Alma-Ata: Akad. Nauk Kaz. SSR), Sb, 431-433.
A new process of cementation of metal was developed for extg. nonferrous metals and Fe from waste slag. The process is based on the ionic theory of liquid slag wherein the high activity of C dissolved in liquid cast iron is utilized. Products of the cementation are pure Pb-Zn sublimates, cast iron, alloyed nonferrous metals, and depleted slags.

SUTÛLOV, A., (1963), Process of Leaching, Precipitation and Flotation, Concepcion, Chile, Universidad de Concepcion, Unstituto de Investigaciones - Tecnologicas 109 p.
Application of process in treatment of oxidized or partially oxidized copper ore.

ZIMMERLEY, S.R., (1963), Reverberatory Smelting of Copper Sulphide Ores with Oxygen, (to Kennecott Copper Corp.), U.S. 3,102,806 (Cl. 75-74) Sept. 3 (Appl. June 10, 1958).
After fusion of the charge is started along the side walls O is blown downward onto those parts of the melting charge either alone or mixed with air.

1964

ANON, (1964)(1), Blast-Furnace for Melting Copper Scrap, Metallurgia 70, 235, Nov.

ANON, (1964)(2), Chemical Process for Copper gets Tryout, Chem. and Eng. N. 42, 30, Dec. 28.

ANON, (1964)(3), Continuous Casting Installation for Copper Billets, Metallurgia, 70, 275, Dec.

ANON, (1964)(4), Continuous Casting Plant for Copper, Engineering 198, 625, Nov.

ANON, (1964)(5), Copper Quality and Output Rise in Continuous Casting Setup, Iron Age 194, 60-1, Oct. 15.

ANON, (1964)(6), How Nchanga's Improved Metallurgy Raises Copper Recovery, World Mining 17, (9), 28-31, Aug.

ANON, (1964)(7), Kennecott System Improves Copper Recovery, Chem. and Eng. N. 42, 46-7, Nov. 2.

ANON, (1964)(8), Mount Isa Pilots New Methods, Pushes Expansion, Min. Eng. 16, 82-9, Oct.

ANON, (1964)(9), Mount Lyell: Tasmania's Copper Producer, Min. Eng. 16, 90-1, Oct.

ANON, (1964)(10), Mount Morgan: Queensland Copper Producer, Min. Eng. 16, 92-3, Oct.

ANON, (1964)(11), New Concentrate Handling System Cuts Cost at Smelter, Met. Min and Processing 1, (2), 36-9, Feb.
Modernization of concentrate handling facilities at Inspiration, Ariz., smelter has reduced labour requirements by 70% over old system and provided more precise blending of smelter feed; new installation provides storage, bedding and blending for more than 12,000 tons of copper concentrates and precipitates that await reduction; smelter feed includes concentrate, cement copper and ore from various sources.

ANON, (1964)(12), Partial Roasting of Concentrate Boosts Smelter Output at Tennessee Copper, Metal Min. and Processing 1, (11), 32-4, Nov.
By partially roasting concentrate in fluidized bed roaster before charging it to reverberatory furnace, Tennessee smelter reclaims additional 17 tpd of sulphur; capacities of reverberatory furnace and converter are increased by one-third; roasting is autogenous; copper is increased in matte.

ANON, (1964)(13), Recovery of Copper from Refractory Ores, South African Min. and Eng. J. 75, (3746), 1427-8, Nov. 20.
Plant for treatment of refractory ores to be built at Kitwe, Northern Rhodesia, will have initial capacity of 500 tpd; main ores to be treated are copper silicate and cupriferous vermiculite.

ANON, (1964)(14), Small Plant Recovers Cement Copper, Metal Min. and Processing, 1, (2), 48-50, Feb.
Oxide copper ore at central Arizona deposit, heaped into 2 large dumps laid out on surface. Dilute sulphuric acid is pumped to ponds; pregnant solution is pumped to precipitation cells charges with shredded, de-tinned cans.

BACK, A.E., FISHER, K.E., and KOCHERHANS, J., (1964), Recovery of Copper from Dilute Acid Solutions (to Kennecott Copper Corp.) U.S. 3,154,411 (Cl. 75-109), Oct. 27, (Appl. Mar. 20, 1862) 7 pp. App. is described for contacting large vols. of dil. aq. solns., such as mine waters contg. 0.4-4 g Cu/l with metallic Fe preferably sponge Fe reduced from pyrite, and collecting the Cu particles pptd. from a continuous soln. flow in 2-3 hours, then discharging the collection to a filter, and repeating the cycle with a fresh feeding of Fe into the app.

BELL, J.A., (1964), Tiltable Rotary versus Reverberatory Furnaces for Refining Copper, Wire and Wire Prd. 39, 77-8+, Jan.

BOGERT, J.R., (1964)(1), Asarco's Modernization Program Brings New Life to 52-Year Old Smelter, Metal Min. and Processing 1, (8), 23-5, Aug.
As result of modernization program at Hayden, Ariz., smelter, concentrates are processed at rate of 24,000 tons/mo. Modernisation of concentrate-handling and reverberatory design.

BOGERT, J.R., (1964)(2), Kennecott Casts Bigger Blister Cakes, Metal Min. and Processing 1, (6), 46-7, June.
Smelter at McGill, Nevada, casts blister copper in 6000 lb cakes.

BOGERT, J.R., (1964)(3), More Muscle Developed for Matte Tapping, Metal Min. and Processing 1, (4), 36-8, Apr.
Combination rotary drill and pressurised mud gun mounted on electric-powered 4-wheeled chassis has been introduced at Hayden, Ariz., smelter in order to automate tapping and plugging of reverberatory furnace.

EK, C., and MASON, A., (1964), Some Modern Procedures in the Pyrometallurgical Extraction of Nonferrous Metals, Rev. Universelle Mines 20, (8), 223-32 (Fr).
Innovations are discussed in the pyrometallurgical recovery of Cu, Pb and Zn, such as (1) fluid-bed roasting; (2) direct charging of sulfide concentrates into a molten slag layer; (3) use of "reformed" gaseous fuel in deoxidn.; (4) autogenous smelting.

ENBAEV, I.A., and VORONINA, L.I., (1964), Cast Iron Shavings as a Precipitant for Cementation Copper, Dobycha i Obogashch. Rud. Tsvetn. Met., Nauchn.-Tekhn. Ref. Sb. (7), 42-3 (Russ).
The effect of the size of cast iron shavings on distribution of active Fe according to fraction and on Cu cementation for different Cu amts. was studied.

EVANS, S., ROMANCHUK, S., and MACKIW, V.N., (1964), Treatment of Copper-Zinc Concentrates by Pressure Hydrometallurgy (Sherritt Gordon Mines Ltd, Fort Saskatchewan, Alberta, Can.), Can. Mining Met. Bull 628, (8), 857-66.
Pressure-leaching processes and techniques were designed for treating a mixture of concentrates from the Bagacay and Sipalay Mines on the Philippine Islands of Samar and Negros resp. Following NH_3 leaching and soln. purification, Cu was recovered as powder with controlled characteristics by redn. of the soln. with H under pressure. Zn was recovered by electrolysis.

GROVES, R. D., (1964), Preparation of Copper Powder from Leach Solutions After Precipitation with Iron, U.S. Bur. Mines - Report Investigations, 6486, 23 p, see also Metal Industry 105, (9), 273-5, Aug.
High-grade copper powders can be produced under controlled conditions from cement copper products.

HENYCH, R., KADLEC, F., SEDLACEK, V., (1964), Refining of Copper by Gaseous Ammonia, Hutnicke Listy 19, (9), 645-50, Sept.
Use of gaseous ammonia for poling copper is said to have been found preferable to customary use of wood.

HOLY, V., (1964), Design of Industrial Cyclone Reactor by Using Data Obtained in Pilot Plant Installation, Hutnicke Listy 4, (19), 262-8.
Cyclone smelting of Cu, Cu-Sb, Cu-Pb and Zn concentrates, pyrite and other ores is a new pyrometallurgical method.

KARLYSHEV, B.N., (1964), Mineralogical Composition of Slags in Cyclone Smelting, Tr. Inst. Met. i Obogashch., Akad. Nauk Kaz. SSR 11, 185-91, (Russ.).
Smelting for the production of crude Cu and matte was done in an exptl. cyclone.

KOLAROV, M., KUNCHEV, N., DIMITROV, I., and LYUTIBRODSKI, V., (1964), Direct Smelting of Complex Concentrates in a Converter, Rudodobiv Met. (Sofia) 19, (10), 21-5, (Buld.).
The conventional horizontal converter of 4 tons capacity was used for direct smelting of a flotation concentrate containing Cu 17.90, Pb 10.82, Zn 2.55. The concentrate was compressed with 10% shaft furnace dust. Blister Cu was produced and Pb was extd. into the dust.

KOMLEV, G.A., LEVKOVSKII, O.V., and SHIROKOV, A.V., (1964), Reduction of Liquid Oxidized Copper with Natural Gas, Tsvetn. Metal. 37, (9), 13-14 (Russ.).
In lab. expts. converter Cu contg. Cu 98.6 and O 0.4% was deoxidised with reformed natural gas of varying compn. Deoxidn. with a rich gas to 0.15-0.20% O and then with a lean gas (H 1.5-2 and CO 1-1.2%) yielded dense Cu.

KRAUME, E., (Ed) (1964), Die Metallischen Rohstoffe, Band 4 - Kupfer, Ferdinand Enke Verlag, Stuttgart.

KUPERMAN, G.M., et al, (1964), Extraction of Copper from the Madneuli Sulphide Ore, Tr. Inst. Khim., Akad. Nauk Gruz. SSR 17, 13-37 (Russ.)
Lab. expts. were made on leaching of Cu sulphide concentrates (Cu 10.43-12.07). The optimum conditions of leaching in autoclaves were: temp. 160^0, partial pressure of O 10-15 atm. (total pressure 15-20 atm.), ratio of solid: liquid 1:4-1:5, particle size of ground ore 74 microns, and duration of leaching 8-12 hr. Under these conditions 97-9% Cu was extd. and 90% of the S was oxidized into sulphates.

KUPRYAKOV, Yu. P. and MILLER, O.G., (1964), The Use of Oxygen in Reverberatory Smelting of Copper Concentrates, Izv. Akad. Nauk Uz. SSR, Ser. Tekhn. Nauk 8, (5), 78-80 (Russ.).
Tests showed that enriching the blast with $\leq$25% O increased the smelting of the charge by 19.1%, decreased the fuel consumption by 15.3%, increased the thermal efficiency of the furnace by 37.8%, decreased the amt. of exhaust gases by 33%, decreased the exhaust dust by 45%, and increased the SO_2 content in the exhaust gases from 1.5-2.0%. These figures were for a charge contg. $\leq$20% pyrites.

LEVIN, A.I., and MUKHIN, V.A., (1964), Application of Channeled Electrolyzers for Electrolytic Refining of Copper, Tsvetnye Metally (6), 18-22, June.
E
This design makes it possible to change position of electrodes in relation to each other; system of circulation, providing movement of electrolyte parallel to electrodes, reduces concentration polarization and makes it possible to increase current density; schematic layout of installation is given; investigation shows that introduction of channeled baths will more than double productivity of electrolytic plant; cathode copper; with allowable content of silver and gold, can be obtained with current density of 500 amp/sq m.

LEVIN, A.I., and NOMBERG, M.I., (1964), Present State of Electrolytic Copper Refining and Economical Current Density, Tsvetnye Metally (Eng. Trans) 5, (2), 19-24, Feb.

OKABE, T., and ITO, H., (1964), New Hydrometallurgy of Copper, Tohoku Univ. - Faculty Eng. - Technology Reports 29, (1), 157-68.
Method uses ammonium sulphite as reducing agent of cupric sulphate; highly pure copper is produced without electrolytic refining.

ONAEV, I.A., et al, (1964), Complex Treatment of Balkhash Copper Charges by a Cyclone Method, Vestn. Akad. Nauk Kaz. SSR 20, (2), 42-9.
Results obtained on a pilot-plant cyclone installation were used to carry out balanced smelting of Balkhash Cu charges by the cyclone method. The extn. of Cu was $\geq$98%.

ONAEV, I.A., PANFILOV, P.F., and SHUMAKOV, V.V., (1964), Impoverishment of Converter Slags in Copper Smelting Plants, Tsvetn. Metal 37, (3), 82-5.
The most efficient way of reducing useful metal content in converter slags was processing the slag in the liquid state in an elec. furnace to which a mixt. of coke and CaO (or $CaCO_3$) was added.

OPIE, W.R., (1964), Factors Affecting Purity and Structure of Commercial Grades of Copper, Wire and Wire Products 39, (9), 1309-10, 1312-14, 1367, Sept.
Sources of copper (ore and scrap), as well as refining treatment, influence end product and properties.

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Pilot-plant smelting of ore contg. 21.71 Cu, 1.26 Pb, 0.61 Zn, 14.33 Fe, 19.7 S, with coal dust as fuel and secondary air preheated to 450-500^0 and carrying 30-3% O is closely examined and compared with the same practice but omitting O addn.

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Cyclone smelting allows the ore to be efficiently treated in a compact. app. with a max. diffusion and a max. heat exchange. The process is suitable for the treatment of Cu and complex concentrates because it allows the accompanying elements to be removed by sublimation. Cu may be freed completely from Pb and Cd and from most of its Zn, Se, Te and Rh by this method. Almost complete sublimation of Rh and Mo may be obtained with O-enriched air.

OXFORD, D. D., and WILLIAMS, P. H., (1965), Computerized System of Metallurgical Accounting and Control at Nchanga Consolidated Copper Mines Ltd, Min. Eng. 17, 55-9, Jan.

RAMACHANDRAN, T.R., KRISHNAN, K.N., and MALLIKARJUNAN, R., (Indian. Inst. Technol. Bombay) (1965), Oxygen Top Blowing of Copper Matte, Trans. Indian Inst. Metals 18, 56-8, March.
Top blowing of Cu mattes led to the production of Cu. The expected advantages of the process are listed.

APPENDIX C

DEFINITIONS AND DATA USED IN COST ESTIMATION

1. CAPITAL COST

1.1 Subdivision

The total capital investment required for a process consists of fixed capital (total cost of processing installations, buildings, auxiliary services and engineering involved in the creation of a new plant) and working capital (funds required for the normal conduct of business)[a][b]. The fixed capital is subdivided as follows:

		$
Purchased equipment		
Equipment installation		
Piping		
Instrumentation		
Insulation		
Electrical		
Buildings		
Land and Yard improvements		
Utilities		
	Physical Plant Cost	$
Engineering and construction		
	Direct Plant Cost	$
Contractor's fee		
Contingency		
	Fixed Capital	$

1.2 Estimation

In the absence of precise data one of the following three methods has been used to estimate the fixed capital cost for each of the processes discussed.

1.2.1 Percentage of Equipment Cost

Given the purchased cost of equipment, the following percentages can be allocated to the components of the fixed capital cost:[a]

(a) ARIES, R.S., and NEWTON, R.D., "Chemical Engineering Cost Estimation", McGraw Hill, 1955.

(b) BUCHANAN, R.H., and SINCLAIR, C.G., "Cost and Economics of the Australian Process Industries", West Publishing Corp., 1964.

	% of Equipment Cost
Purchased equipment	100
Equipment installation	25
Piping: solid process	14
solid-fluid process	36
Instrumentation:	
few or no controls	5
some specific controls	15
extensive controls	30
Insulation	8
Electrical substations, feeders and wiring	10-15
Buildings (outdoor construction)	30-40
Land and yard improvements	10-15
Utilities (complete new services)	75
	% of Physical Plant Cost
Engineering and construction	20-30
	% of Direct Plant Cost
Contractor's fee	4-10
Contingency	10-25

1.2.2 Lang's Method

This method also requires the purchased cost of equipment:

Fixed Capital Cost = Purchased Equipment Cost x Lang's Factor.
(Lang's Factor for a solid-fluid process is 3.63).

1.2.3 Seven-Tenths Method

This method requires the fixed capital cost of a plant for the same process, but of different capacity:

$$\frac{\text{Fixed Capital of Plant A}}{\text{Fixed Capital of Plant B}} = \left\{\frac{\text{Production Rate of Plant A}}{\text{Production Rate of Plant B}}\right\}^{0.7}$$

2. MANUFACTURING COST

2.1 Subdivision

Manufacturing cost is the sum of all the direct, indirect and fixed expenses incurred as a result of the actual manufacture of a product. Manufacturing cost is subdivided as follows: (Aries and Newton, loc cit)

Raw materials	$	
Labour		
Supervision		
Maintenance		
Plant supplies		
Royalties and Patents		
Utilities		
		Direct Manufacturing Cost $
Payroll overhead	$	
Laboratory		
Plant overhead		
Packaging		
Shipping		
		Indirect Manufacturing Cost $
Depreciation	$	
Property taxes		
Insurance		
		Fixed Manufacturing Cost $
		Manufacturing Cost $

2.2 Estimation

The following prices and proportions will be used for estimation purposes:

Direct Manufacturing Cost.

Materials (at smelter or refinery)

Raw Materials (cost not included in Treatment Cost)

		Cost per Ton $
Reagents:	Limesand	4
	Silica sand	2
	Ironstone	2
	Coke	22

Labour:	Assume unskilled labourers paid \$3000 pa, \$1.50 per man hr Assume skilled labourers paid \$4000 pa, \$2.00 per man hr (inclusive of overtime, shift allowance, bonuses, etc.)
Supervision:	10-20% of direct labour cost
Maintenance: (average wear)	6% annually of fixed capital investment.
Plant supplies:	15% of maintenance costs
Royalties and Patents:	1-5% of sales price of product

Utilities:

	Unit	Cost, \$
Fresh water	1000 gal	0.20
Sea water	1000 gal	0.04
Electric power	kWh	0.016
Fuel oil	ton	16.0
Fuel oil	gal	0.064
Compressed air (100 psi)	1000 cfm	0.04
Blower air	1000 cfm	0.01

Indirect Manufacturing Cost

Payroll overhead	15-20% of labour cost
Laboratory	10-20% of labour cost
Plant overhead	50-100% of productive labour cost
Packaging	-
Shipping	-

Fixed Manufacturing Cost

Depreciation	8-10%	annually of fixed capital investment
Property taxes	1-2%	annually of fixed capital investment
Insurance	1%	annually of fixed capital investment

FIGURES 1 TO 4

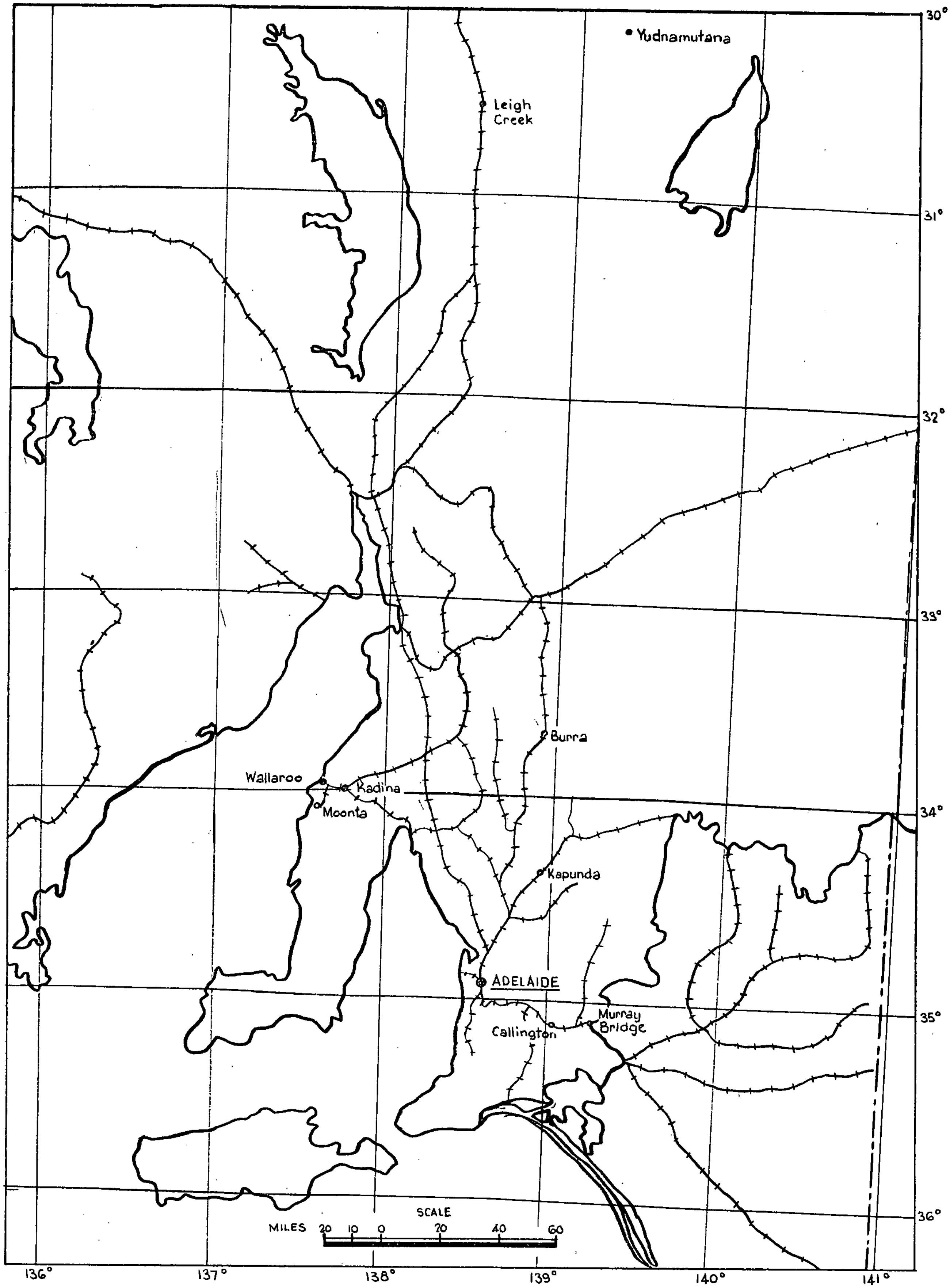

FIG. 1: SOUTH AUSTRALIA
Showing some copper occurrences

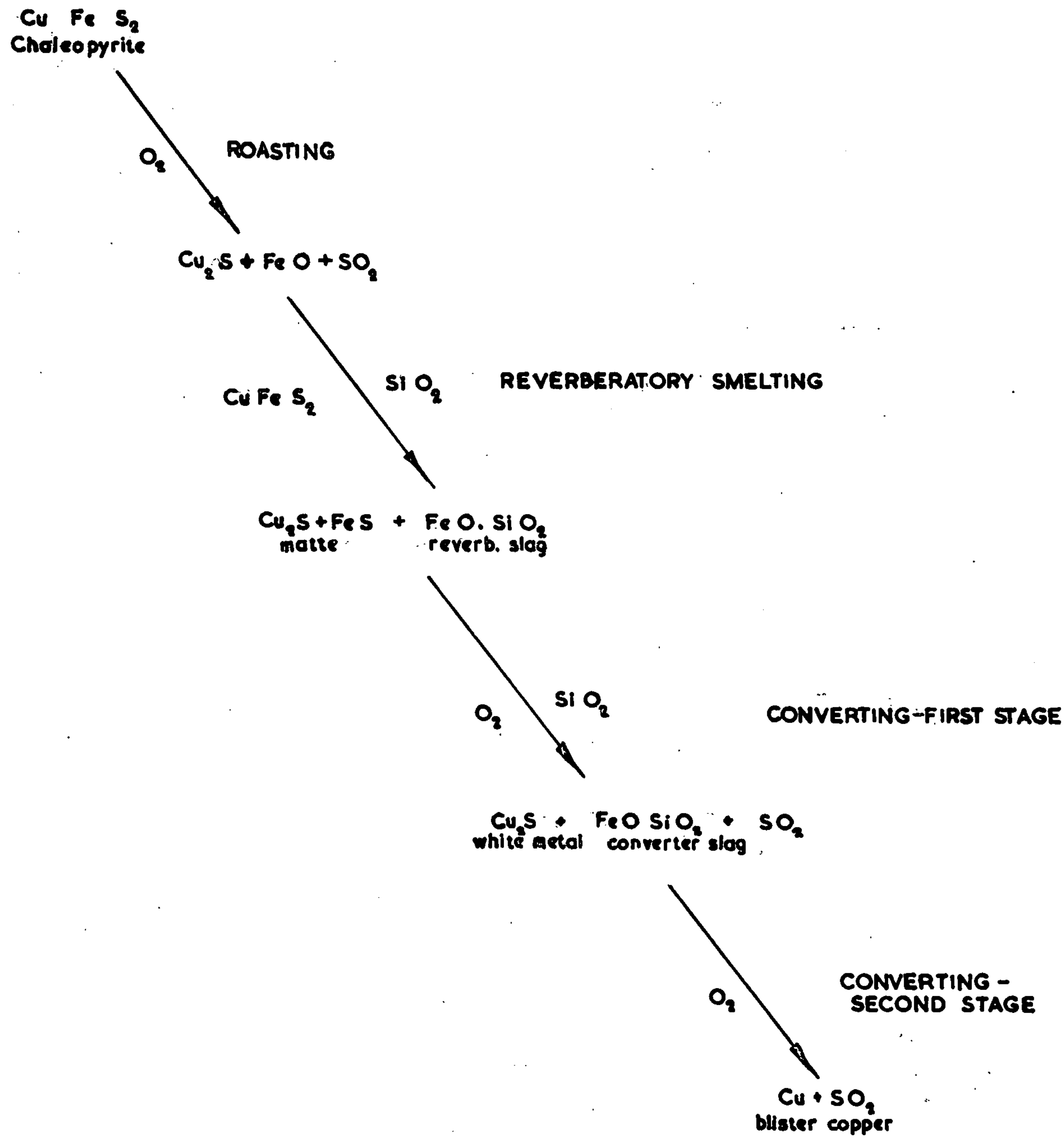

FIG. 2: REACTIONS OCCURRING IN COPPER SMELTING
Simplified, diagrammatic

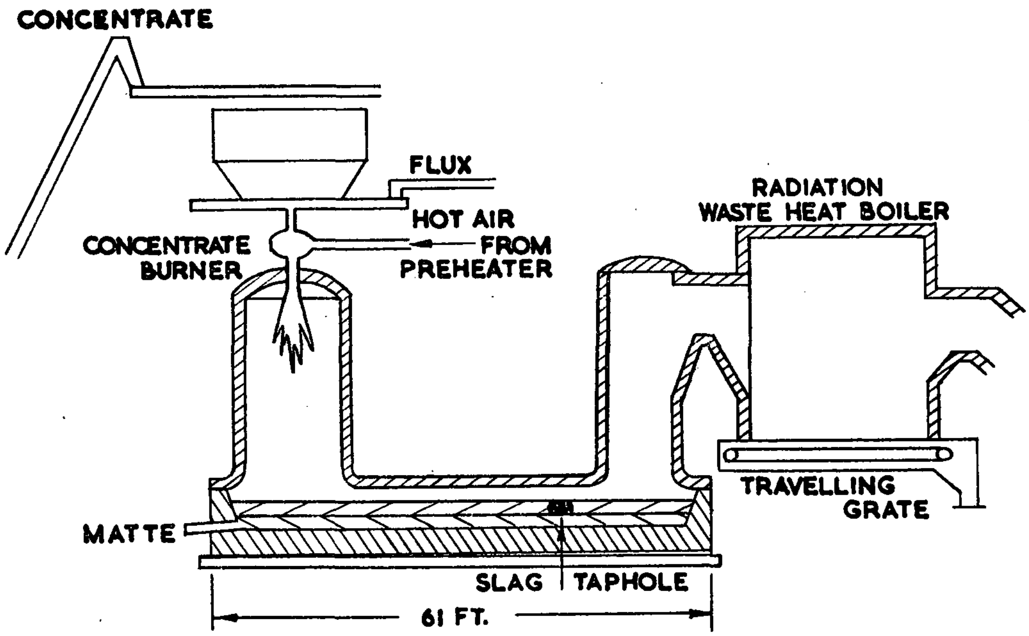

FIG. 3: HARJAVALTA FLASH SMELTING FURNACE
500 tpd concentrate

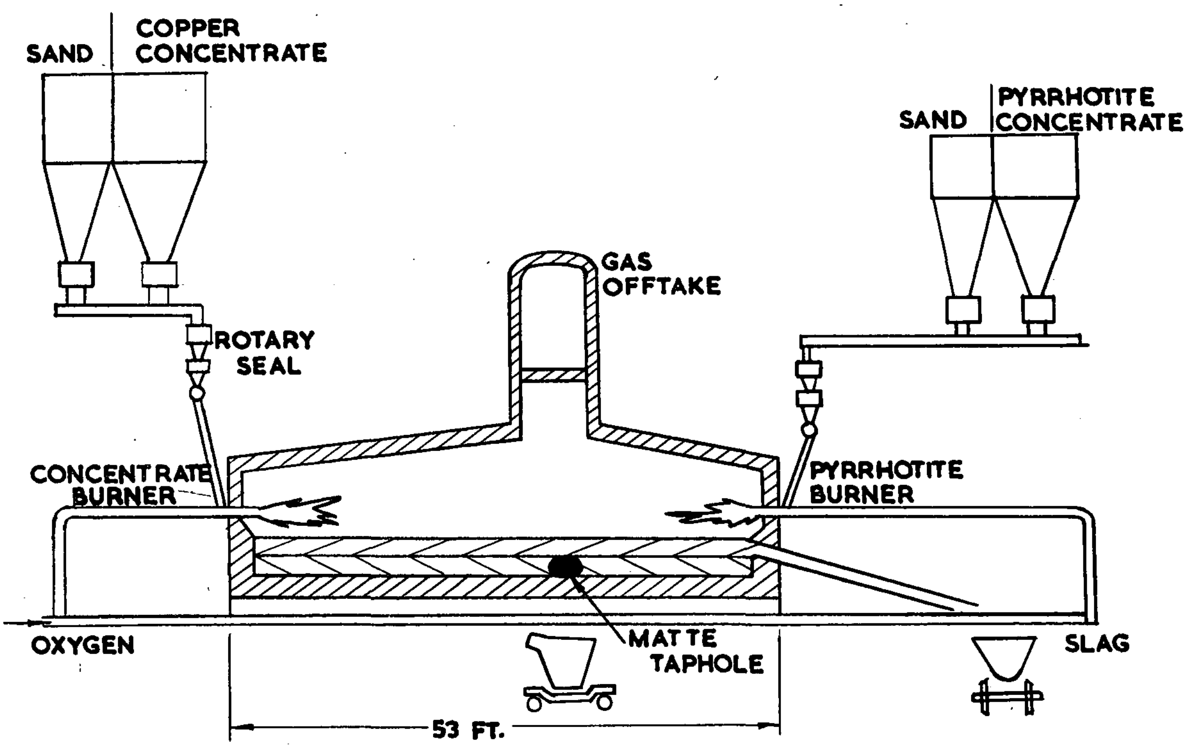

FIG. 4: INCO FLASH SMELTING FURNACE
First commercial unit
500 tpd concentrate