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# COPPER SMELTER AND REFINERY FEASIBILITY STUDY

by

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NOTE

Some information on the Anaconda-Treadwell process has become available since the report was written, and developments will be observed. This process, for which a pilot plant is almost complete, involves baking sulphide concentrates with sulphuric acid to yield elemental sulphur and a copper sulphate solution. A cyanide route has been tested for copper recovery from the sulphate solution.

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SUMMARY

Background

The South Australian Government Department of Mines commissioned Amdel to undertake a feasibility study on the establishment of smelting and refining facilities for copper in this State. The need for such a study has arisen from the intense exploration activity in the State and the desire to promote industry within South Australia.

Objective

The object of this project was to:

1. Investigate processes for producing copper and determine the most feasible process;
2. Determine the minimum volume and grade of concentrate which could be treated economically; and
3. Investigate the most suitable location for a processing plant.

Summary of Work Done

A technical study of methods of smelting copper concentrates has been made, along with an economic comparison of some of these methods at different levels of production. Electrolytic refining of blister copper and hydrometallurgical methods of producing copper from concentrates were studied, and factors influencing the location of a plant have also been examined.

Conclusions

The following general conclusions and comments for South Australian conditions are drawn from this study. These conclusions may require modification for some special cases:

1. It is definitely not economic at a copper price of $1200 per ton to build a new smelter to smelt less than 400 tons per day of 25% Cu concentrates, and a higher tonnage may be necessary to attract the investment of money in such a venture.

2. With a $1200 per ton copper price and the tariff structure as has been assumed in this study it may be economic to treat sulphide concentrates of 20% Cu grade and possibly lower (provided that the quantity of copper metal, as implied in 1. above, is adequate). Credits for sulphuric acid recovery would vary with sulphur content.

3. While a flash smelter would be preferable if an assured source of 500 tons per day of sulphide concentrate were available, and developments in the Worcra process may soon result in a more attractive process, a conventional reverberatory smelting operation, preceded by fluid-bed roasting, has been selected as the most versatile process based on current technology.

4. The reverberatory furnace could accept a reasonable proportion of oxide concentrate, but the sulphur:copper ratio of the blended feed should be at least 1:2, resulting in a matte approaching 60% Cu. A higher sulphur content would be preferable for fuel in converting.

5. If the ratio of oxide concentrate to sulphide concentrate available is too high to provide the above sulphur content, then a blast furnace to produce matte, as at Port Kembla, or black copper, as was done in campaigns at Union Minière du Haut Katanga, would have to be used — probably with pre-agglomeration.

6. It is not considered economic to build a new electrolytic refinery for a throughput of less than 50,000 tons of copper per year.

## Recommendations

If a new orebody is discovered, or an appreciable supply of copper concentrate becomes available, Amdel should be given the opportunity to modify the costs given in this report to cater for the particular geographical situation, prevailing copper price, and any innovations in smelting technology which would be applicable at that time.

# 1. INTRODUCTION

In South Australia, at the present time, mining companies are interested in copper mining and ore beneficiation. To be in a position to provide data which would make it possible, at any time, to assess the feasibility of establishing a copper smelter and refinery in this State, the South Australian Government Department of Mines commissioned this study.

The increasing demand for copper and the recent appreciable rise in the price of the metal have caused an intensification in copper exploration. Known deposits and old workings and dumps are being re-examined. Techniques of mining and concentration have improved and now mineable propositions have been created from occurrences which were formerly dismissed as mere mineralisation. Open-cut mining at a high tonnage rate has greatly reduced the cost of mining a ton of ore and developments such as autogenous grinding, larger flotation cells, more specific reagents, and automatic controls have reduced concentration costs.

Developments and innovations in the extractive metallurgy of copper are leading to larger units incorporating automation and computer control, and to the integration of the previously-separated stages of smelting.

The known ores available for exploitation in South Australia have a copper content between 0.5% and 10% with the bulk of the deposits having between 1 and 2% Cu. The ores are sulphide or oxide or a mixture of the two, the term "oxide" including carbonates and silicates as well as copper oxide. The orebodies are located in areas from 40 to 440 miles from Adelaide and at Tennant Creek 1,300 miles north of Adelaide.

A study of processes for producing copper is presented in this report. The type, size and location of plant, and the grade of concentrate are discussed. The stage to which the refining of copper is taken is discussed in relation to conditions in South Australia.

Recent developments in hydrometallurgy have been significant and point to radical changes in the treatment of complex ores. For the purpose of completeness a section is included on wet methods of concentration, extraction and refining.

# 2. SMELTING OF COPPER CONCENTRATES

## 2.1 General

Pyrometallurgical methods involve the successive or continuing oxidation of copper-iron sulphide concentrates to copper metal or the reduction of oxide concentrates. Furnaces of different types have been in use for steps in the process, and developmental work is directed towards a single continuous furnace.

Blast Furnace Smelting. The blast furnace technique has been used successfully for high-grade oxidised ores in lump form not requiring concentration, and for sulphide concentrates after roasting, smelting and agglomerating — the products being metal and slag.

Suitable oxidised ores are not available in South Australia, and the roasting and sintering step is a costly pre-treatment. Sulphide lump ore or concentrates can be smelted in a blast furnace to yield matte, but dust emission when using unsintered flotation concentrates is high. In general, the blast furnace has lost favour and has been replaced by other types of furnace, but it does have the advantage of producing a more concentrated sulphur dioxide gas, and the Momoda side-flue blast furnace is a comparatively recent development.

Electric smelting is used in Scandinavia where cheap electric power is available, but is not considered applicable in South Australia.

Reverberatory Smelting. The reverberatory furnace has had a long history of smelting sulphides. Chalcopyrite concentrates are roasted in some plants prior to smelting to remove sulphur and oxidise some of the iron present and hence increase the matte grade. Suitable blends of unroasted sulphide and oxidised concentrates have been used. The sulphur:copper ratio of the blended feed should be at least 1:2 resulting in a matte approaching 60% Cu, and a higher sulphur content would be preferable. Multihearth mechanical roasters are the classical units but in recent years there has been a strong trend away from them, and some fluid-bed roasters are now used. However, with higher grades of concentrates and developments in reverberatory furnace design there is less need for roasting. While roasting removes some of the sulphur and dries and preheats the reverberatory charge, the benefits are in many cases considered insufficient to offset the disadvantages of handling the hot dusty calcine, and the cost of roasting and dust collection. A number of new plants have eliminated roasters and charge the wet concentrate directly to reverberatory furnaces.

Fluid-bed roasting is an interesting development. The capital cost of a fluid-bed roaster is lower than that for a mechanical-hearth roaster, but the operating cost would be higher. With the properly restricted air supply and fine temperature control possible in fluid-bed roasting, the sulphur dioxide content in the flue gases may be as high as 15%. Recovery of 85% of the $SO_2$ as sulphuric acid in a plant handling 200 tons of concentrate a day has been recorded by Blair (1965) in the plant at Copperhill, Tennessee.

The removal of sulphur dioxide is becoming necessary with tighter government control over fume emission, and is becoming more economic with better plant design, process control and higher tonnages handled.

Suspension Smelting. The autogenous flash smelting furnace established at Outokumpu Oy in Finland and at Copper Cliff in Canada has been modified at Ashio in Japan (Okazoe et al, 1965). The air is preheated and extraneous fuel is only required for start-up.

The Inco flash smelter is also rendered autogenous by the use of highly oxygen-enriched blast, and produces concentrated sulphur dioxide for liquefaction and sale.

It is understood that an Outokumpu flash furnace is under construction at Mount Morgan, and Peko Mines NL at Tennant Creek are understood to be interested in the process.

Worcra Process. The Worcra process (West, 1969), developed by Dr H.K. Worner for Conzinc Riotinto of Australia (CRA) might be considered to produce metallic copper in the reverberatory furnace rather than copper matte. A diagram of the furnace is given in Figure 1. The floor of the furnace slopes towards the output end. Preheated concentrate, flux and fuel are injected into the smelting zone, and the matte, which is first formed, and then the copper sink to the bottom in the smelting and converting zones, whilst the lighter slag floats on the matte. The slag moves in a counter direction passing over a weir in the separation zone into the slag tapping chamber. The matte passes downward to the converting zone in which submerged oxygen lances supply the necessary oxygen and maintain the turbulence. Hot gases passing out of the furnace preheat the incoming charge materials, and sulphur is recovered from them. The copper in the tapping chamber contains up to 1% sulphur which is reduced by maintaining an oxidising atmosphere within the chamber to produce copper of 99.0 to 99.7% purity, with copper losses in the slag of less than 0.5%.

Many of the sulphide ores investigated, as reported by West, could be smelted without the application of external heat and smelting was facilitated with oxygen enrichment of the air. Fuel requirements without oxygen enrichment were between 20 and 40% of those of the equivalent reverberatory furnace. A further advantage claimed for the Worcra plant is that it can run economically at a significantly smaller capacity than the normal reverberatory furnace.

A plant has been constructed for full-scale trials at Port Kembla and the results will be eagerly awaited by the copper-smelting industry. However, at this stage it is not possible to make a feasibility study of the commercial potential of this process.

Rotary Reverberatory Furnace. For small scale of operation there is the possibility of using a rotary reverberatory furnace. These furnaces are used in working up intermediate products in European copper smelters and have been proposed for tin smelting by Davey (1969) who has estimated costs for a plant to produce 1500 tons of tin per year.

## 2.2 Economics of Smelting

### 2.2.1 Type and Size of Smelter

In the study of the economics of smelting, three types of furnaces are considered. The rotary reverberatory furnace (Fig.2) has been selected to complete the small throughput end of the range (50-200 tons concentrate per day, nominally 4,000-16,000 tons copper per year). The middle range is covered by the reverberatory furnace, and the large throughput (500-1000 tons concentrate per day, nominally 40,000-80,000 tons copper per year) is covered by both reverberatory and suspension smelting.

The assumptions and calculations leading to estimates of the fixed capital investment and operating costs for concentrates containing 25% copper are set out in Appendix A and the results are summarised in Table 1.

Figure 3 shows the relation between the fixed capital investment and the production capacity of each type of smelter. Figure 4 shows the relation between the operating costs per ton of copper produced and the production capacity of each type of smelter.

### 2.2.2 Grade of Concentrate

Assuming that in each plant the tonnage of concentrate treated per day will remain constant when lower-grade concentrate is supplied it will be found that the items of operating cost per ton of copper contained will vary inversely with the grade of the concentrate. Certain of the expenses may increase at a higher rate because the lower grades may require extra flux and fuel, while producing more slag and yielding less copper. The method of payment for concentrates must take this into account, and a penalty could be added to concentrates of a composition requiring extra flux. Lower sulphide content of concentrates would yield a lower credit for sulphuric acid recovery.

The copper recovery will be lower for lower grades and it is assumed that this is in direct proportion, namely:

96% for 25% Cu concentrate
95% for 20% Cu concentrate
93.7% for 15% Cu concentrate

The calculations for the operating costs of concentrates containing 25%, 20% and 15% Cu are set out in Appendix A and summarised in Table 2.

The increase in operating costs over those for concentrates of 25% Cu for both reverberatory and flash smelting are in the order of 35% for the 20% concentrate and 85% for the 15% concentrate. These increases in operating costs are met in part by the lower value of the concentrates and the tariff used, and a very similar net profit is apparent.

The working capital required when lower grades of concentrates are treated is assumed to remain at the level for 25% Cu operation. Figure 5 shows the relation between operating costs and grades of concentrates for the reverberatory furnace and flash smelter at 500 and 1000 tons per day production capacity.

2.2.3 Return on Investment

New methods of economic evaluation have been developed in recent years. For this preliminary feasibility study in which capital investment has been scaled up from equipment costs by the use of a factor it is considered that the more elaborate methods are not warranted (Allen, 1969). The return on investment calculation, however, at the blister stage of manufacture can give useful information on the economic feasibility of the process.

In its most simple form the return on investment before taxes

$$= \frac{\text{Expected Net Profit x 100}}{\text{Fixed Capital Investment + Working Capital}}$$

Working capital is taken as two months' costs of operation.

In view of the conservative predesign cost estimation factors used in this economic evaluation, it is considered that a 10% return on investment is the minimum figure for profitability.

For the return on investment calculation it is necessary to use the market price of copper and the value of copper concentrates and blister copper. The market price of copper is taken for the purpose of this project as $1200 per ton. The price of concentrates is calculated to give the producer the same net return as he would get by shipping the concentrates to the Electrolytic Refining and Smelting Company of Australia Pty Ltd at Port Kembla. The value of blister copper is based on the above market price, less quoted charges for refining, assaying and freight.

Calculations have been made for different levels of operation and for three grades of copper content of concentrates and are set out in Appendix B.

A summary of the investment returns on operations treating concentrates of 25% copper is set out in Table 3. Figure 6 shows the relation between return on investment and production capacity for reverberatory and flash smelting. Figure 7 shows the relation between return on investment and concentrate grade, and Figure 8 shows the relationship when the calculations are based on a copper price of $1500 per ton.

The capitalisation or rate of fixed capital investment per annual ton of copper production is set out in Table 4, and illustrated in Figure 9.

## 2.3 Discussion of Economics of Smelting

The process of copper smelting involves the expenditure of a large amount of capital. The amount of copper held in process is also a substantial capital item. The annual charges resulting from capital invested form a major item of operating expense which is completely outside the control of the smelter superintendent. When spread over larger volumes of product these capital charges per ton of copper are reduced.

Maintenance is another major item of expense, arbitarily taken here at 10% annually of fixed capital investment, but in any case a costly item for smelting.

From the investment return figures presented in Tables 3 and 4 it is apparent that the batch-type rotary reverberatory furnace is not a profitable means of smelting copper. The amount of fixed capital invested per annual ton of copper production for this process is considerably higher than that for either reverberatory smelting or flash smelting.

Reverberatory smelting for plants of 500 and 1000 tons per day appears economically feasible on these figures and current developments in smelting, such as the Worcra process, could further enhance the profitability of such a venture.

Flash smelting, similarly, is a feasible proposition for 500 and 1000 tons per day and any practical development which integrates smelting and converting in the same furnace could possibly make this even more attractive.

## 3. REFINING OF BLISTER COPPER

The refining of blister copper usually involves three steps, namely fire refining, electrolytic refining and cathode copper remelting. In special circumstances the latter two steps may be omitted.

### 3.1 Fire Refining

Fire refining is a preliminary step effective for removing certain impurities, e.g. sulphur, iron and zinc, and is based upon the relatively weak affinity between copper and oxygen, as compared with the affinity between oxygen and the above impurities. The fusion and air-bubbling oxidises impurities, some of which are volatilised while others float on the molten copper and may be skimmed off. Fire refining should be carried to the highest degree that is economically possible in order to have a uniform high-grade product to send to the electrolytic process, as it is generally much cheaper to eliminate impurities by a furnace treatment than by electrolytic methods. Reverberatory furnaces are usually employed although rotary converter-type furnaces have been used for small capacity plants.

When the copper has reached a given oxygen content it is deoxidised by "poling" with green wood or reformed natural gas, and cast into anodes. Blister copper of high grade is still refined to a satisfactory degree for some uses by fire refining in England by Enfield Copper Refining Company Ltd and General Metal Utilization Co. Ltd (Ryan 1968), although the market trend in the world over the past twenty years has reduced the demand for fire refined copper.

<u>Capital and Operating Costs.</u> Capital costs for the fire refining process are estimated for a plant set up separately from smelting. Costs are based on an output rate of 150 tons per day (50,000 tons copper per year), this rate being the output from a smelter handling 500 tons concentrate per day with an additional capacity of 30 tons of copper per day for material from other sources.

The equipment required for the fire refining plant would include a small reverberatory anode furnace with fuel and air ancillaries, a casting wheel and handling equipment. The fixed capital investment* would be $\$1.6 \times 10^6$, the

* See Appendix C.

operating costs $15 per ton of copper produced, and the return on investment would be 19%.

A plant scaled up to twice the output, namely 100,000 tons of anode copper per year would require a fixed capital investment of $2.6 x $10^6$. Operating costs would be $13 per ton of copper produced and the return on investment would be 23%.

## 3.2 Electrolytic Refining

In electrolytic refining the copper anode is dissolved in an electrolyte under the influence of an electric current and is deposited on the cathode as pure metal. By this means bismuth, tellurium, selenium and nickel are eliminated, and precious metals recovered. Impurities may remain behind at the electrode, dissolve and remain in solution, or fall to the bottom of the cell as a slime. Some, however, may be trapped with the copper in the electrodeposition. Provision is necessary for purification of the electrolyte because a build-up of soluble impurities increases the cell resistance, resulting in a greater power consumption. Recent developments in dialysis techniques have been incorporated in the Phelps Dodge plant in Texas (Queneau 1961). The slimes collected in the cells may contain precious metals which are recovered by furnace methods.

A plant producing 50,000 tons of cathode copper\per year using the multiple system of electrode arrangement would require 400,000 square feet of cathode surface. This could be contained in 500 cells which would cover an area of 25 acres. Equipment would include direct current rectifiers, slime treatment, electrolyte purification, cathode starting sheet manufacture, cathode copper melting furnace for final refining, and casting into saleable billets and shapes.

Capital and Operating Costs. Capital costs of the electrolytic process are based on figures quoted by Jenkins (1962) with allowance made for price increments. The fixed capital investment* would be $3.1 x $10^6$, cost of manufacture $27 per ton of copper produced, and the return on investment would be 16%.

An electrolytic plant scaled up to twice the capacity, namely, 100,000 tons of copper, would require a fixed capital investment of $5.2 million. Operating costs would be $24 per ton and the return on investment 19%.

* See Appendix D.

### 3.3 Discussion on Refining

Electrolytic refining of copper is a process which removes the remaining 1 to 2% of impurities and produces copper of a minimum purity of 99.95%. The process output is low and requires careful manual attention and supervision and high electricity consumption. The plant occupies a large area of land and ties up a great quantity of copper-in-process.

The designer of a tank house must consider the high operating labour and supervision content and high maintenance costs. Power costs are largely overshadowed by these costs. The greatest effect of design, however, is in the amount of money tied up in plant and copper which must be set aside to achieve the refining operations.

For a production of 50,000 tons of copper per year the returns on investment for fire refining and electrolytic refining are just satisfactory. The higher figures for 100,000 tons of copper per year indicate that the operation would be more attractive at the higher tonnage.

The estimated values taken for copper in the various stages of refining are based on a domestic price of copper of $1200, and would in practice move nearly proportionally with variations in this price. For this reason it is considered that a rise of 50% on this price would give only a small rise in the figures for return on investment.

## 4. HYDROMETALLURGICAL METHODS OF PRODUCING COPPER FROM CONCENTRATES

### 4.1 Leaching

#### 4.1.1 Ammonia Leaching of Sulphide Concentrates

The Sherritt-Gordon company developed an ammonia pressure leaching process for copper sulphide concentrates, similar to the process already commercially operated for nickel which is to come on stream at Kwinana WA. Despite apparently successful pilot runs in Canada and wide publicity the plant projected for the Philippines was never in fact built. Cost estimates made previously at Amdel show it to be a very expensive process capital-wise due to the stainless-steel pressure equipment, and also in operating cost due to the cost of ammonia which has to be finally disposed of as ammonium sulphate.

#### 4.1.2 Ammonia Leaching of Oxide Material

Ammonia leaching is used to recover copper from scrap in a plant in Kansas City, USA but has not been used commercially in primary metallurgy since the Kennecott ammonia-leaching operation closed down in the 1930s.

It is applicable to ore in which the native copper or copper oxide minerals cannot be separated economically from an acid-consuming gangue.

The process is most likely to be used on ore as there would be little point in preconcentration, and is hence discussed in Section 5.2.1.

### 4.1.3 Acid Leaching of Sulphide Concentrates

Some laboratory work has been done on direct leaching of sulphide concentrates under acid conditions but the more favoured approach is to roast the concentrate (with sufficient sulphur this is autogenous) under conditions which produce copper as oxide or sulphate, but minimise ferrite formation. The calcine is then leached. A pilot plant involving fluid-bed roasting, leaching and electrowinning was operated at Bagdad in Arizona but its use was discontinued.

### 4.1.4 Acid Leaching of Oxide Concentrates

Besides leaching of oxide ores directly, oxide concentrates are agitation-leached at Union Minière du Haut Katanga, Congo, and at N'changa Copper Mines, Northern Rhodesia.

## 4.2 Pregnant Solution Purification and Metal Recovery

Leaching of a copper concentrate will produce a reasonably concentrated solution so that processes such as solvent extraction would not normally be considered in this context but might be advantageous in some circumstances. Solvent extraction is further described in Section 5.2.1 which gives some cost data, to which, for concentrate (oxide Cu or cement Cu) refining, the cost of a leaching circuit should be added. Liquor feed to the solvent extraction (SX) circuit generally contains 1 to 2 g per litre of copper at pH 1 to 2, but from a concentrate leaching circuit, a higher copper concentration liquor would be obtained. This would not substantially affect the size of the SX circuit as the solvent volume would be much the same, and for preliminary costing, the capital figures quoted in Section 5.2.1 can be used.

### 4.2.1 Electrowinning

Cooper 1967 states that electrowinning is only feasible for solutions containing more than 10-25 g per litre of copper. At Katanga, iron is reduced to below 3 g per litre and some of the aluminium is removed from the feed solution to the electrowinning plant by neutralising a portion of the solution with fresh ore. At N'changa the solution is agitated with manganese ore and limestone.

Electrowinning requires 8-10 times as much power as does electrorefining but produces marketable copper directly, regenerating sulphuric acid. The insoluble antimonial-lead anodes have a service life of the order of 10 years. A new cell developed by Continental Copper and Steel Industries Inc. is capable of using a current density of 30-35 amp per square centimetre, removing more copper per pass and extending the economic copper concentration to lower levels.

Electrowinning is also used by Ranchers Exploration and Development Corporation at its Bluebird mine, Arizona and at Inspiration, Arizona.

4.2.2 Hydrogen Precipitation

Copper can be precipitated as powder from either ammoniacal or acid copper solutions by treatment with hydrogen under pressure. The former is practised by Sherritt-Gordon, and also at the plant of Whitaker Cable Corporation, Kansas City. The latter is used at the plant of Arizona Chemcopper, Bagdad, a well-publicised plant designed to produce 25 tons per day of copper powder from cement copper. The fixed capital cost of the leaching and precipitation plant, including a packaged hydrogen plant, a circulating cooling water system with an evaporative tower, a 20,000 lb per hour steam boiler with auxiliaries, a small water-treatment plant capable of supplying both softened and de-alkalysed water, plant and instrument air facilities, emergency power-generation facilities, and an inert-gas generator is stated to have been $US3.2 million. Personal communication from Mr J. Rosenbaum of US Bureau of Mines indicates, however, that the plant may not have been sufficiently well-designed for its arduous high-pressure acid duty, as the plant is frequently shut down for maintenance and the management is fortunate to obtain 4 tons per day when it is operating. In this process the pregnant solution is freed from iron by neutralising with fresh feed.

## 5. BRIEF OUTLINE OF CONCENTRATION METHODS

### 5.1 Physical Beneficiation

5.1.1 Sulphide Ores

Sulphide ores, which yield 85-90% of the world's primary copper, are generally concentrated by flotation, generally to a grade of about 25% Cu from ores ranging down to 0.5% Cu, open-cut mining and large-tonnage milling making economic the treatment of low-grade ores.

5.1.2 Oxide Ores

Oxidised ores contain copper in the form of carbonate (malachite and azurite), oxide (cuprite), basic sulphate (brochantite), silicate (chrysocolla) or native metal. If the copper mineral is sufficiently coarse gravity concentration can be used — jigging, tabling, and spiralling, but grinding is to be avoided as the carbonates being soft are easily overground. The heavy-medium cyclone installed at Amdel has proved very selective on malachite azurite ore where the copper is liberated at plus 30 mesh.

Flotation can also be used, generally involving sulphidisation to condition the copper minerals before floating.

## 5.2 Chemical Methods

5.2.1 Leaching Processes

In Situ and Dump Acid Leaching. The current status of leaching processes for copper ores is described by Cooper 1967. Low-grade ore can in favourable circumstances be leached *in situ* using the acidified ferric sulphate solution produced by the action of air and water on sulphides. Several strains of bacteria assist the process. "Project Sloop" has been designed to test underground fracturing by a nuclear explosive as a preliminary to *in situ* leaching and a first estimate of the costs of fracturing and leaching a 450-ft thick, 50-million-ton orebody amounts to $8 million, apart from cementation.

Dump leaching with sulphuric acid solutions is practised in the USA to extract copper into solutions containing an average of 1.5 g per litre of copper.

Cementation. The older method of copper precipitation for dilute leach solutions is by cementation on scrap iron. Ferric iron contributes to iron consumption which ranges from 1.3 to 4 lb per ton of copper precipitated (theoretically 0.87 lb iron per pound Cu) and operation of conventional cementation launders requires considerable labour. Kennecott's cone-type precipitator and Phelps Dodge's V-trough precipitator represent significant improvements. The latter uses sponge iron instead of scrap and a similar innovation has been developed at Rum Jungle NT.

The cement copper is generally smelted, but is refined chemically by Arizona Chemcopper (see Section 4.2.2). It might be leached with ammonia instead of acid if ammonia were cheap, and solvent extraction from either acid or ammoniacal solution could be used if required.

Dissolved copper can also be precipitated as sulphide by addition of chalcopyrite concentrates.

Solvent Extraction. Two kerosene-soluble reagents have been developed by General Mills Inc. — LIX-63 an α-hydroxime which extracts copper specifically from ammoniacal solution, and LIX-64 which extracts copper strongly and ferric iron slightly from acid solution.

Two pilot plants and one production plant are operating in Arizona. A second larger production plant is being constructed (for Bagdad Copper).

All plants extract copper from relatively dilute dump leach solutions. The copper concentration in the stripping circuit is approximately 30 g per litre of copper. This solution goes to electrowinning and is returned to the strip circuit containing 26 to 27 g per litre of copper.

The main cost of the solvent extraction (SX) process is represented by the cost of the make-up solvent. Solvent loss costs approximately 20 cents (US) per 1000 gallons (US) of aqueous feed liquor.

Current efficiency in the electro-circuit is approximately 80%. High-grade cathode copper is produced — 99.995% Cu.

The costs are summarised below:

Capital Costs: (installed including solvent)

| | SX + Electrowinning | |
|---|---|---|
| Pilot plants | 180 ton Cu/yr | $US. 200,000 |
| Production plant | 5,000 ton Cu/yr (15 ton/day, 330 day yr) | $US. 3.0 million |

Operating:

The production plant operates with approx. 6 men per shift (plus day maintenance etc.).

Ammonia Leaching. This method has been mentioned in Section 4.1.2 as applicable to an oxide ore in which there is a considerable quantity of acid-consuming gangue. The ammonia is not consumed, its loss can be reduced by steaming to less than 1 lb per ton of material treated, and copper is exclusively extracted. It is, however, not effective on chrysocolla (copper silicate). This process appears to be suited to the treatment of Burra ore.

Other Ore-Leaching Processes. The Banner Mining Company built a pilot plant In Arizona for leaching of copper ores with caustic soda. Cyanide-leaching has also been acclaimed as a breakthrough in treatment of low-grade ores.

5.2.2 Leach-Precipitation Flotation

In this process the ore is leached and copper precipitated in the pulp in metallic form by addition of sponge-iron, without solids-liquid separation. The copper grains are then recovered by flotation, along with liberated grains of copper sulphide minerals.

Costs for the leach-precipitation-flotation process applied to Copper Mountain oxidised ore are compared with other methods by Bryce, Cerigo and Jennings (1968).

5.2.3 Segregation Process

The segregation process involves heating of a finely-disseminated copper ore with a chloride and carbon to above 600°C, whereupon copper segregates as metallic grains rimming the carbon. The process is quite old, but the development of the fluid-bed Torco process will greatly enhance its application. The older versions are reviewed by Rey (1967) and the Torco process by Pinkney and Plint (1967) who cite budget operating data and direct treatment costs for plant with treatment rates of 1000 tons per day and 10,000 tons per day (approx. $2.50 and $1.50 per ton of ore).

## 6. PLANT LOCATION

### 6.1 General Discussion

The location selected for a copper treatment plant should permit the copper ore or concentrates and other raw materials to be assembled, processed and delivered to the market at minimum cost. An important consideration for the selection of the location is the basis of operation of the proposed plant, and needs to be resolved before details can be discussed. The basis of operation could be one of the following:

1. A custom treatment plant handling copper-bearing materials on a price-on-site basis; or
2. A plant associated with a major ore deposit, but open to custom treatment from other deposits.

In the former case the smelter would almost certainly be at a sea-port but preferably as near to the deposits as possible, as it could then compete for concentrates on an equal footing with interstate or overseas buyers who would require the concentrates to be freighted to a port. In the latter case the economics of smelter location would have to be calculated for each deposit, but there are again strong reasons for using an established sea-port.

With a specific deposit in mind, the selection of location should be examined in relation to the following general factors:

1. The cost or difficulty of carrying on the process at any given location;
2. The cost or difficulty of transporting materials to the location and products to the market;
3. The economic need to minimise the number of steps in handling raw materials and products; and
4. The desirability of locating the the plant in association with other plants for the complementary integration of raw materials, products, by-products and services.

In South Australia the location of the plant comes down to two types of areas, namely,

a. at an established port; or

b. at the site of a major ore deposit.

Possible locations for consideration under a. are Port Adelaide, Port Pirie, Port Augusta and Wallaroo. The prima facie advantage of these locations is the already available shipping facilities, because it must be kept in mind that the bulk of the copper metal produced would be shipped either overseas or interstate. Another advantage is the abundant supply of cooling water from the sea and the ease of effluent disposal. Port Adelaide and Port Pirie would have the additional advantages of being major industrial centres.

The location of the plant at the site of a major ore deposit b. would eliminate freight costs on concentrates, reduce the number of handling steps and possibly allow a lower grade of concentrate to be used. Freight costs on copper products, fuel, raw materials and stores would be set against these advantages.

## 6.2 Particular Cost Factors

Some of the more important cost factors affecting plant location are now discussed.

Markets. If the product from the plant is electrolytic copper, consideration should be given to the complementary establishment of a plant to manufacture copper products. Location near markets in Adelaide may be desirable. In any case much of the copper would be exported or shipped interstate and a port site would be an advantage.

If the product is blister copper or anode copper then this would be shipped to one or other of the two existing refineries in the Eastern States or to refineries overseas, and a port site would be an advantage.

Processing Materials and Stores. The processing materials and stores required depend on the type of process selected. A pyrometallurgical process would require fluxes such as limestone and silica, possibly pyritic copper ores, furnace bricks and other heavy stores. A leaching process requires lixivants such as acids or ammonia, and other bulk materials such as scrap iron or solvents. A port site would be desirable for most of these materials and proximity to manufacturers and suppliers would be a distinct advantage. Location at Port Adelaide would be the most favourable port.

Labour. Wage rates would be similar at each location, although wages and salaries would be higher in remoter areas. Skilled labour and supervision resources would be more easily available at Port Adelaide and Port Pirie and the reliability and efficiency of labour may be better in established industrial centres.

Electricity. In considering processing to the blister copper stage, electricity would be available at any of the suggested port sites at a rate which, through negotiation with the State Government, should be comparable. Supply to remote areas could possibly cost more.

If electrolytic refining or electrowinning is included in the process, then the cheapest possible rate would be required and a location near a power station such as at Port Adelaide or Port Augusta may be an advantage.

Fuel. Depending on the process and the economics of the supply, the choice of fuel would include oil, coal, and natural gas. Natural gas is available at Port Adelaide and Port Augusta and would require a distribution main at any other location. A port location would be an advantage when considering other fuels.

Water. Fresh water is available at the port sites at rates which would be comparable. Large volumes of low cost cooling water would be available at coastal sites and also at Murray Bridge in relation to the Kanmantoo deposit.

Rail Transportation. All locations under consideration are close to railways. However, the change in gauge is a serious problem and the need to avoid transferring freight from one gauge to another would be a prominent factor in selecting the location.

Shipping. Data for the ports under consideration are given in Table 5. Port Augusta is at a disadvantage compared with the other three ports.

Waste and By-product Disposal. Reverberatory slag disposal at any site should not prove difficult, although accumulation for possible future retreatment may be difficult at Port Adelaide. Solid filling will be in demand at Port Adelaide and Port Pirie for many years.

The disposal of water from a smelter would probably have no problems at any of the ports. However, acidic effluents from an electrolytic refinery or leaching plant may be a problem at any port.

Air pollution would be controlled at any location. If the sulphur dioxide evolved from a smelter is not to be collected, a stack of substantial height would be required and attention should be given to foundation requirements and any height limitations. Where the process includes recovery of $SO_2$, the various alternatives for marketing should be considered. These may include liquefied $SO_2$, sulphuric acid, or as gas piped to an existing acid plant such as at Port Adelaide or Port Pirie. The proximity to a user such as a superphosphate plant or a copper leaching operation would be desirable.

Copper sulphate if produced would require an assured market with, for example, the fertilizer industry.

Site and Buildings. Real estate costs would be higher at Port Adelaide. Erection costs could be higher at any location other than Port Adelaide.

Climate. The process is not affected by climatic conditions. Summer heat of the more northern areas may have some minor influence on the efficiency of labour.

## 6.3 Preferred Location

The discussion of plant location has been in general terms because no particular location can seriously be determined until it is known from which deposit the plant will draw the bulk of its material. The advantages of locating the proposed smelter and refinery at either Port Adelaide or Port Pirie outweigh those of other locations particularly in regard to shipping facilities, sulphuric acid manufacture and the industrial nature of the cities. Port Adelaide has preferred advantages in regard to natural gas supply, electric power station and fertilizer manufacture. However, for concentrates railed from northern deposits the freight for the extra distance from Port Pirie to Port Adelaide would cost approximately $7 per ton, which may be sufficient to lose the supply to the competition of other orebuyers.

## 7. CONCLUSIONS

The following general conclusions and comments for South Australian conditions are drawn from this study. These conclusions may require modification for some special cases:

1. It is definitely not economic at a copper price of $1200 per ton to build a new smelter to smelt less than 400 tons per day of 25% Cu concentrates, and a higher tonnage may be necessary to attract the investment of money in such a venture.

2. With a $1200 per ton copper price and the tariff structure as has been assumed in this study it may be economic to treat sulphide concentrates of 20% Cu grade and possibly lower (provided that the quantity of copper metal, as implied in 1. above, is adequate). Credits for sulphuric acid recovery would vary with sulphur content.

3. While a flash smelter would be preferable if an assured source of 500 tons per day of sulphide concentrate were available, and developments in the Worcra process may soon result in a more attractive process, a conventional reverberatory smelting operation, preceded by fluid-bed roasting, has been selected as the most versatile process based on current technology.

4. The reverberatory furnace could accept a reasonable proportion of oxide concentrate, but the sulphur:copper ratio of the blended feed should be at least 1:2, resulting in a matte approaching 60% Cu. A higher sulphur content would be preferable for fuel in converting.

5. If the ratio of oxide concentrate to sulphide concentrate available is too high to provide the above sulphur content, then a blast furnace to produce matte, as at Port Kembla, or black copper, as was done in campaigns at Union Minière du Haut Katanga, would have to be used — probably with pre-agglomeration.

6. It is not considered economic to build a new electrolytic refinery for a throughput of less than 50,000 tons of copper per year.

## 8. RECOMMENDATIONS

If a new orebody is discovered, or an appreciable supply of copper concentrate becomes available, Amdel should be given the opportunity to modify the costs given in this report to cater for the particular geographical situation, prevailing copper price, and any innovations in smelting technology which would be applicable at that time.

## 9. ACKNOWLEDGEMENT

L.H. Goldney has contributed solvent extraction data based on his recent visit to Arizona.

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## APPENDIX A

## CAPITAL INVESTMENT AND OPERATING COSTS

### Rotary Reverberatory Smelting

Assumptions: 50 tons of 25% Cu concentrates per day smelted at 96% recovery yields 12.0 tons copper per day, ie. 3960 tons (nominally 4,000 tons) per year of 330 operating days.

Fixed Capital Investment. For a daily throughput of 50 tons of concentrate the throughput data given by Schwartz (1956) indicate that two rotary reverberatory furnaces would be required for smelting to matte and one for converting the matte to blister and possibly fire-refining. A fairly simple plant of low capital cost would be indicated for this low tonnage, and the following list would be the essential items of plant required:

| | $ |
|---|---|
| Raw material handling | 80,000 |
| Charging hopper with chute to each furnace | 18,000 |
| Three rotary reverberatory furnaces 3 m x 3 m, with oil or natural gas burners @ $120,000 | 360,000 |
| Three air blowers | 48,000 |
| Oil storage | 10,000 |
| Dust collection, fan, stack | 50,000 |
| Travelling crane | 30,000 |
| Blister casting and slag granulation disposal equipment | 50,000 |
| Purchased Equipment Cost | 646,000 |

Fixed Capital Investment - using Lang's factor of 3.63* = $2.3 x 10^6$

The addition of a fourth rotary reverberatory furnace and a 8 ft dia. by 18 ft long converter to treat the matte produced in the rotary reverberatory furnaces would increase the throughput to approximately 100 tons per day. The additional equipment would cost $300,000, which would bring the total fixed capital investment for 100 tons per day to $3.4 x 10^6$.

* ARIES, R.S. and NEWTON, R.D. (1955). Chemical Engineering Cost Estimation, McGraw Hill.
Lang's factor is a factor by which the purchased equipment cost may be multiplied to give a budget estimate fixed capital investment. For a solid-fluid process a value of 3.63 has been found by experience to be reasonable.

The addition of four rotary reverberatory furnaces would increase the throughput to 200 tons per day. The converter above would produce 50 tons of blister copper per day. Additional equipment would cost $500,000.

Fixed capital investment for 200 tons per day would be $\$5.2 \times 10^6$.

Operating Costs

| | $ per Ton Copper | | |
|---|---|---|---|
| Concentrates of 25% Cu treated, | 50 tpd | 100 tpd | 200 tpd |
| Labour and supervision[a] | 40 | 31 | 26 |
| Utilities, fuel and power[b] | 16 | 16 | 16 |
| Materials (silica sand flux)[b] | 4 | 4 | 4 |
| Maintenance[c] | 56 | 41 | 31 |
| Indirect manufacturing cost[d] | 27 | 20 | 16 |
| Fixed manufacturing cost[e] | 72 | 54 | 41 |
| Add value of copper lost assuming 96% recovery | 48 | 48 | 48 |
| Operating Costs per ton copper | 263 | 214 | 182 |

(a) Labour and supervision -

| | Tons 25% Cu conc/24-hr day | | |
|---|---|---|---|
| | 50 | 100 | 200 |
| Shift crew: furnace attendants | 7 | 11 | 20 |
| casters | 1 | 1 | 2 |
| crane drivers | 1 | 2 | 2 |
| | 9 | 14 | 24 |
| Day Gang | 3 | 5 | 7 |
| Man-hours per ton | 20 | 15.7 | 13.2 |
| Cost @ $1.8 per man-hour | 36 | 28.3 | 23.8 |
| Supervision (10% of labour) | 3.6 | 2.8 | 2.4 |
| | 39.6 | 31.1 | 26.2 |

(b) Utilities -

| | $ per Ton Cu |
|---|---|
| Fuel oil (14% of conc. Wt.) $20 per ton | 11 |
| Electricity and water (Gooden, 1966) | 5 |
| | 16 |
| Materials - | |
| Silica sand flux, 2 tons/ton Cu @ $2/ton | 4 |

(c) Maintenance -

10% of fixed capital investment.

(d) Indirect Manufacturing Cost -

75% of cost of labour and supervision

(e) Fixed Manufacturing Cost -

13% of fixed capital investment.

## Reverberatory Smelting

Three levels of production are costed, viz. 300, 500 and 1000 tons of concentrate per day. In view of more stringent air-pollution requirements and the desirability of recovering as much sulphur dioxide as possible at acid-making concentration a fluid-bed roaster has been assumed ahead of the reverberatory furnace. For a feed-rate of 300 tons per day a 12 ft diameter fluid-bed roaster has been assumed (as used at Tennessee Copper Co.) a reverberatory furnace 105 ft by 25 ft and a converter 9 ft dia. by 20 ft. For 500 tons per day the fluid-bed roaster would be 16 ft diameter, the reverberatory 105 ft by 30 ft and the converters (one spare) 11 ft dia. by 25 ft.

For 1000 tons per day two 16 ft fluid-bed roasters are assumed, two reverberatory furnaces (one spare) 105 ft by 35 ft and two converters (one spare) 13 ft dia. by 30 ft.

### Equipment Required

| | Estimated Costs Thousands of Dollars | | |
|---|---|---|---|
| | 300 tpd | 500 tpd | 1000 tpd |
| Roasting (excluding acid plant) - | | | |
| Fluid-bed roaster | 90 | 135 | 280 |
| Fuel burner storage | 15 | 20 | 40 |
| Air blower | 15 | 20 | 40 |
| Calcine surge storage & conveyors | 10 | 15 | 20 |
| Smelting - | | | |
| Furnace | 250 | 420 | 900 |
| Air blowers and preheater | 50 | 70 | 100 |
| Waste heat boiler and power gen. | 200 | 250 | 300 |
| Flue dust collection | 10 | 15 | 20 |
| Stack | 100 | 110 | 150 |
| Slag granulation | 5 | 10 | 10 |
| Matte handling | 15 | 20 | 40 |
| Crane | 40 | 45 | 50 |
| Converting - | | | |
| Converter | 150 | 200 | 400 |
| Flue dust collection | 50 | 60 | 120 |
| Slag handling | 20 | 20 | 40 |
| Blister casting | 100 | 150 | 300 |
| Air blowers | 40 | 60 | 120 |
| Equipment cost | 1,160 | 1,620 | 2,930 |
| Fixed Capital Investment, $ million ( by use of Lang's factor = 3.63)* | 4.2 | 5.9 | 10.6 |

* Aries & Newton, loc cit.

| | \$ Million | | |
|---|---|---|---|
| | 300 tpd | 500 tpd | 1000 tpd |
| Sulphuric Acid Plant (including electrostatic precipitators) - | | | |
| Fixed capital investment[a] | 2.2 | 3.2 | 5.5 |
| Total Fixed Capital Investment | 6.4 | 9.1 | 16.1 |

(a) For sulphide concentrates of 25% Cu, sulphuric acid produced

$$= \frac{98}{32} \times \frac{1}{2} \text{ of Cu content} = 1.5 \times \text{tonnage of Cu}^{*}$$

| Plant Size tons/day | Fixed Capital Investment per daily ton of acid |
|---|---|
| 110 | \$20,000 |
| 190 | \$17,000 |
| 370 | \$15,000 |

* Assume roasting stage -

$$2CuFeS_2 \rightarrow Cu_2S + FeS + FeO + 2SO_2$$

$\uparrow 2(64)$ $\uparrow 2(2\times32)$ $\uparrow 2(32)$

## Operating Costs

| | \$ per Ton Copper | | |
|---|---|---|---|
| Concentrates of 25% Cu treated - | 300 tpd | 500 tpd | 1000 tpd |
| Labour and supervision[a] | 11 | 7 | 5 |
| Utilities, fuel and power[b] | 27 | 27 | 27 |
| Materials[b] | 4 | 4 | 4 |
| Maintenance[c] | 25 | 22 | 20 |
| Indirect manufacturing costs[d] | 8 | 5 | 4 |
| Fixed manufacturing costs[e] | 33 | 29 | 25 |
| Add value of copper lost assuming 96% recovery | *48* | *48* | *48* |
| Less value sulphuric acid recovery from roaster[f] | *33* | *34* | *35* |
| Operating Costs per ton copper | 123 | 108 | 98 |

(a) Labour and supervision -

| Shift crew: | men | Tons of 25% Cu conc/24-hr day 300 tpd | 500 tpd | 1000 tpd |
|---|---|---|---|---|
| Fluid bed - | 2 | | | |
| Reverb. furnace - (tapping, silica spraying, fuel supply) | 5 | 15 | 16 | 18 |
| Converter - (leading hand, punchers, crane drivers & chasers, skimmer) | 8 | | | |

| | $ per Ton Copper | | |
|---|---|---|---|
| | 300 tpd | 500 tpd | 1000 tpd |
| Day gang | 4 | 5 | 7 |
| Man hours per ton | 5.5 | 3.5 | 2.6 |
| Labour at $1.8 per hour | 9.9 | 6.3 | 4.7 |
| Supervision, 10% of labour | 1.0 | 0.6 | 0.5 |
| Cost of labour & supervision | 10.9 | 6.9 | 5.2 |

(b) Utilities -

| | $ per Ton Cu |
|---|---|
| Fuel oil, 1.01 ton/ton Cu, $20/ton | 20.2 |
| Electricity, 500 kWh/ton Cu, $0.02/kWh | 10 |
| | 30 |
| Credit, waste heat - 150 kWh/ton Cu | 3 |
| Net cost of utilities | 27 |

Materials -

| | |
|---|---|
| Silica sand flux, 2 ton/ton Cu, $2/ton | 4 |

(c) Maintenance -

10% of fixed capital investment

(d) Indirect Manufacturing Cost

75% of cost of labour and supervision

(e) Fixed Manufacturing Cost -

13% of fixed capital investment.

(f) 1.5 Tons of sulphuric acid at $27 per ton produced per ton of Cu less cost of labour, power etc.

Operating Cost for Lower-Grade Concentrates. The calculations for concentrates of 20 and 15% copper are set out briefly below:

| | Cost $ per Ton Cu | | |
|---|---|---|---|
| Concentrates treated per day, | 300 tpd | 500 tpd | 1000 tpd |
| *(Manufacturing cost @ 25% Cu* | *108* | *94* | *85 )* |
| Manufacturing cost @ 20% Cu | 135 | 117 | 106 |
| Add value of copper lost assuming 95% recovery | *60* | *60* | *60* |

| | Cost $ per Ton Cu | | |
|---|---|---|---|
| | 300 tpd | 500 tpd | 1000 tpd |
| Less value of sulphuric acid recovery[(a)] | 33 | 34 | 35 |
| Operating cost for 20% Cu concentrate | 162 | 143 | 131 |
| Manufacturing cost @ 15% Cu | 180 | 157 | 142 |
| Add value of copper lost assuming 93.7% recovery | 76 | 76 | 76 |
| Less value of sulphuric acid recovery[(a)] | 33 | 34 | 35 |
| Operating cost for 15% Cu concentrate | 223 | 199 | 183 |

(a) Sulphuric acid recovery is taken as the same as for 25% Cu concentrates on the assumption that the concentrate is more pyritic.

## Suspension Smelting

The suspension or flash-smelting furnace at Ashio in Japan has a charge capacity of 500 tons of dried concentrates per day. Equipment costs have been estimated for a plant of this size and for one of 1000 tons per day. The equipment includes materials-handling, drying, flash-smelter, electric furnace for slag cleaning, waste heat boiler and power generation, $SO_2$ recovery, dust collection, converting and blister casting.

Equipment Required

| | Estimated Costs, Thousands of $ | |
|---|---|---|
| | 500 tons conc/day | 1000 tons conc/day |
| Dryer | 25 | 45 |
| Oil burner | 4 | 5 |
| Air blower | 10 | 20 |
| Dust collection | 15 | 25 |
| Surge storage and conveyors, and feeder | 12 | 18 |
| Flash smelting furnace | 600 | 800 |
| Oil storage & preheating | 5 | 8 |
| Air blowers & preheater | 100 | 180 |
| Waste heat boiler | 200 | 400 |
| Electricity generators | 250 | 500 |
| Stack | 200 | 300 |
| Slag granulation | 15 | 20 |
| Slag electric furnace | 25 | 40 |
| Crane | 50 | 80 |
| Matte handling | 15 | 80 |
| Converter | 200 | 400 |
| Slag handling | 20 | 40 |

| | Estimated Costs, Thousands of $ | |
|---|---|---|
| | 500 tons conc/day | 1000 tons conc/day |
| Blister casting | 150 | 300 |
| Dust collection | 60 | 120 |
| Air blowers | 60 | 120 |
| Total Equipment Cost | 2016 | 3441 |
| Fixed Capital Investment, - $ million (by use of Lang's factor = 3.63)* | 7.3 | 12.5 |
| Sulphuric acid plant[(a)], - $ million (including electrostatic precipitators) | 3.0 | 5.0 |
| Total Fixed Capital Investment $ million | 10.3 | 17.5 |

(a) Assume sulphur recovered = 2/3 copper content, i.e. 2 tons acid per ton of copper.

## Operating Costs

| | $ per Ton Blister Copper | |
|---|---|---|
| Concentrates of 25% Cu treated, | 500 tpd | 1000 tpd |
| Labour and supervision[(a)] | 5 | 2 |
| Utilities, fuel and power[(b)] | 12 | 12 |
| Materials[(b)] | 4 | 4 |
| Maintenance[(c)] | 25 | 21 |
| Indirect manufacturing costs[(d)] | 4 | 2 |
| Fixed manufacturing costs[(e)] | 33 | 28 |
| Add value of copper lost for assumed 96% recovery | *48* | *48* |
| Less credit for sulphuric acid recovery[(f)] | *46* | *48* |
| Operating Costs | 85 | 69 |

(a) Labour and supervision -

| | | Tons of 25% Cu conc/24-hr day | |
|---|---|---|---|
| | | 500 tpd | 1000 tpd |
| Shift crew, | men | 12 | 14 |
| Day gang, | men | 4 | 6 |
| Man-hours per ton | | 2.5 | 1.1 |
| Labour @ $1.8/hr | | $ 4.5 | $ 2.0 |
| Supervision 10% of labour | | $ 0.5 | $ 0.2 |
| Cost of labour and supervision | | $ 5.0 | $ 2.2 |

* Aries & Newton, loc cit.

(b) Utilities -

| | | $/Ton Cu |
|---|---|---|
| Fuel oil, 0.17 ton/ton Cu | $20/ton | 3.4 |
| Electricity, 622 kWh/ton Cu, | $ 0.02/kWh | 12.4 |
| less credit waste heat | $ 3.4/ton | 3.4 |
| | | 12 |

Materials -
Silica sand flux, 2 tons/ton Cu, $2/ton - $4

(c) Maintenance -
10% of fixed capital investment.

(d) Indirect Manufacturing Cost
75% of cost of labour and supervision.

(e) Fixed Manufacturing Cost
13% of fixed capital investment.

(f) Credit for $SO_2$ recovery -

2 ton sulphuric acid per ton of copper produced at $27 per ton.

| | | |
|---|---|---|
| for 500 ton conc/day level, value = | $54 | |
| less - cost of labour, power etc. | 8 | $46 |
| for 1000 ton conc/day level, value = | $54 | |
| less - cost of labour, power etc. | 6 | $48 |

Operating Costs for Lower-Grade Concentrates. The calculations for concentrates of 20 and 15% copper are set out briefly below:

| | Cost $ per Ton Cu | |
|---|---|---|
| Concentrates treated, | 500 tpd | 1000 tpd |
| *(Manufacturing costs at 25% Cu* | *83* | *69 )* |
| Manufacturing costs at 20% Cu | 104 | 86 |
| Add value of Cu lost assuming 95% recovery | *60* | *60* |
| Less value of sulphuric acid assuming recovery same as for 25% Cu | *46* | *48* |
| Operating cost, 20% Cu conc. | 118 | 98 |
| Manufacturing cost at 15% Cu | 138 | 115 |
| Add value of Cu lost assuming 93.7% recovery | *76* | *76* |
| Less value of sulphiric acid assuming same recovery as for 25% Cu | *46* | *48* |
| Operating cost, 15% Cu conc. | 168 | 143 |

## APPENDIX B

## CALCULATIONS FOR RETURN ON INVESTMENT

The market price of electrolytic copper is taken as $1200 per ton and forms the basis of these calculations.

Value of Blister Copper at SA Plant

| | $/ton |
|---|---|
| Price of electrolytic copper | 1200 |
| Deduct estimated refining and assay charges | 60 |
| freight SA to Port Kembla | 30 |
| Estimated value of blister = | 1110 |

Concentrates 25% Cu at SA Plant. The calculation is based on ER & S rates which include the following:

| | |
|---|---|
| Draftage: | 1% deduction from weight |
| Handling: | 1.3 units deduction from grade |
| Price basis: | 96% of Australian price (say, $1200) |
| Smelting charge: | $9 per ton of concentrate |
| Converting and refining: | $0.90 per unit paid for |

Estimated charge for converting only is taken at $0.30 per unit paid for.

Value of copper paid for = (25 x 0.99 - 1.3) x 0.96 x 1200

= 270

less smelting charge 9

less converting charge 0.3 x 23.45 = 7

less freight SA to Pt Kembla 15

$239 per ton concentrate

or $956 per ton Cu contained.

Similar calculations for concentrates of 20% and 15% Cu, yield values of $920 and $854 per ton of copper respectively.

Return on Investment. It is assumed that the production year is 330 days.

## Rotary Reverberatory Smelting

| | | | |
|---|---|---|---|
| Level of operation: | 50 tons concentrate per day | | |
| Grade of concentrate: | 25% | 20% | 15% |
| Yearly output of copper: | 4000 | 3100 | 2300 |
| | $\$ \times 10^6$ | $\$ \times 10^6$ | $\$ \times 10^6$ |
| Sales value @ $1110/ton | 4.44 | 3.44 | 2.55 |
| less operating cost/ton Cu, | 263 | 338 | 458 |
| concentrate value | 956 | 920 | 854 |
| | 1219 | 1258 | 1312 |
| Cost for year | 4.88 | 3.90 | 3.02 |
| Net profit | -0.44 | -0.46 | -0.47 |
| Fixed Capital Investment | 2.3 | 2.3 | 2.3 |
| Working Capital 2/12 of expenses | 0.8 | 0.6 | 0.5 |
| | 3.1 | 2.9 | 2.8 |
| Return on Investment | -14% | -15% | -15% |
| Similar calculations for 100 tons concentrate/day | | | |
| Yield return on investment | - 9% | -10% | -10% |
| Calculations for 200 tons concentrate/day | | | |
| Return on investment | - 6% | - 5% | - 6% |

## Reverberatory Smelting

| | | | |
|---|---|---|---|
| Level of operation: | 300 tons concentrate per day | | |
| Grade of concentrate: | 25% | 20% | 15% |
| Yearly output of copper: | 24,000 | 19,000 | 14,000 |
| | $\$ \times 10^6$ | $\$ \times 10^6$ | $\$ \times 10^6$ |
| Sales value @ $1110/ton | 26.6 | 21.1 | 15.5 |
| less operating cost/ton Cu | 123 | 162 | 223 |
| concentrate value | 956 | 920 | 854 |
| | 1079 | 1082 | 1077 |
| Annual cost | 25.9 | 20.6 | 15.1 |
| Net profit | 0.7 | 0.5 | 0.4 |
| Fixed Capital Investment | 6.4 | - | - |
| Working Capital 2/12 of expenses | 4.3 | - | - |
| | 10.7 | - | - |
| Return on Investment | 7% | 5% | 4% |

Similar calculations for 500 and 1000 tons per day give the following results:

| | | | |
|---|---|---|---|
| 500 tons concentrate per day | 12% | 9% | 8% |
| 1000 tons concentrate per day | 15% | 12% | 11% |

Flash Smelting

| | | | |
|---|---|---|---|
| Level of operation: | 500 tons concentrate per day | | |
| Grade of concentrate: | 25% | 20% | 15% |
| Yearly output of copper: | 40,000 | 31,000 | 23,000 |
| | $\$ \times 10^6$ | $\$ \times 10^6$ | $\$ \times 10^6$ |
| Sales value @ $1110/ton | 44.4 | 34.4 | 25.5 |
| less operating cost | *85* | *118* | *168* |
| concentrate value | *956* | *920* | *854* |
| | *1041* | *1038* | *1022* |
| Annual cost | 41.7 | 32.1 | 23.4 |
| Net profit | 2.7 | 2.3 | 2.1 |
| Fixed Capital | 10.3 | - | - |
| Working Capital 2/12 expenses | 7.0 | - | - |
| Return on Investment | 16% | 13% | 12% |

Similar calculations for 1000 tons per day give the following results 23% 19% 18%

# APPENDIX C

## CAPITAL AND OPERATING COSTS FOR FIRE REFINING

Equipment:

| | |
|---|---|
| Reverberatory furnace | \$300,000 |
| Anode casting wheel | \$ 70,000 |
| Handling equipment | \$ 80,000 |
| | \$450,000 |

Fixed Capital Investment -
(by use of Lang's factor = 3.63)* = $\$1.6 \times 10^6$

Operating Costs:

| | | \$/ton Cu |
|---|---|---|
| Labour, (10 men x 4 shifts = 1.6 man hr/ton) | | 3 |
| Materials | | 1 |
| Fuel and power | | 2 |
| Maintenance | 10% FCI | 3 |
| Indirect expenses | 75% labour | 2 |
| Fixed capital expenses | 13% FCI | 4 |
| Operating Cost for Fire Refining | | 15 |

Return on Investment

| | | |
|---|---|---|
| 50,000 ton sales value @ \$1150 | | $\$57.5 \times 10^6$ |
| Expenses: Operating costs | 15 | |
| Blister | 1110 | |
| | 1125 | $56.3 \times 10^6$ |
| FCI | $1.6 \times 10^6$ | |
| Working capital 1/12 expenses | $4.7 \times 10^6$ | |
| | $6.3 \times 10^6$ | |
| Return on Investment | | 19% |

* ARIES, R.S. and NEWTON, R.D. (1955). Chemical Engineering Cost Estimation, McGraw Hill.
Lang's factor is a factor by which the purchased equipment cost may be multiplied to give a budget estimate fixed capital investment. For a solid-fluid process a value of 3.63 has been found by experience to be reasonable.

# APPENDIX D

## CAPITAL AND OPERATING COSTS FOR ELECTROLYTIC REFINING

| | | |
|---|---|---|
| Electrolytic plant | | $\$2.3 \times 10^6$ |
| Cathode furnace | | $0.7 \times 10^6$ |
| Land | | $0.1 \times 10^6$ |
| | Fixed Capital Investment | $\$3.1 \times 10^6$ |

| Operating Costs | | $/ton copper |
|---|---|---|
| Labour | | 4 |
| Power, electrolytic | 230 kWh/ton Cu @ $0.02 | 6 |
| Maintenance, materials, fuel | | 6 |
| Indirect expenses | | 3 |
| Fixed Capital Expenses | | 8 |
| | Operating Cost | 27 |

Return on Investment

| | | | |
|---|---|---|---|
| 50,000 ton sales, | value $1200 | | $60.0 \times 10^6$ |
| Expenses: | Operating costs | 27 | |
| | Anode copper | 1150 | |
| | | 1177 | $58.8 \times 10^6$ |
| | | Net profit | $1.2 \times 10^6$ |
| FCI | | | $3.1 \times 10^6$ |
| Working capital (7% of annual value of production)* | | | $4.2 \times 10^6$ |
| | | | $7.3 \times 10^6$ |
| | | Return on Investment | 16% |

* JENKINS, J.C. (1962). Plant Design for Electrolytic Copper Refining, *Aust. Engr* 54, April, 42-6.

TABLES 1 to 5

FIGURES 1 to 9

TABLE 1: CAPITAL AND OPERATING COSTS

| Process | Level of Operation tons/day (a) | Fixed Capital Investment $ million | Operating Cost $/ton Cu |
|---|---|---|---|
| Rotary Reverberatory | 50 | 2.3 | 263 |
| | 100 | 3.4 | 214 |
| | 200 | 5.2 | 182 |
| Reverberatory | 300 | 6.4 | 123 |
| | 500 | 9.1 | 108 |
| | 1000 | 16.1 | 98 |
| Flash Smelter | 500 | 10.3 | 85 |
| | 1000 | 17.5 | 69 |

(a) Nominal tons concentrate per day.

TABLE 2: OPERATING COSTS FOR THREE GRADES OF CONCENTRATES

| Process | Level of Operation tons/day(a) | Operating Costs(b) | | |
|---|---|---|---|---|
| | | 25% | 20% | 15% |
| Rotary Reverberatory | 50 | 263 | 338 | 458 |
| | 100 | 214 | 275 | 375 |
| | 200 | 182 | 236 | 320 |
| Reverberatory | 300 | 123 | 162 | 223 |
| | 500 | 108 | 143 | 199 |
| | 1000 | 98 | 131 | 183 |
| Flash Smelter | 500 | 85 | 118 | 168 |
| | 1000 | 69 | 98 | 143 |

(a) Nominal tons concentrate per day.

(b) Operating costs, $/ton Cu for grades of concentrates (% Cu).

TABLE 3: RETURN ON INVESTMENT
Smelting and converting

| Process | Level of Operation tons/day[a] | Return on Investment % |
|---|---|---|
| Rotary Reverberatory | 50 | not profitable |
| | 100 | - ditto - |
| | 200 | " |
| Reverberatory | 300 | 7 |
| | 500 | 12 |
| | 1000 | 15 |
| Flash Smelter | 500 | 16 |
| | 1000 | 23 |

(a) Nominal tons concentrate per day, 25% Cu content.

TABLE 4: CAPITALIZATION

| Process | Level of Operation tons/day (a) | Fixed Capital Investment / Annual Copper Production $/ton |
|---|---|---|
| Rotary Reverberatory | 50 | 570 |
| | 100 | 430 |
| | 200 | 320 |
| Reverberatory | 300 | 270 |
| | 500 | 220 |
| | 1000 | 200 |
| Flash Smelter | 500 | 250 |
| | 1000 | 210 |

(a) Nominal tons of concentrate per day, 25% Cu content.

TABLE 5: DATA FOR SELECTED SOUTH AUSTRALIAN HARBOURS
Supplied by Marine and Harbours Board

| Harbour | Maximum Overall Length of Vessel | Depth in Channel[a] | Depth at Wharf[a] | Comments |
|---|---|---|---|---|
| Port Adelaide | - | - | - | Will accommodate largest coastal vessel, say 60,000 tons. |
| Port Pirie | 580 ft | 21 ft | 27 ft | Accommodate vessel of the order of 20,000 tons. |
| Wallaroo | 650 ft | - | 27.8 ft | - |
| Port Augusta | 420 ft | 16 ft | 20 ft[b] | Vessel presently used to carry Peko concentrates was specially built for harbour limitations. |

(a) Height of tide to be added to figures quoted.

(b) Maximum.

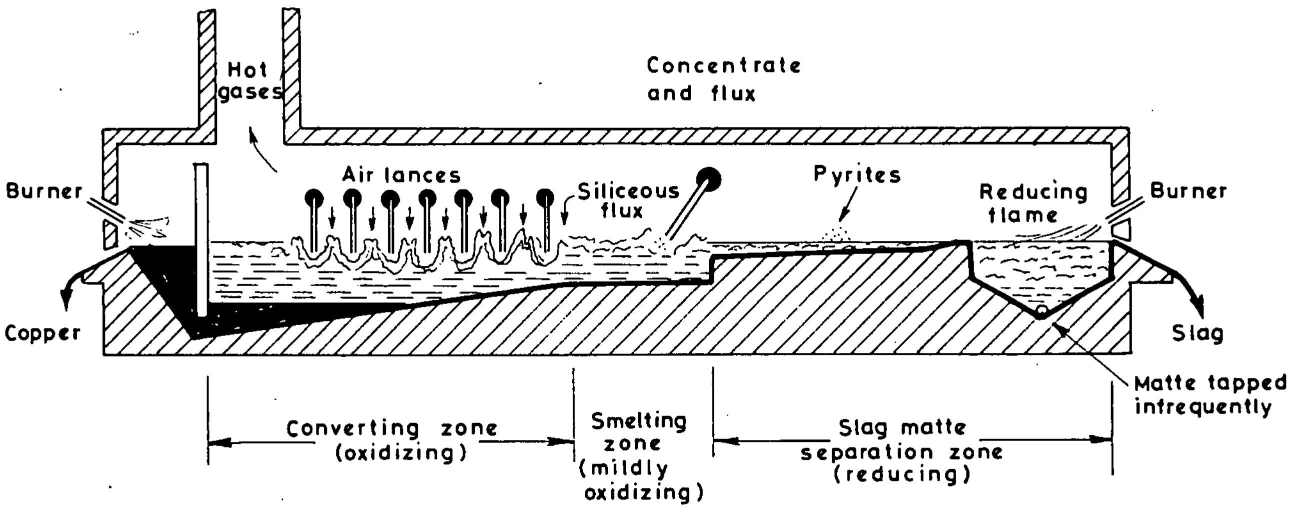


FIGURE 1: VERTICAL SECTION THROUGH A WORCRA COPPER SMELTING CONVERTER (after West 1969)

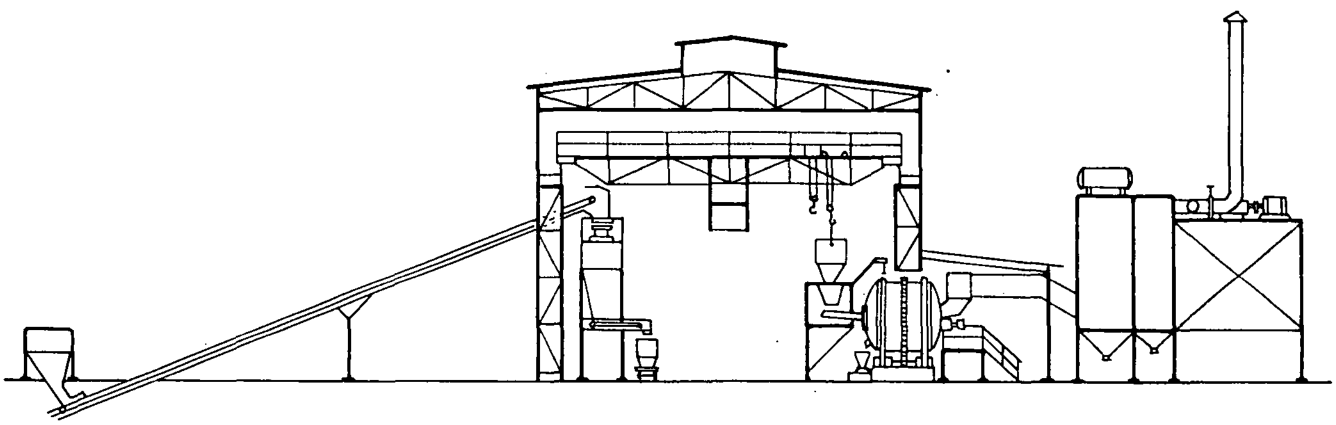

FIGURE 2: SHORT ROTARY REVERBERATORY FURNACE WITH WASTE HEAT BOILER IN A COPPER SMELTING PLANT
(after Lurgi Gesellschaft für Chemie und Hüttenwesen M.B.H.)

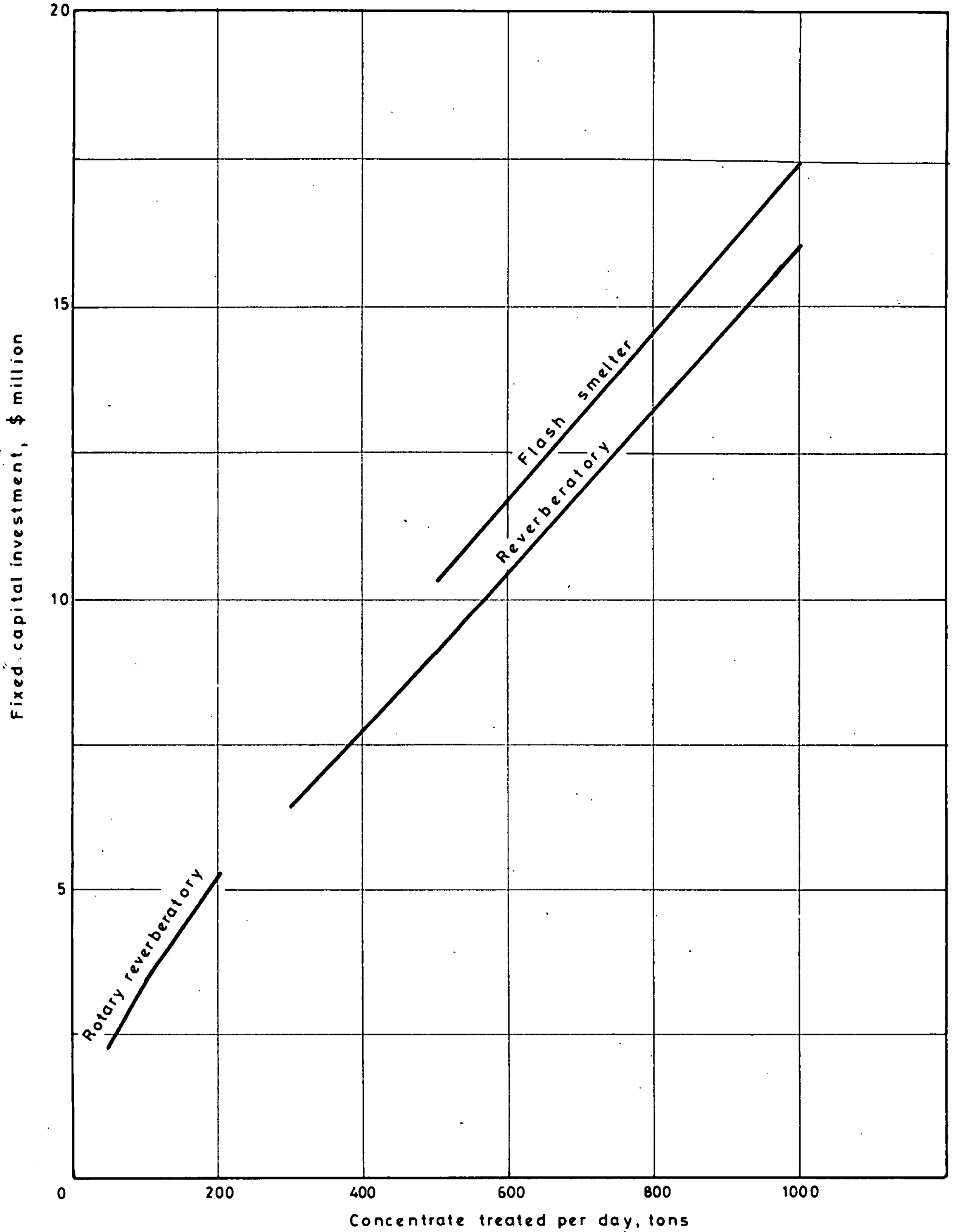


FIGURE 3: FIXED CAPITAL INVESTMENT

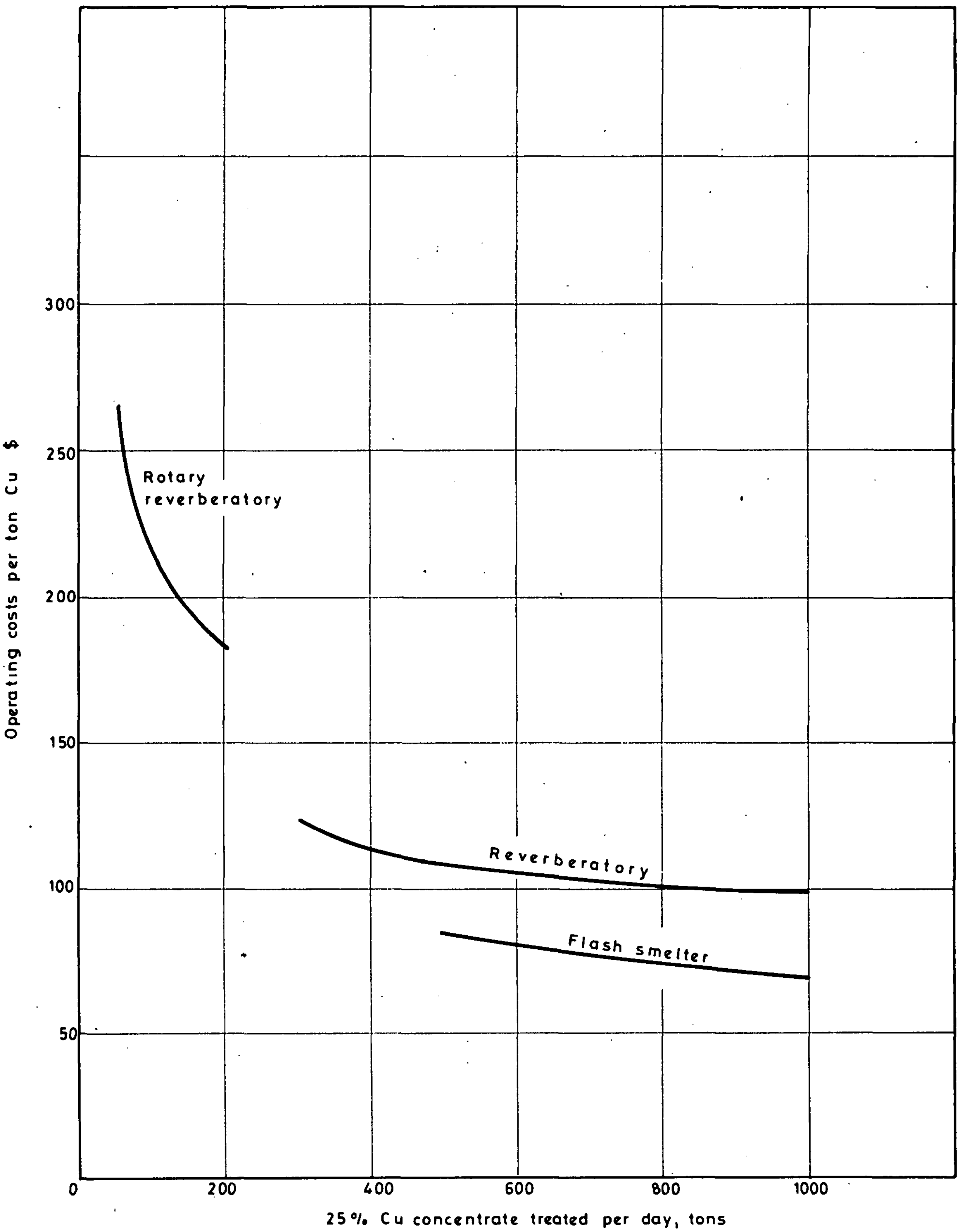


FIGURE 4: OPERATING COSTS PER TON OF Cu

RG.5083(6as)

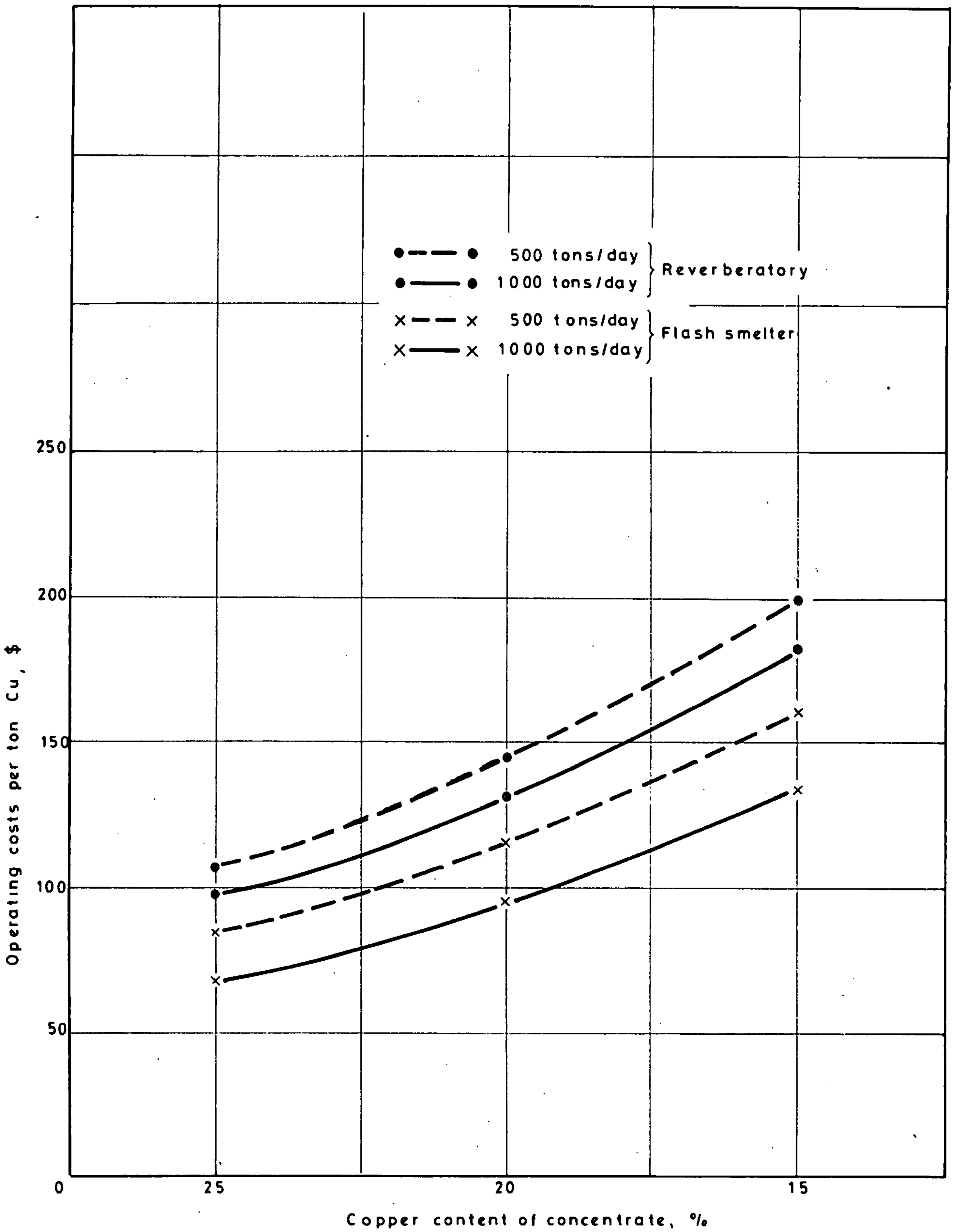


FIGURE 5: EFFECT OF GRADE OF CONCENTRATES ON OPERATING COSTS

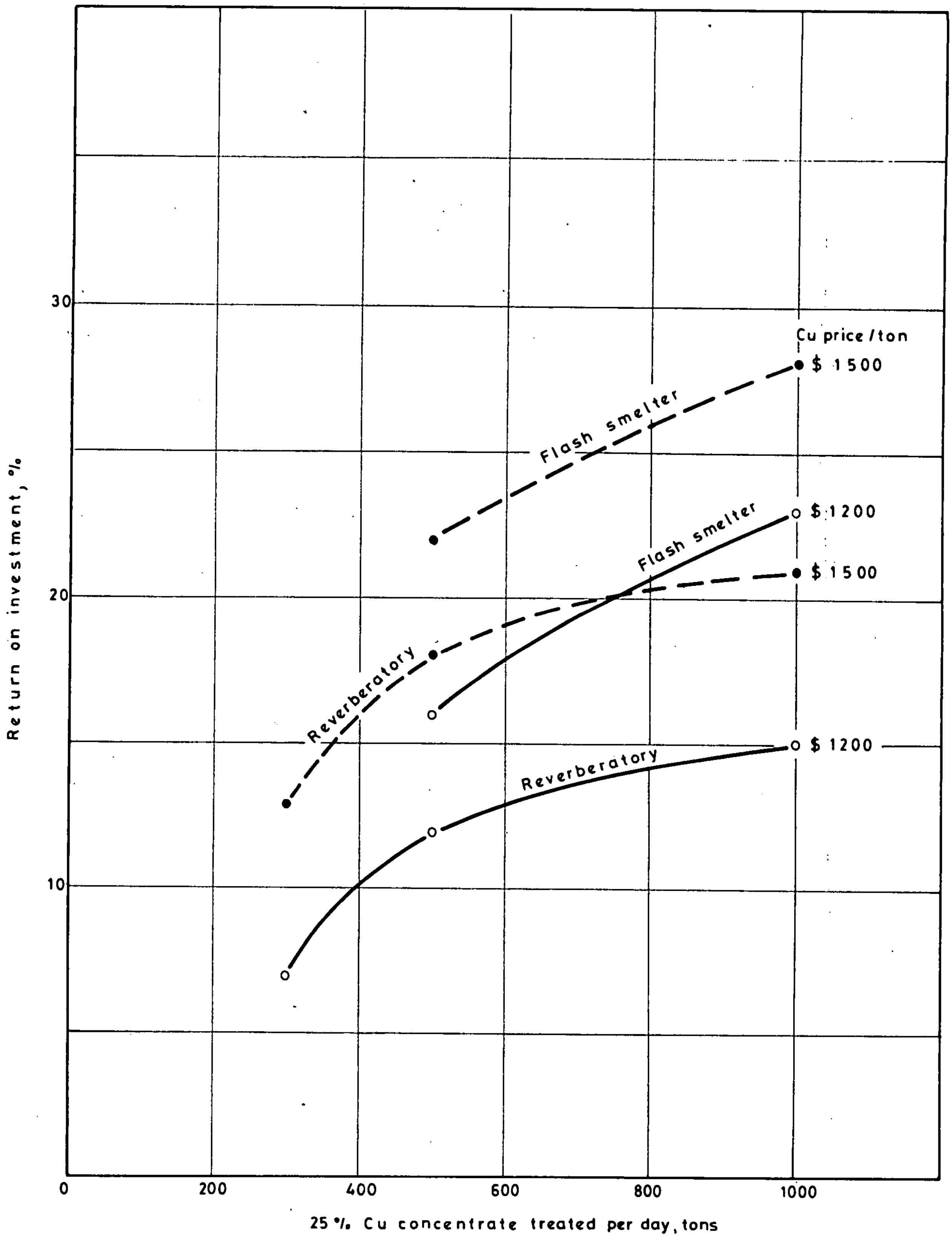


FIGURE 6: RETURN ON INVESTMENT

RG 5085 (695)

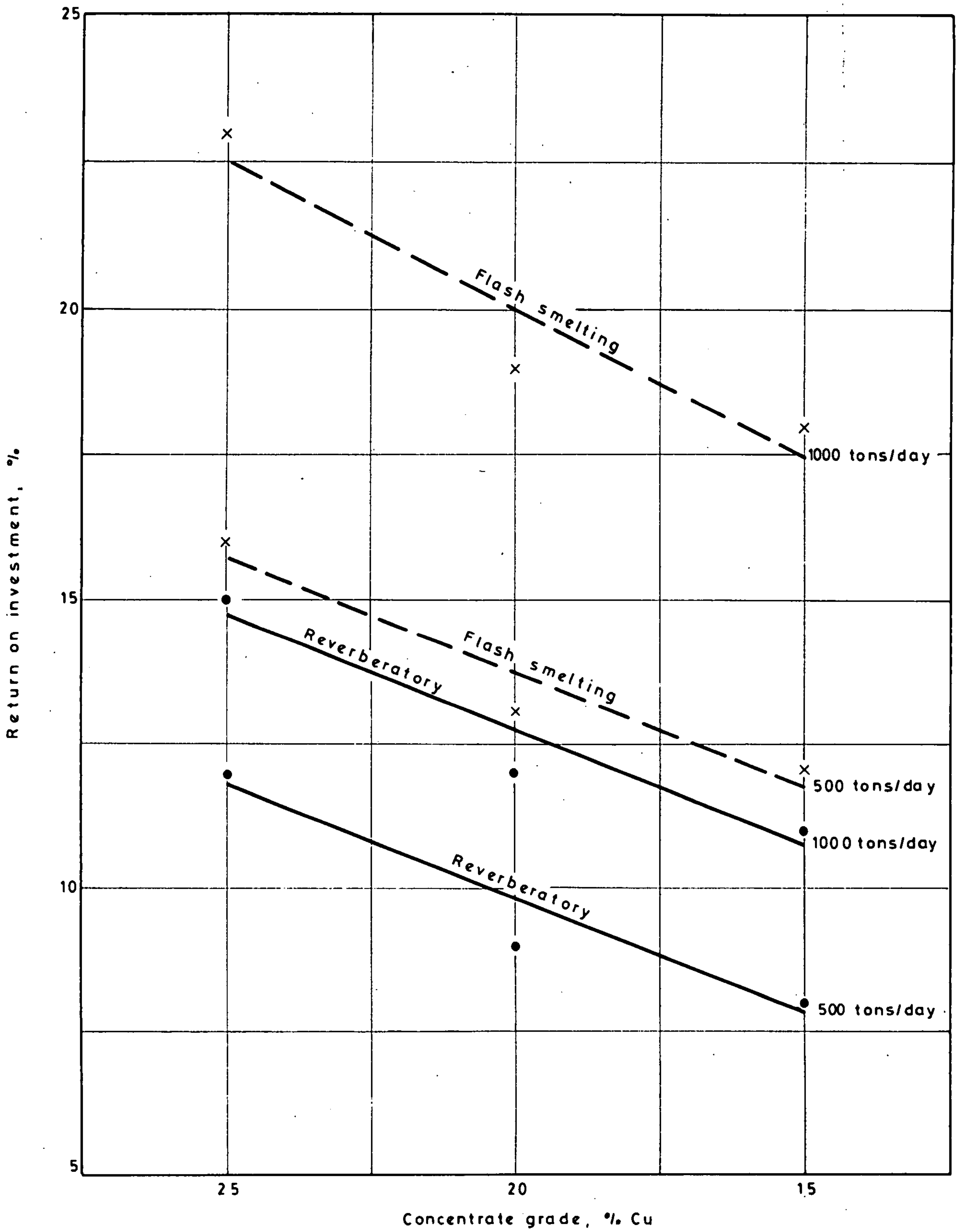


FIGURE 7: EFFECT OF GRADE OF CONCENTRATE ON RETURN ON INVESTMENT
Based on copper price $ 1200/ton

RG 5086 (6/65)

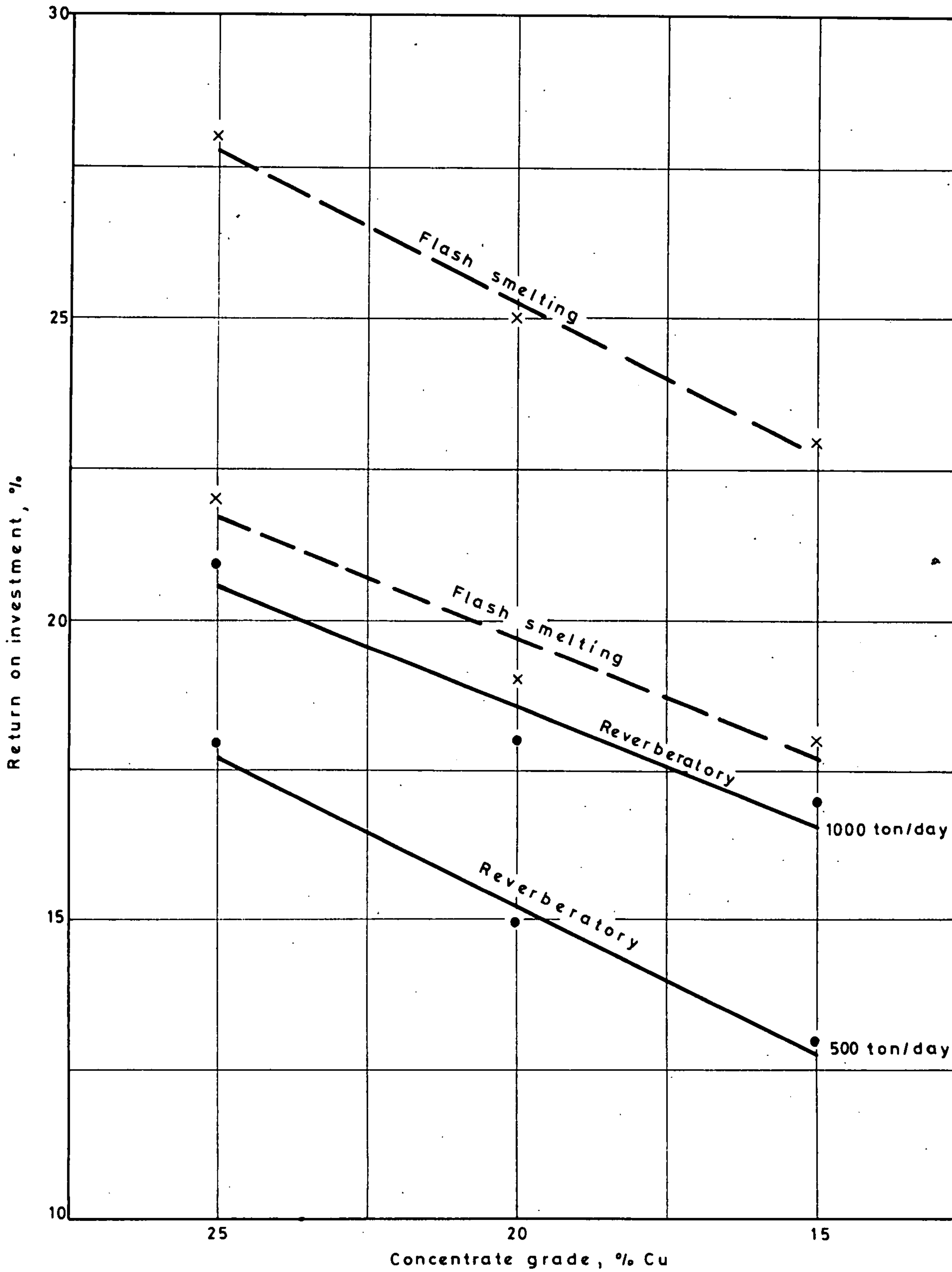


FIGURE 8: EFFECT OF CONCENTRATE GRADE ON RETURN ON INVESTMENT
Based on copper price $ 1500/ton

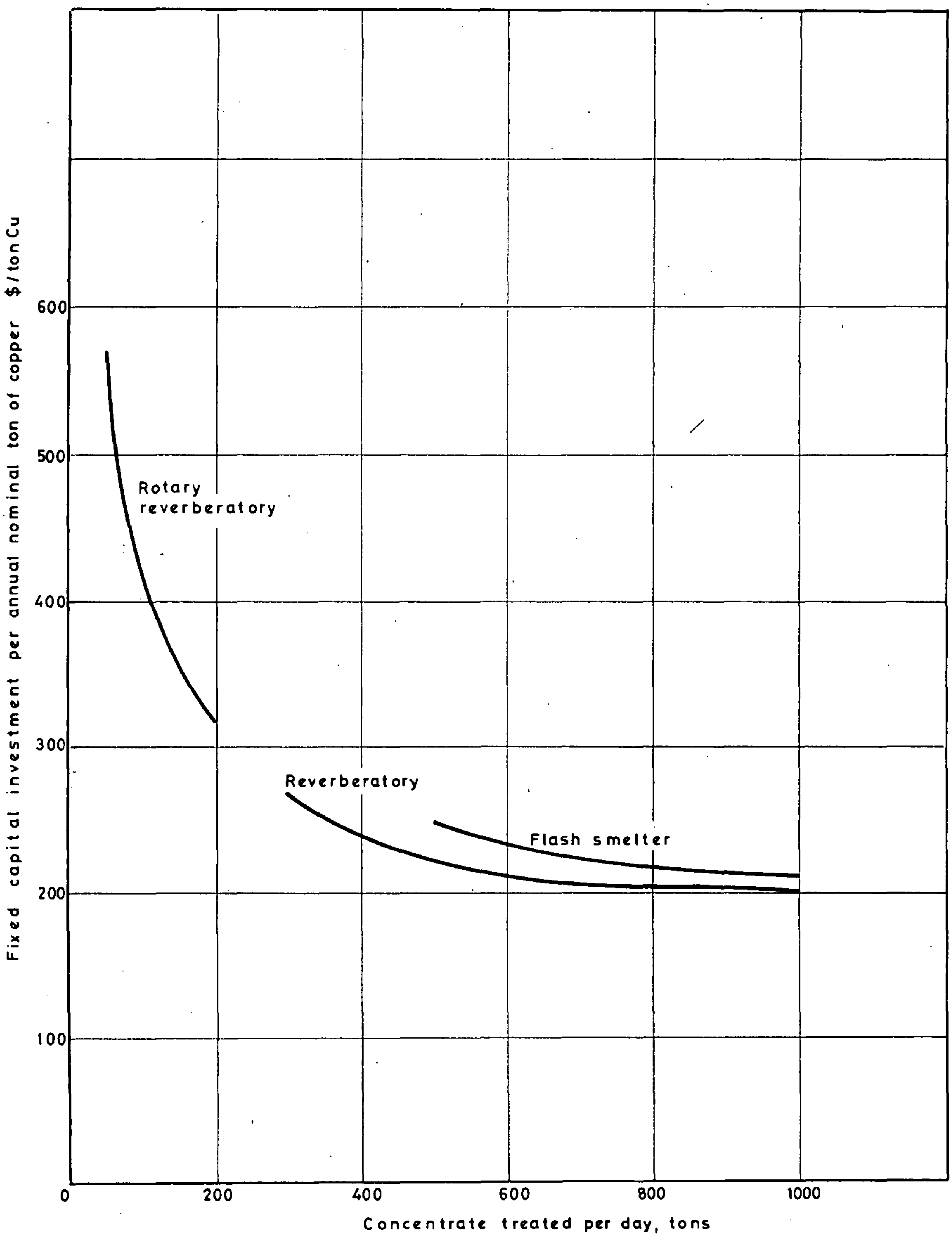


FIGURE 9: CAPITALIZATION FOR VARIOUS LEVELS OF OPERATION

RG 5088 (6/95)