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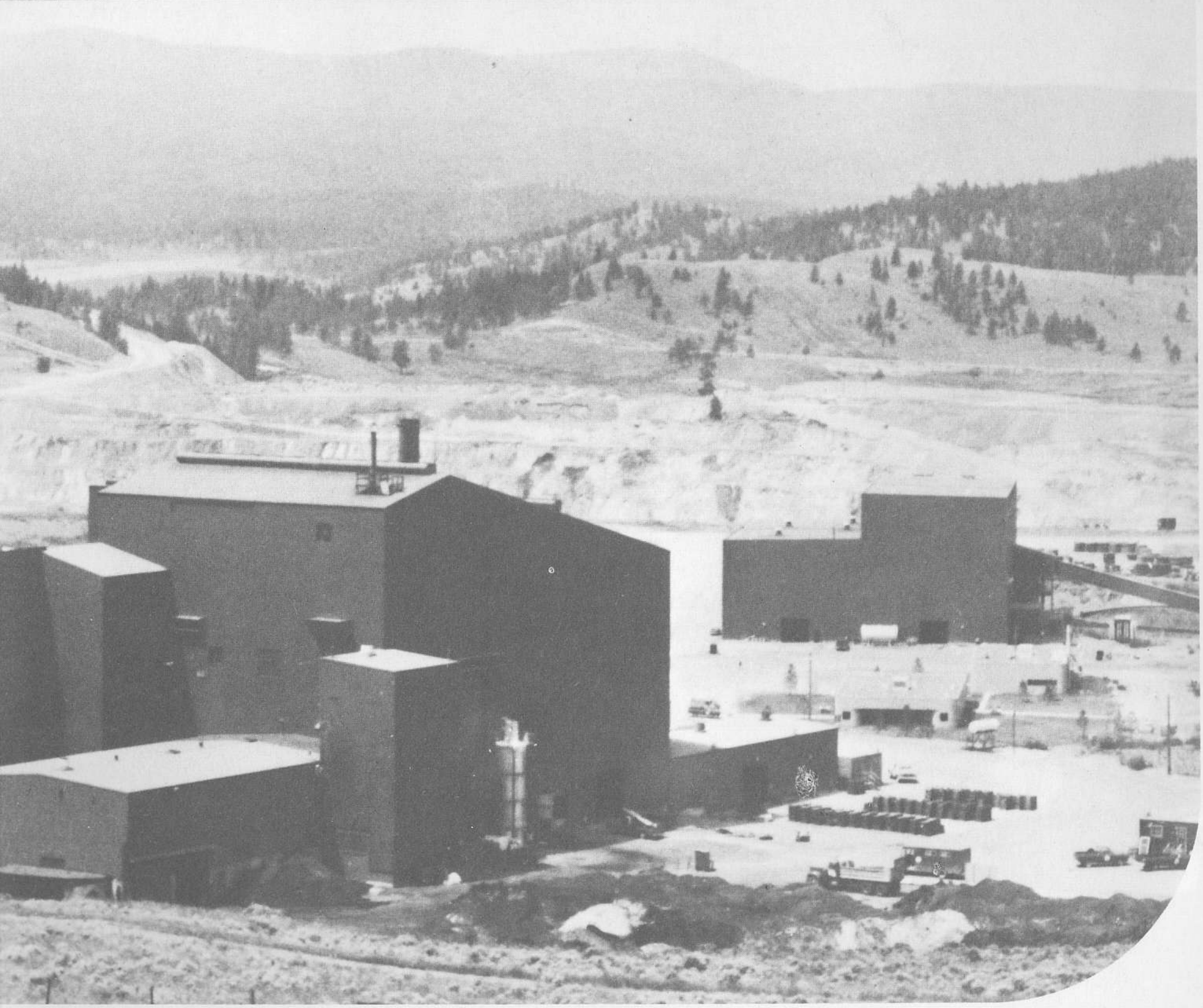
November 5 – 6, 1980



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AND

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SECTION 1

INTRODUCTION

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The commissioning of Teck Corporation's Afton mine and smelter in the early 1970's represented the first contemporary smelting technology installed in the Province of British Columbia. Prior to this, smelting of copper ores had occurred at several locations in B.C., notably at Crofton and Ladysmith in the early 1900's, and at Granby's Anyox mine in 1914. The older operations were closed in the 1930's.

Concurrent with the mineral commodities boom in the late 1960's and early 1970's, and associated with the opening up of large-scale surface mining for copper in porphyry deposits, several companies investigated the feasibility of smelting copper in British Columbia. However, the prevailing economic circumstances were far more attractive for the mining and milling of copper concentrates than to carry on through the prime metal stage producing blister copper, anode or cathode. To some extent, these conditions still prevail today. Where B.C. concentrates are sold under contract, treatment and refining charges by the smelters are still reasonably attractive. The disposal of sulfuric acid, a by-product of pyrometallurgical processes for smelting sulfide copper concentrates, still poses a potential economic cost on smelting within B.C. due to the limited size of domestic markets for acid.

However, there are indications that some of these adverse circumstances are changing. As smelter capacity in Japan and the U.S. reaches full utilization, the economics of new greenfields smelter operations are tending to favour locations closer to the sources of concentrate in supply regions.

This seminar focuses on pyrometallurgical and hydrometallurgical technologies for the extraction of copper that might be applicable to British Columbia ores and concentrates. We asked the authors to address the subject from the viewpoint of process description, energy utilization, environmental features and economics. With this perspective, we hoped that mining companies and other agencies represented in our seminar audience would increase their awareness of the major features of contemporary smelting technology.

For reasons of time, we had to restrict the presentations to those technologies presently available and marketable on a commercial basis. Many other technologies are being developed and are approaching commercial availability. We are hopeful that over the next few years, as parties seriously interested in smelting copper in British Columbia begin investigations, and as the Province's many advantages as a site for further processing become better understood, the whole range of appropriate technologies will be examined.

SECTION II

INCO OXYGEN FLASH SMELTING OF COPPER CONCENTRATES

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INCO OXYGEN FLASH SMELTING OF COPPER CONCENTRATE

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ABSTRACT

INCO Metals Company has operated an Oxygen Flash Smelting Process at its Copper Cliff, Ontario, Smelter for almost 30 years. This paper describes the existing operations with particular emphasis on the copper concentrate flash smelting process. The details of the process metallurgy are also discussed with special attention being paid to the effect of molten converter slag recycle on the flash furnace operation and methods of controlling the furnace matte grade. This slag recycle feature has been a standard operating practice at INCO's Copper Cliff installation for about one and a half years. The process economics, as they relate to the treatment of a typical B.C. concentrate, are described. This data includes a mass balance, direct process energy requirements and estimated operating and capital costs. The paper concludes with a summary of the major advantages of the INCO Oxygen Flash Smelting Process.

COPPER CLIFF OPERATION

INTRODUCTION

The Ontario Division of the INCO Metals Company typically mines about 15 million tonnes of ore per year and produces about 150 000 tonnes of nickel, 140 000 tonnes of copper, almost as much as the nickel, and 600 000 tonnes of iron ore. INCO also recovers cobalt, Se & Te, gold, silver and other precious metals. In addition 80 000 tonnes of liquid SO_2 and 800 000 tonnes of sulfuric acid are recovered by Canadian Industries Limited from the process off-gases at Copper Cliff, Ontario.

COPPER CLIFF SMELTER OPERATIONS

Nickel and copper concentrates produced at INCO's mills in the Sudbury district are supplied to the Copper Cliff Smelter. The smelter circuits are quite complex because of the copper in the nickel concentrate and nickel in the copper concentrate (1-4). A simplified version of these two parallel smelting operations is shown in Figure 1.

The fluxed nickel concentrate is treated in multi-hearth roasters, and reverberatory furnaces. The slag produced is discarded, while the matte, is delivered to Pierce Smith converters. The converter product, called Bessemer Matte, and containing about 50% Ni & 25% Cu, is partially cooled in designated converters prior to delivery molten to the matte casting, cooling and separation processes (5) which yield three products, a nickel sulfide concentrate, a copper sulfide (chalcocite) concentrate and a metallics fraction. The nickel converter slag is recycled molten to the reverberatory furnaces.

The fluxed copper concentrate, containing about 30% Cu and 1% Ni is fluid bed dried prior to injection into the flash furnace using a stream of oxygen. The resulting off-gas, containing 70-80% SO_2, is delivered to liquefaction and the matte, typically containing 40-50% Cu, is delivered to Pierce Smith converters. The furnace slag, containing 0.6-0.7% Cu, is discarded. In the converters, the furnace matte is blown with flux and the chalcocite concentrate (6) is produced in matte separation, to yield a blister copper, which is delivered molten to the copper refinery for anode furnacing and casting. A portion of the copper converter slags are reverted to the flash furnace molten while the remainder is reverted to the nickel circuit.

Until quite recently none of the converter slag was reverted to the flash furnace as this stream was used to bleed nickel from the copper circuit. However, with improved separation in the mill, and as the result of slag reversion testwork, about 50% of the converter slag is now being reverted to the flash furnace.

FLASH SMELTING PROCESS DEVELOPMENT

Laboratory and Pilot Plant testing of the INCO Oxygen Flash Smelting Process was initiated at Copper Cliff in 1946 (7, 8). Based on the data obtained, the first commercial furnace was commissioned in early 1952. This unit was rated at 450 tonnes per day of concentrate and had the dimensions shown in Figure 2. The second furnace with double the capacity and slightly larger in size was commissioned in late 1953. This unit operated until 1968 when its capacity was increased by extending the shorter end of the furnace to make it symmetrical about the center line. This unit was rated at 1360 tonnes per day.

The development and present operational practice of INCO's Oxygen Flash Smelting Process has been extensively covered in the technical literature (9-18). Its application to the smelting of nickel concentrates has also been reported (17-19).

CURRENT FLASH FURNACE CHARACTERISTICS

The current operating furnace is 24.4 m (80') long, 7.3 m (24') wide and 5.7 m (18') high at the ends, has a volume of 600 m^3 (21 200 cu. ft.) and a hearth area of 140 m^2 (1,500 sq. ft.). The roof is a sprung arch with the off-gases leaving through the center uptake. About 20% of the sidewall is cooled with water jackets located in the mid furnace area. The furnace is totally enclosed in a mild steel shell except for the slag line area which is externally cooled by air jets. Slag is skimmed through a single hole at one end of the furnace and matte is tapped through one of two holes located on one side of the furnace also near the mid point.

The furnace refractories consist of a magnesite bottom and chrome magnesite in the walls and roof.

FLUID BED DRYING OF COPPER CONCENTRATES

At Copper Cliff, the concentrates, at about 8% moisture, are delivered by conveyor along with about 10% sand flux to the top of the fluid bed dryer. These units are fired by natural gas, though light oil has also been used. The dried product, at about 0.1% moisture, is conveyed by the fluidizing air to the baghouses where the furnace feed is collected for delivery to the furnace feed bins. The flowsheet is shown in Figure 3.

There are two dryers each, nominally 2.4 m (8') diameter at the hearth with fluidizing shafts about 4.9 m (16') high. They are each capable of drying about 900 tonnes per day of concentrate. The windbox temperature is typically 315°C (600°F) the bed temperature 120°C (250°F) and the off-gas 105°C (200°F). The fluidizing blowers have a capacity of 37 600 Nm^3/hr (25 000 SCFM) and the combustion chamber rating is 5.2 x 10^6 cal/hr (20.5 x 10^6 Btu/hr).

The dust loading at the baghouse inlet is typically about 900 G/Nm^3 (370 Gr/SCF) while the outlet is about 0.5 G/Nm^3 (0.19 GR/SCF) for a dust collection efficiency of about 99.95%. The unit is designed to handle about 49 000 Nm^3/Hr (32 500 SCFM) with typical inlet and outlet temperatures of 105°C (220°F) and 90°C (195°F) respectively. The bag differential varies between 0.5-2.0 KPA (2-9" WC) depending on the condition of the bags, which are typically changed once per year.

FLASH SMELTING OF COPPER CONCENTRATES

The flash smelting flowsheet is also depicted in Figure 3. The dried feed, from the baghouse hoppers, is delivered by screw conveyors to feed bins located above and at either end of the furnace. In addition to the feed bins, recycle dust and touch up flux bins are also located in this area. The dust collected in the furnace off-gas settling chamber and in the copper converter cottrell are both recycled to the furnace. The touch up flux is used as required to make minor adjustments to the silica content of the furnace slag but is now also used to raise the silica level of the molten converter slag recycle, which enters the furnace at about 20% SiO_2 and is discharged at about 32% SiO_2. The furnace feed rate is controlled by variable flow control valves associated with weightometers located under these bins. The feed materials are then transported by screw conveyors to drop pipes which discharge directly into the furnace burners. In these burners the falling feed is injected into the furnace by a horizontal flowing stream of oxygen. The oxygen flow is ratioed to the burner feed rate and at Copper Cliff is currently 20-21% weight of the feed.

A cut-away view of the flash furnace is shown in Figure 4. The feed is delivered as just described to four concentrate/oxygen burners, two located at each end of the furnace. These burners are water jacketted. Spontaneous ignition occurs with the oxygen reacting with part of the sulfur and iron in the concentrate to form SO_2 and iron oxides. Silica, contained in the flux and concentrate, combines with these oxides to form slag. Copper, except for the minor amount lost in the slag, and the remaining iron and sulfur collect in the matte. The converter slag is reverted down a launder into the end of the furnace opposite to the slag skimming hole. The off-gases leave through the furnace uptake and pass to the adjacent settling chamber.

The oxygen/concentrate reactions supply all the heat required by the process so no extraneous fuel is required - or in other words, the smelting operation in the INCO furnace is totally autogenous.

Our current experience has given us three years life on the furnace refractory though with recent modifications we are expecting four or possibly five years life. The furnace bottom has never shown any evidence of magnetite build up and is the original bottom installed in 1953 except for the portion added during the 1968 extension. It has been in use for almost thirty years and shows no sign of failure.

The slag is skimmed from one end of the furnace. This operation is only carried out with a full hole to preclude the possibility of the escape of furnace gases. The hole is fitted with a vent hood as is the launder and hood over the slag pot. This launder is water cooled.

Both the slag skimming and the matte tapping holes are of refractory block and are opened and closed manually.

There are two matte holes and two matte launders, with one in operation and the other on stand-by. The matte flows down the launders into ladles located in a tunnel below the matte tapping platform. The matte holes are vented at the furnace as are the launders and tunnel.

A total of 250-350 tonnes per day of converter slag is being reverted to the furnace at present. This represents about 50% of the slag generated. The slag is poured from ladles down a 13 M (43') launder and enters the furnace through an opening in the end of the furnace opposite to that from which the discard furnace slag is skimmed.

This return hole is only opened when slag is about to be returned and is closed immediately after the operation is completed. A door is used to cover the opening and is packed with clay as a seal. There is a venting system over the hole. The furnace, which typically operates at about neutral pressure, is put under slight draft during this slag reversion operation.

GAS CLEANING SYSTEM

The furnace off-gases leaving through the uptake enter a settling chamber which is fitted with cooling fins and air cooling pipes. The furnace gas enters at about 1200°C (2,200°F) and leaves at about 750°C (350°F). In this unit approximately half the dust generated by the furnace is collected, and is removed by screws located in the hoppered bottom for return, through the recycle dust bins, to the furnace.

Prior to delivery to an SO_2 liquefaction plant the flash furnace gases are cleaned in a wet scrubbing system which follows the settling chamber and is shown in Figure 5.

The system consists of a splash tower, three venturi scrubbers and a wet mist precipitator in series. The scrubber water passes to a settling cone with the overflow being recycled through heat exchangers to the scrubbers. The cone underflow, containing the collected solids, passes to aeration for SO_2 removal followed by neutralization with lime and return to the Copper Cliff concentrator for recycling (20). The cleaned furnace off-gas is delivered to a liquefaction plant operated by Canadian Industries Limited (21).

If the furnace gases were to be used for sulfuric acid production the recommended gas cleaning system would consist of a water spray cooler to lower the gas temperature to about 320°C (600°F) followed by an electrostatic precipitator.

FLASH FURNACE OPERATIONAL DATA

The furnace feed rate has been varied from about 700 to 1500 tonnes per day as production requirements dictate. These changes can be easily achieved by shutting off either one, two or even three of the burners. Rates of over 1800 tonnes per day have been attained with the ultimate capacity of the unit still not established because of limitations external to the furnace such as concentrate and oxygen availability, material handling constrictions, converter capacity, anode casting capability, etc. A turn-down of almost 3-1 is achievable very readily and quickly.

At a rate of 1200 tonnes per day the specific smelting rates are 200 tonnes per day/100 m^3 (62 STPD/1000 cu. ft.) of furnace volume or 8.6 tonnes per day/m^2 (0.9 STPD/sq. ft.) of hearth area.

Typically the furnace holds 100-200 tonnes of matte and 100-150 tonnes of slag. The matte is tapped at about 1170°C (2140°F) and the slag is skimmed at about 1230°C (2240°F) while the off-gases also leave at about 1230°C (2240°F).

Because commercial oxygen, at about 97% purity, is used in the furnace, off-gases from the furnace are very low, averaging only 140-170 m^3/tonnes of concentrate (4500-5500 SCFM/ST). As a result of the low volumes being produced by the furnace, the furnace uptake velocities are only 1 m/s (3-4'/sec) so the dusting rate from the furnace is also very low, at only 2-3% of the concentrate rate. The gas handling and cleaning system is also small. In our opinion the small off-gas volume and therefore its small heat content do not justify the installation of a waste heat recovery system.

The refractory utilization during our most recent campaign amounted to 1.1 kg/tonnes feed (2.0 lbs/ST). This includes all the refractory installed in the furnace and its associated equipment at the time of the major rebuild prior to the campaign and the minor amounts installed during the campaign itself.

PROCESS METALLURGY

INTRODUCTION

As already discussed INCO oxygen flash smelting is an autogenous process. The finely divided sulfide-concentrate, carried by the oxygen, ignites and burns as soon as it enters the furnace. The oxygen-concentrate reactions are extremely fast.

FLASH SMELTING FLAME CHARACTERISTICS

Observation of the oxygen-copper concentrate flame at INCO's research stations during a pilot plant flash smelting campaign conducted in a top blown rotary converter shows visible ignition of the concentrate occurs at a very short distance from the tip of the burner. Sampling this flame as a function of distance from the burner tip, to investigate how fast the reactions are occurring, has been carried out.

It was found that at about 1 m from the burner the SO_2 content of the gas was 75%, proving that most of the oxygen had already reacted. At two metres only slag was collected in the sampling steel spoon, indicating that the matte was liquid. The slag sample had the same composition as the slag in the converter, aside from matte prills. This evidence showed that the reactions in the oxygen flame are extremely rapid and occurred in under one-tenth of a second (22).

FLASH SMELTING CHEMICAL REACTIONS

On Table 1 are listed the main chemical reactions which take place in flash smelting. Also shown in this table are the corresponding heats of reaction. The heat generated in the flash smelting flame corresponds mainly to the combustion of the labile sulfur and FeS. A sufficient amount of oxygen per unit weight of concentrate must be supplied to satisfy the heat balance of the operation. If the furnace temperature drops the oxygen to concentrate ratio is increased. Consequently, more iron sulfide is combusted and the furnace thermal equilibrium is re-established, and vice-versa.

It follows, then, that the furnace heat balance determines the matte grade to be obtained from a given concentrate. However, as will be discussed later, there are methods available to permit operation of the process to obtain a matte product with a desired composition.

FLASH FURNACE HEAT BALANCE

Oxygen flash smelting of INCO's chalcopyrite copper concentrate at Copper Cliff at the normal throughput of about 1300-1400 tonnes per day (1400-1500 short tons per day) requires 20-21 weight percent tonnage oxygen. About 430 000 KCal/tonne of concentrate (1.55 x 10^6 Btu/ST) are released by the oxygen-concentrate reactions. About 60% of this heat satisfies the heat requirements of the matte and the slag. The heat content of the off-gas, represents only 20% of the heat generated, because of its small volume. Typical heat losses of the Copper Cliff flash furnace are about 72 000 KCal/tonne of concentrate (0.26 x 10^6 Btu/ST) Because most of the heat generated is used in heating up the products, the flash flame temperature is only slightly above 1450°C (2,640°F). The foregoing data is given in Table 2.

FLASH FURNACE MASS BALANCE

The matte grades produced at Copper Cliff vary between 40% and 50% Cu, depending on furnace throughputs and the amount of secondaries reverted to the furnace. Typical mass balance data for the operation of INCO's flash furnace are given in Table 3. The information was derived from average shift data for three months of steady state operation, just prior to the converter slag recycle experimental campaign. Within this period the furnace throughput was 1360 tonnes per day of concentrate. The dust recovered from the furnace settling chamber, about 10-12 tonnes per day, is an internal recirculation and it is not shown in this table. The oxygen requirement for smelting the Copper Cliff concentrate, at these throughputs, is 20.6% by weight of the concentrate to yield a 40.5% Cu matte. The copper content of the slag, 0.57%, is typical of this operation. The data collected during many years of operation show that the slag losses in the flash furnace are directly proportional to matte grades, the best correlation being

$$\frac{\% \, Cu_m}{\% \, Cu_s} = 70 + 10$$

This copper partition coefficient is valid for matte grades up to about 55% Cu.

COMPOSITION OF FURNACE OFF-GAS

The use of commercial oxygen in flash smelting results in a small volume of off-gas with a high SO_2 content, as shown in Table 4. Oxygen efficiency in the flash furnace is 100%. Because there is some air in-leakage into the furnace, the composition of the gas in the furnace uptake is typically 83% SO_2. After cooling and cleaning, the gas analyzes 75% to 80% SO_2, because of the additional air dilution which normally occurs during gas treatment.

The high strength gas is used at Copper Cliff for producing liquid SO_2. In most cases, this gas will provide an excellent continuous base load for a sulfuric acid plant as can be noted from the following data.

The sulfur distribution in a hypothetical INCO flash smelting operation is shown in Figure 6. The smelter feed is a chalcopyrite concentrate analyzing 28% Cu, 29% Fe and 32.6% S which yields a 50% Cu matte at a throughput of 1100 tonnes per day.

In the flash smelting operation, 57% of the sulfur is eliminated from the concentrate as a strong, low volume gas stream. Except for the minor amount in the slag, the remainder of the sulfur reports to the matte and is eliminated during conversion. The best designed converter hoods cannot avoid a gas dilution of at least 100%. Further dilution in the gas treatment line makes it difficult to attain an average SO_2 content in these converter gases exceeding about 4.5%. However, the gas resulting from blending the flash furnace gas with the converter gas increases the strength of the feed to the acid plant to almost 10%.

Today a number of conventional reverb smelters are processing their converter gases in acid plants. The same acid plants, with only minor modifications, could achieve fixation of almost all of the sulfur input to these smelters if the reverb furnaces were replaced with oxygen flash furnaces. In fact the combined flash furnace-converter gas, at about twice the SO_2 strength of the converter gas alone, would have a smaller volume than that of the converter gases from reverb smelting, because of the higher matte grade which the flash furnace would yield. In addition, the blended gas would provide a much steadier feed to the acid plant.

MOLTEN CONVERTER SLAG RECYCLE

As previously explained, the presence of Ni in the Cu concentrate and the Cu in the Ni concentrate results in substantial intercircuit transfer streams at the Copper Cliff smelter. Copper converter slags are an important bleed for nickel from the copper circuit. Until very recently, 100% of these slags were reverted to the nickel reverb furnaces.

However, this practice greatly complicated the traffic along the converter aisle. It was thought, then, that at least partial reversion of the copper converter slags to the flash furnace could alleviate this problem. At the same time, the question of the capability of the INCO flash smelting furnace for cleaning converter slags had been raised. An experimental campaign was then conducted at Copper Cliff early in 1978 to answer these questions.

For this purpose one of the furnace burners was removed from the wall closest to the converter aisle to provide a converter slag recycle port. The slag was poured into the furnace via a 13 m (43') long launder. During the experimental campaign, which lasted for almost two months, the furnace operated on only three burners at an average throughput of 1300 tonnes per day with a maximum of 1630 tonnes per day.

The mass balance data shown in Table 5 reflects a seven-day operating period during which the average converter slag recycle to the furnace was 214 tonnes per day, corresponding to about 43% of the total converter slag generated. Feed rate during this period averaged 1345 tonnes per day of concentrate.

Production of flash furnace slag was about 600 tonnes per day, which is double that produced in the normal operation. In spite of this, the copper partition was similar to that usually observed without slag recycle. It should be noted that the same is true for the nickel and the cobalt partitions, a clear indication that the showering of matte through most of the slag layer in the INCO flash furnace provides excellent conditions for matte/slag chemical interaction.

Converter slag recycle results in an increase in the matte grade, 45% versus 40.5% for the normal operation at the same throughput. This is mainly a consequence of the extra heat required to melt the additional flux fed to the furnace to compensate for the low SiO_2 in the converter slag.

The possibility of bottom build-up when returning converter slag to the flash furnace was a particular concern during the campaign. Periodic magnetite and chromia balances indicated that no bottom build-up was taking place. In fact, these balances showed that substantial reduction of Fe_3O_4 was taking place in the furnace.

In addition, the flash furnace was drained before and after the experimental campaign. There was no visual evidence of any bottom build-up.

The conclusion was that the INCO flash furnace could handle highly oxidized slags over long periods without any problems. As a result, as mentioned earlier, converter slag recycle to the flash furnace is now a normal operating practice at Copper Cliff.

CONTROL OF MATTE GRADE

The possibility of controlling the oxygen flash smelting operation so as to obtain a matte with a desired grade from any concentrate will now be discussed. This may lead to a matte grade either lower or higher than that which would be obtained from a normal autogenous operation.

Small additions of coal to the feed, less than 1%, result in substantial decreases in matte grade of from 10 to 15 percentage points depending on the composition of the feed. This possibility was established when Thompson copper concentrates, which contain several percentage points graphite, were flash smelted as part of the Copper Cliff furnace charge. The graphite was totally combusted in the furnace. We have simulated this type of operation, at INCO's pilot plants and in a mini-plant flash smelting installation which we have at INCO's research laboratory, with excellent and predictable results.

Increasing the matte grade can be achieved by the addition of revert materials, such as dust, ground matte and slag skulls, etc., to the feed or injection of water into the furnace. All these alternatives consist of introducing a coolant into the system to consume the heat which would be generated above that corresponding to a normal autogenous operation. Because of this, they have the limitation of not being energy efficient. A second alternative consists of roasting part of the concentrate and flash smelting a blend of the resulting calcine with the remainder of the green concentrate. The desired composition of the matte to be obtained can be controlled by adjusting the ratio of calcine to green concentrate in the feed. What this technique actually does is to adjust the fuel value of the feed to that required to yield the desired matte grade on autogenous oxygen flash smelting.

Testwork on autogenous oxygen flash smelting of various blends of roasted and unroasted copper sulfide concentrate with the compositions shown in Table 6 was conducted in a mini-plant furnace at INCO's J.R.G. Research Laboratory. The Cu and the Fe in the calcine were mainly as copper ferrite, $CuFe_2O_4$. Minor amounts of CuO and Fe_3O_4 were also present. The amount of oxygen required to simulate a commercial autogenous operation was calculated from heat and mass balances which predicted the commercial matte grades which would be obtained at the various experimental calcine/green concentrate ratios. The actual matte grades obtained, as shown in Table 6, were very close to those expected. The slag compositions are also shown.

In these tests it was observed that ignition and combustion of the green concentrate was not affected by the dilution with calcine. The slags were fluid and good separation of mattes and slags was observed.

The matte/slag Cu partition for the 50% Cu matte was similar to that observed at Copper Cliff. At higher matte grades the Cu partition decreased. However, more testwork is required before developing data which are statistically meaningful.

In summary this development offers some interesting advantages:

(a) With respect to conventional autogenous oxygen flash smelting.

- The possibility of obtaining a product with a desired composition regardless of the composition of the concentrate and furnace throughput. There is potential here for producing white metal or even blister copper in a one step smelting operation.

(b) With respect to alternative methods for controlling the matte grade by adding coolant.

- Less oxygen consumption to obtain the same matte from a given concentrate.
- A higher specific capacity of the furnace.
- Less dusting.

This mode of operation makes INCO oxygen flash smelting a very flexible process, in particular for processing low grade concentrates, normally high in pyrite, which on conventional oxygen flash smelting would yield low grade mattes.

PROCESS ECONOMICS

INTRODUCTION

The process economics of INCO oxygen flash smelting are based on the treatment of a typical B.C. concentrate and include a review of a proposed operation with its mass balance, direct process energy requirements and its estimated operating and capital costs.

PROCESS MASS BALANCE

The simplified mass balance for the treatment of 1360 tonnes per day (1500 STPD) of a typical B.C. concentrate is shown on Table 7. It has been assumed that full molten converter slag, furnace dust and converter dust recycle would be practised.

The assumed concentrate analysis is 30% Cu, 24% Fe, and 30% S. There is sufficient SiO_2 in this concentrate to make it self fluxing. The oxygen requirement amounts to about 20% of the concentrate weight. Autogenous operation will yield a matte containing about 55% Cu, and a discard slag containing less than 0.8% Cu. The furnace offgases will contain about 80% SO_2 and amount tc about 7700 Nm^3/hr (5100 SCFM). The copper recovery of the process is 98.6%.

DIRECT PROCESS ENERGY REQUIREMENTS

On the basis of the foregoing mass balance, the direct energy requirements for the flash smelting and converting operations have been estimated. The approach taken is conservative and it is believed that a detailed analysis would decrease rather than increase the figures shown in Table 8.

In the drying and smelting area the only fuel requirement is for concentrate drying. A moisture of 12% was assumed for the concentrate and it will therefore require about 470 000 Btu/ST concentrate. The process electrical requirements can be broken down into three major areas. Power for oxygen production, based on 360 Kwh/ST at 20.2% by weight of the concentrate, amounts to 74 Kwh/ST concentrate. Other process power, based on our Copper Cliff experience, amounts to about 23 Kwh/ST concentrate, while furnace gas cleaning is about 3 Kwh/ST concentrate, for a total of 100 Kwh/ST concentrate. Using the generally accepted factor of 10 500 Btu/Kwh, which is for thermally generated power, this becomes 1.05 million Btu/ST concentrate. The total drying and smelting energy requirement for the B.C. concentrate is thus the 1.52 million Btu/ST.

In the converting area, assuming the feed is the 55% Cu furnace matte, the estimated direct process energy requirements are also shown in Table 8. The holding fire requirements are based on 212 000 Btu/ST of matte treated while the electrical power for blast air and gas cleaning are based on 1.4 Kwh/1000 SCF and 0.43 Kwh/1000 SCF respectively. These factors are those used by H.H. Kellogg and J.M. Henderson in their paper entitled "Energy Use in Sulfide Smelting of Copper". An allowance has been added for other items such as material handling, crane transfers, converter rotation, etc. This yields a total converting energy requirement of 620 000 Btu/ST concentrate.

The total drying, smelting and converting direct process energy requirements are therefore estimated at 2.14 million Btu/ST concentrate compared with figures in the 4-5 million range for a standard reverb/ converter operation. Of this 2.14 million figure approximately 72% is for power and 28% is for conventional fuel.

As noted above this total is based on the 10 500 Btu/Kwh factor which is for thermally generated power (23, 24). In British Columbia where almost all power is hydro based a more realistic factor might be 3 400 Btu/Kwh. Using this figure the total process energy requirements for drying, smelting and converting using the INCO Oxygen Flash Smelting Process are only 1.1 million Btu/ST.

OPERATING COST ESTIMATE

The total energy requirements are, of course, an important factor in process selection, however, to the producer the major concern is the total cost of the process energy. Table 9 shows an approximate direct operating cost for the INCO oxygen flash furnace treating 1360 tonnes per day (1500 short ton per day) or 449 000 tonnes per year (495 000 short ton per year) of B.C. concentrate.

The costs shown are for the drying and smelting portion of the process only and exclude all indirect and overhead costs which will vary substantially depending on location and philosophy.

Only 8 men per shift are required to operate this area which at a cost of $25,000 per man, totals about $900,000/yr. The front line supervision will be $180,000/yr. The natural gas for drying the concentrate, at an assumed price of $2.50/1000 SCF, will cost $460,000/yr. Direct electrical power, at 2.0 cts/Kwh, totals $270,000, while the tonnage oxygen, estimated at $15/ST, totals $1,530,000/yr. Both the natural gas and power costs used in this estimate are, it is believed, conservative for a B.C. location.

Repair labour, supervision and supplies, including refractory for major rebuilds every three years, have been calculated at $1,100,000/yr An allowance for miscellaneous operating supplies of $150,000/yr. and a contingency of 10% have been included to arrive at a total of $5.05 million/yr.

This amounts to $11.25/tonne ($10.20/ST) of concentrate or 1.7 cts./lb. Cu in matte. Total copper production would be about 290 million lb/yr (132 000 tonnes per year). The total energy charges amount to less than 33% of this total or $3.66/tonnes ($3.32/ST) or less than 0.6 cts./lb. Cu.

CAPITAL COST ESTIMATE

A pre-feasibility type capital cost, which has a process scope from receipt of the wet feed at the storage bins through to matte and slag in the furnace launders and the off-gas cleaned ready for delivery to an acid plant, has been estimated. It is based on a greenfield site in North America but does not include any items outside the immediate drying and smelting area, except for a turn key oxygen plant (270 tonne or 300 STPD), which is included. No working capital, escalation through construction, insurance and taxes, finance charges and the like are included.

The figures are as shown in Table 10, and total approximately $35 million, of which about $15 million is for the oxygen plant and $20 million for the process area. It should be noted that no capital expenditures are required for slag cleaning or waste heat recovery equipment while attaining over 98.5% copper recovery with a totally autogenous smelting operation.

SUMMARY

MAJOR ADVANTAGES OF INCO OXYGEN FLASH FURNACE

The major advantages of the INCO oxygen flash furnace facility can be summarized as follows:

(1) The process facility requires a low capital cost investment because of its simple construction and relatively small size. The furnace structure is entirely independent of the building. The production rate per unit volume is from 3 to 10 times that for such conventional units as reverbs and electric furnaces. Capital expenditure is not justified for waste heat recovery because the heat that can be recuperated is very small.

(2) The gas cleaning equipment capital costs are low because the off-gas volume from the furnace is only 130 Nm^3/m (or 5000 SCFM) for a 1360 tonnes per day (1500 STPD) operation. This is only one-fifth or less of that for other conventional processes. Because of the low volume, the gas velocities in the furnace are low and, as a result, the furnace dusting rates are low, typically 2-3% of feed at our Copper Cliff operation. Therefore, the capital expenditure for dust handling is also low.

(3) The capital cost for sulfur fixation is also low. After gas cleaning the off-gas strengths are typically 70-80% SO_2. This gas is therefore ideally suited for liquefaction, a cheap fixation process. Alternatively, it can be used directly as a continuous base load for an acid plant with the minimum of air dilution or ideally as a "sweetener" for other smelter gases, such as converter gas. The use of this low volume, high strength, furnace gas permits the acid plant to be designed to a small overall inlet volume thus making its capital costs minimal.

(4) The furnace operating costs are low. There is only one major operating unit. The drying and smelting operations can be satisfactorily operated with only 7 to 8 operators/shift. Conventional fuel is required for drying only. The smelting operation requires no fuel. The costs for oxygen will be less than those for conventional fuels.

(5) Over almost thirty years of operation a variety of concentrates, both nickel and copper, have been successfully smelted. The grades of these feeds have varied from 10% to

33% Cu plus nickel. On occasion, a portion of the furnace feed has been our matte separation copper sulfide concentrate running 72% Cu.

(6) With an autogenous operation such as the INCO Process, with a fixed furnace size, a fixed feed rate and a specific concentrate grade one will obtain a specific matte grade. However, the addition of coal in very small amounts will decrease this matte grade, while smelting of blends of roasted and unroasted concentrate, or the injection of water will increase this matte grade.

(7) Over a wide range of matte grades (35-55% Cu) the INCO furnace produces a discard slag at a partition coefficient of 70 which typically represents a recovery of 99-99.5% copper. Even during converter slag recycle operation this high partition coefficient can be duplicated, though, of course, copper recoveries fall to the 98-99% range, because of the increased slag quantities discarded. The demonstrated high partitions eliminate the need for separate slag cleaning facilities. Up to 50% of the iron in the concentrate leaves with the furnace slag if no converter slag recycle is practised; however, this increases to 100% with full converter slag recycle.

(8) The furnace operation is very simple. The major control is the oxygen/concentrate ratio. INCO monitors the slag and matte temperatures and makes minor adjustments to this ratio as necessary. Should the slag temperature fall the oxygen/concentrate ratio is increased slightly and vice-versa. The operation is so simple that no computer control is necessary. Furnace feed rate turn downs of about three to one can be readily and rapidly achieved by shutting down one, two or even three burners.

(9) The furnace has been extremely reliable over its many years of operation. The Copper Cliff units have handled in excess of 8 million tonnes of copper concentrate since 1953. Refractory life at the sidewalls is now three years between major rebuilds but it is expected that this figure will be

exceeded during the current campaign. No magnesite build up on the hearth has been experienced even during the current converter slag recycle operation. This slag reversion has been a standard operating practise at Copper Cliff for almost a year and a half. The original 1953 hearth is still in use.

REFERENCES

(1) "Smelting: I. Copper Cliff Smelter", Canadian Mining Journal, pp 431-445, May, 1946.

(2) "Metallurgical Improvements in the Treatment of Copper-Nickel Ores", The Staff, International Nickel Company of Canada (Presented by P. Queneau), CIM Transactions, Vol.51, pp 187-198, 1948.

(3) "Extractive Metallurgy at International Nickel - A Half Centre of Progress", L.S. Renzoni, Canadian Journal of Chemical Engineering, February, 1969.

(4) "The Sudbury Operations of INCO Metals Co.", W. Schabas, Canadian Mining Journal, May, 1977.

(5) "Treatment of Nickel-Copper Matte", K. Sproule, G.A. Harcour and L. S. Renzoni, Journal of Metals, Vol. 12, No.3, pp 214-219, March, 1960.

(6) "Converter Operating Practice at the Copper Cliff Smelter of the INCO Metals Company", R.J. Neal, R.A. Reyburn, INCO Metals Company presented at 108th AIME Annual Meeting, New Orleans, Louisiana, February, 1979.

(7) "Flash Smelting of INCO Ores Creates New Industry", Northern Mining, July 8, 1948.

(8) "Autogenous Smelting of Sulfides", J.R. Gordon, G.H.C. Norman, P.E. Queneau, W.K. Sproule and C.E. Young, U.S. Patent 2,668,107 filed June 25, 1949, issued February 2, 1954.

(9) "The Oxygen Flash Smelting Process of the International Nickel Company", The Staff, CIM Transactions, Vol.58, pp 158-166, 1955.

(10) "Oxygen Flash Smelting Swings Into Commercial Operation", The Staff, International Nickel Company of Canada, Journal of Metals, Vol.7, pp 742-750, June, 1955.

(11) "Recent Developments in the INCO Oxygen Flash Smelting Process", S. Merla, C.E. Young and J.W. Matousek, 101st AIME Annual Meeting, San Francisco, 1972.

(12) "Copper Smelting by the International Nickel Company of Canada", The Staff, Copper Cliff Smelter, International Symposium on Copper Extraction and Refining, Las Vegas, 1976, in Extractive Metallurgy of Copper, Vol. 1, ed. by J.C. Yannopoulos and J.C. Agarwal, TMS-AIME, pp 218-233, 1976.

(13) "Flash Smelting", J.W. Matousek, Seminar on Extractive Metallurgy of Copper, McGill University, Montreal, Canada, January, 1977.

(14) "The INCO Oxygen Flash Smelting Process", C. Diaz, INCO Metals Company, Process Development, Seminar on Copper Metallurgy, Bogota, Colombia, February, 1978.

(15) "Tonnage Oxygen for Nickel and Copper Smelting at Copper Cliff", R. Saddington, W. Curlook and P. Queneau, Journal of Metals, April, 1966, Vol.18, No.4, pp 440-452.

(16) "Operation of the INCO Flash Smelting Furnace with Recycle of Converter Slags", T.N. Antonioni, A.D. Church, C. Landolt and F. Partelpoeg, 18th Annual Conference of Metallurgists, CIM Sudbury, August, 1979.

(17) "INCO's Oxygen Flash Smelting Process for Copper and Nickel Concentrates", C.M. Diaz, H.C. Garven, M.Y. Solar, INCO Metals Company, Process Research, November, 1978.

(18) "INCO's Oxygen Flash Smelting Process for Copper and Nickel Concentrates; Off-Gas Handling and Impurity Distributions",

M.Y. Solar, A.D. Church, T.N. Antonioni, INCO Metals Company, EPA Symposium on Control of Particulate Emissions in the Primary Non-Ferrous Metals Industries, Monterey, California, March, 1979.

(19) "Smelting Nickel Concentrates in INCO's Oxygen Flash Furnace", M.Y. Solar, R.J. Neal, T.N. Antonioni, M.C. Bell, INCO Metals Company, Journal of Metals, Vol. 31, No. 1, pp 26-32, January, 1979.

(20) "Handling Acid Slurries at the Flash Smelting Plant of the International Nickel Co. of Canada", J.N. Lilley, R.W. Chambers C.E. Young, CIM Annual Meeting, Toronto, April, 1960.

(21) "Sulfuric Acid and Liquid Sulfur Dioxide Manufactured from Smelter Gases at Copper Cliff, Ontario", Trans CIM Vol. LV, 1952, pp 123-125.

(22) "Oxygen Flash Smelting in a Converter", The Staff, INCO Metals Company, Process Research, presented at 106th Annual Meeting, Atlanta, Georgia, March, 1977.

(23) "Energy Use in Sulfide Smelting of Copper", H.H. Kellogg and J.M. Henderson, 105th AIME Meeting, February, 1976; in Extractive Metallurgy of Copper, J.C. Yannopoulos and J.C. Agarwal, editors, AIME - Metallurgist's Society, 1976, Vol. 1, pp 373-415.

(24) "Pollution Control and Energy Consumption at U.S Copper Smelters", D.A. Schultz, Journal of Metals, January, 1978, with further discussion by H.H. Kellogg and D.A. Schultz, May, 1978.

TABLE 1

INCO OXYGEN FLASH SMELTING CHEMICAL REACTIONS

	H_25°C Kcal/Mol
$CuFeS_2 + 1/2O_2 = 1/2Cu_2S + FeS + 1/2SO_2$ (Chalcopyrite)	-23.48
$Cu_5FeS_4 + 1/2O_2 = 5/2Cu_2S + FeS + 1/2SO_2$ (Bornite)	-16.88
$CuS + 1/2O_2 = 1/2Cu_2S + 1/2SO_2$ (Covellite)	-32.57
$FeS_2 + O_2 = FeS + SO_2$ (Pyrite)	-54.07
$FeS + 3/2O_2 = FeO + SO_2$	-112.07
$Cu_2S + 3/2O_2 = Cu_2O + SO_2$	-92.34
$FeO + 1/6O_2 = 1/3Fe_3O_4$	-24.28
$FeO + 1/2SiO_2 = 1/2Fe_2SiO_4$	- 4.33

TABLE 2

INCO OXYGEN FLASH SMELTING
HEAT BALANCE

THROUGHPUT = 1 300-1 400 tonnes per day of concentrate at 30% Cu, 31% Fe, and 33% S

OXYGEN REQUIRED = 20-21 wt % of concentrate

		Kcal/MT CONCENTRATE	BTU/ST CONCENTRATE	DISTRIBUTION %
HEAT GENERATED				
By O_2/Feed Reactions		430 000	1 548 000	100
HEAT REQUIREMENTS				
Heat In:	Matte	182 000	654 000	42
	Slag	77 000	279 000	18
	Off-Gas	88 000	318 000	20
	Dust	11 000	39 000	3
Furnace Heat Losses		72 000	258 000	17
TOTAL		430 000	1 548 000	100

TABLE 3

INCO OXYGEN FLASH SMELTING
MASS BALANCE FOR OPERATIONS WITHOUT CONVERTER SLAG RECYCLE
(AUGUST TO OCTOBER, 1977)

INPUTS	RATE DMTPD	COMPOSITION - WT . %			
		Cu	Fe	S	SiO_2
Concentrate	1 360	30	31.0	33.5	2.0
Recycle Dust	32	42	8.0	19.0	1.7
Flux	95	-	2.2	-	78.0
Tonnage	280 (20.6%)	-	-	-	-
OUTPUTS					
Matte	1.030	40.5	28.8	24.5	-
Slag	295	0.57	43.0	1.3	33.0
Sludge	20	27.00	16.0	18.0	2.6
Off-Gas	410	-	-	-	-

PARTITION COEFFICIENT: $\frac{\%\ \text{Cu IN MATTE}}{\%\ \text{Cu IN SLAG}} = 70$

TABLE 4

FLASH FURNACE OFF-GAS COMPOSITION

(IN VOLUME PERCENT)

	SO_2	A	N_2	CO_2	O_2
AT FURNACE UPTAKE					
Without Dilution	94.5	2.8	1.0	1.7	0
With Dilution	83.0	2.5	13.0	1.5	0.02
AFTER GAS CLEANING	77	2	20	1	1

TABLE 5

INCO OXYGEN FLASH SMELTING

MASS BALANCE FOR OPERATION WITH 43% CONVERTER SLAG RECYCLE

(JANUARY 1978)

INPUTS	RATE DMTPD	COMPOSITION - WT.% Cu	Fe	S	SiO_2
Concentrate	1,345	30.1	30.1	33.3	2.0
Recycle Dust	32	41.8	7.9	19.7	1.7
Converter Slag	214	6.9	47.9	0.9	18.9
Flux	156	-	2.2	-	77.8
Tonnage Oxygen	286 (21.3%)	-	-	-	-
OUTPUTS					
Matte	940	45.0	25.2	24.1	0.3
Slag	598	0.68	43.5	1.4	31.1
Sludge	22	24.0	16.8	18.0	2.2
Off-Gas	434	-	-	-	-

$$\text{PARTITION COEFFICIENT} \quad \frac{\%\ \text{Cu IN MATTE}}{\%\ \text{Cu IN SLAG}} = 68$$

TABLE 6

INCO OXYGEN FLASH SMELTING OF BLENDS OF ROASTED AND GREEN COPPER CONCENTRATE

FEED COMPOSITIONS (WT.%)

	Cu	Ni	Fe	S
Concentrate	29.7	1.0	30.7	35.2
Calcine	35.0	1.2	37.8	0.8

EXPERIMENTAL RESULTS

% Calcine Added By Weight of Green Concentrate	Matte Grade, % (Cu+Ni)*		Slag Composition (%)		
	Expected**	Obtained	SiO_2	Fe	Cu
0	40.0	42.4	30.7	37.0	0.70
5.5	48.5	50.8	30.6	40.8	0.67
11.1	58.5	58.6	35.0	34.7	1.19
22.5	77.0	75.0	33.5	34.9	4.77

* % Ni in mattes: 1.5

** Predicted from heat and mass balance calculations for an autogenous commercial operation.

TABLE 7

INCO OXYGEN FLASH FURNACE

MASS BALANCE FOR OPERATION WITH

B.C. CONCENTRATE AND 100% MOLTEN CONVERTER SLAG RECYCLE

INPUTS	RATE DRY TONNES/DAY	COMPOSITION - WT. %			
		Cu	Fe	S	SiO_2
Concentrate	1360	30.0	24.0	30.0	12.0
Recycle Dust	68	34.2	23.1	10.8	12.4
Converter Slag	322	5.0	47.3	1.5	26.4
Oxygen (95% Purity)	274	-	-	-	-
OUTPUTS					
Matte	765	55.20	20.0	23.7	-
Slag	757	0.79	43.2	1.5	32.9
Dust	56	34.50	27.4	8.5	13.7
Off-Gas (81% SO_2 by Vol.)	493	-	-	45.3	-

TABLE 8

INCO OXYGEN FLASH FURNACE
DIRECT ENERGY REQUIREMENTS
FOR B.C. CONCENTRATE

DRYING & SMELTING	KWH/ST CONC.	BTU/ST CONC.	KCAL/MT CONC.
Fuel: Drying (12% H_2O)	-	470,000	131,000
Electrical Power:			
a) Oxygen (360 KWH/ST @ 20.2% of Conc.)	74	780,000	216,000
b) Material Handling, Drying & Smelting	23	240,000	67,000
c) Gas Cleaning (25% Dilution)	3	30,000	8,000
Electrical Power Sub-Total	100	1,050,000	291,000
Total Drying & Smelting	100	1,520,000	422,000
CONVERTING			
Fuel: Holding Fire	-	120,000	33,000
Electrical Power:			
(a) Blast Air (85% O_2 Efficiency)	30	310,000	86,000
(b) Gas Cleaning (150% Dilution)	13	135,000	38,000
(c) Other (Allowance)	5	55,000	15,000
Electrical power Sub-Total	48	500,000	139,000
Total Converting	48	620,000	172,000
GRAND TOTAL Electrical Power	148	1,550,000	430,000
Fuel	-	590,000	164,000
TOTAL	148	2,140,000	594,000

TABLE 9

INCO OXYGEN FLASH FURNACE

APPROXIMATE DIRECT OPERATING COST ESTIMATE

1360 TONNES PER DAY OF CONCENTRATE

(BASIS 1980 COSTS)

	$1000/YR.
Operating Labour (8 men/shift)	900
Operating Supervision (1 man/shift)	180
Natural Gas ($2.50/1000 C.F.)	460
Power (2.0 cts/KWH)	270
Oxygen ($15/ton)	1,530
Repair Labour Supervision & Supplies	1,100
Miscellaneous Operating Supplies	150
Contingency (10%)	460
TOTAL	$5,050
OR	$11.25/Tonne Conc. 1.7 cts/lb Cu

NOTES:

(1) 449 000 tonnes per year of concentrate treated.

(2) approximately 290 x 10^6 lbs copper/year as blister would be produced.

(3) Includes costs of major rebuild every 3 years.

TABLE 10

INCO OXYGEN FLASH FURNACE

APPROXIMATE CAPITAL COST ESTIMATE

1360 TONNES PER DAY OF CONCENTRATE

(BASIS 1980 COSTS)

	$ x 1000
A. Process Plant	
1. Building	3,900
2. Equipment	11,000
B. Oxygen Plant (Turn Key)	13,500
C. Engineering, Procurement and Project Services	2,800
D. Contingency (20%)	3,600
TOTAL PHYSICAL FACILITIES	$34,800

ASSUMES:

1. Greenfield site.
2. North American location.
3. Battery limits facility - all services available at the building walls.
4. No waste heat generation facilities.
5. No separate service, maintenance or office facilities.
6. No mobile equipment.
7. No land acquisition costs, licenses, permits, fees.
8. Equity - financed. No interest or finance charges included.
9. No working capital.
10. No owner's management or preproduction operating costs.
11. No project insurance and taxes.
12. No escalation through construction.

FIGURE I

COPPER CLIFF SMELTER FLOWSHEET

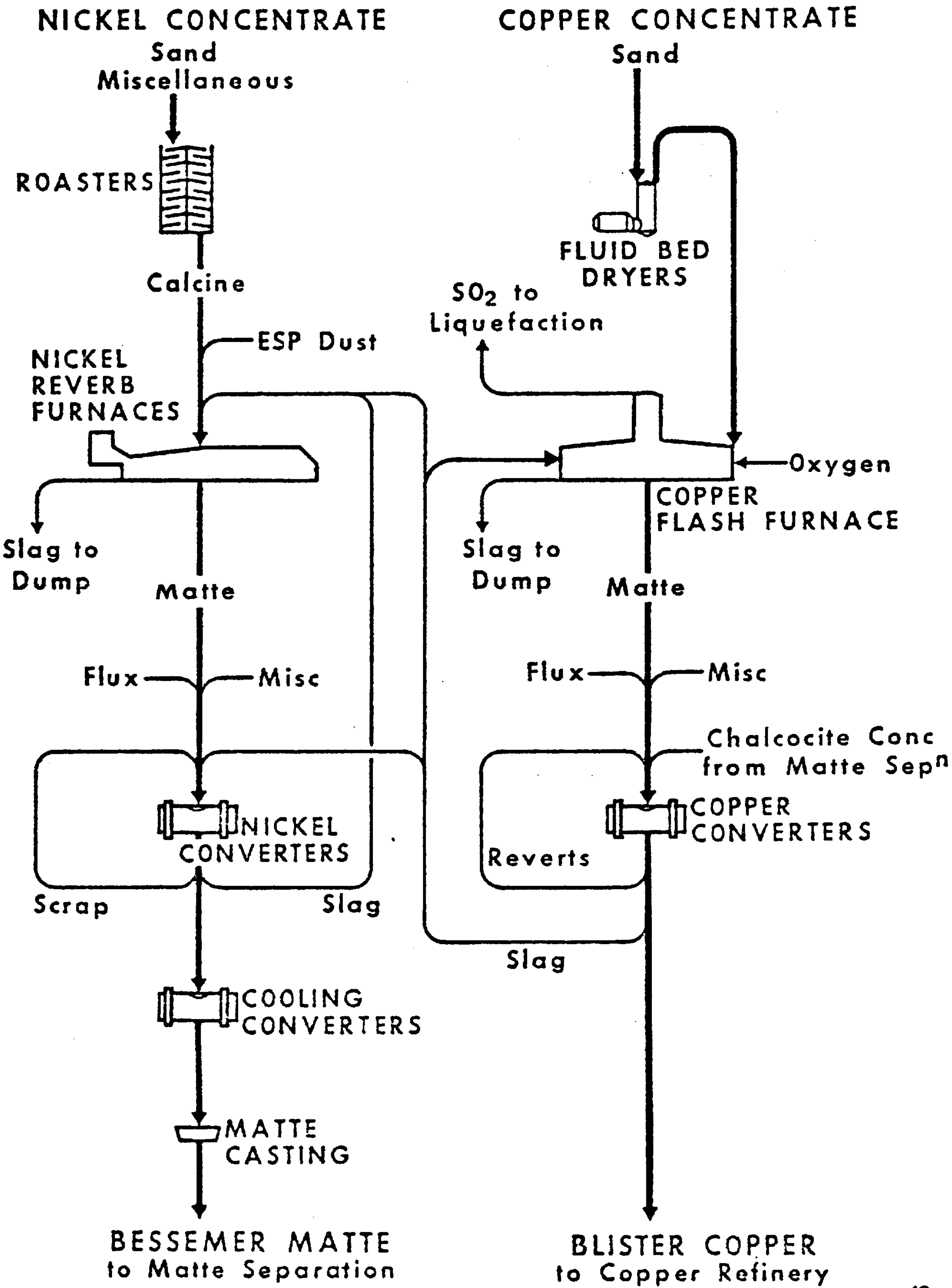


FIGURE 2

STAGES OF FLASH FURNACE DEVELOPMENT

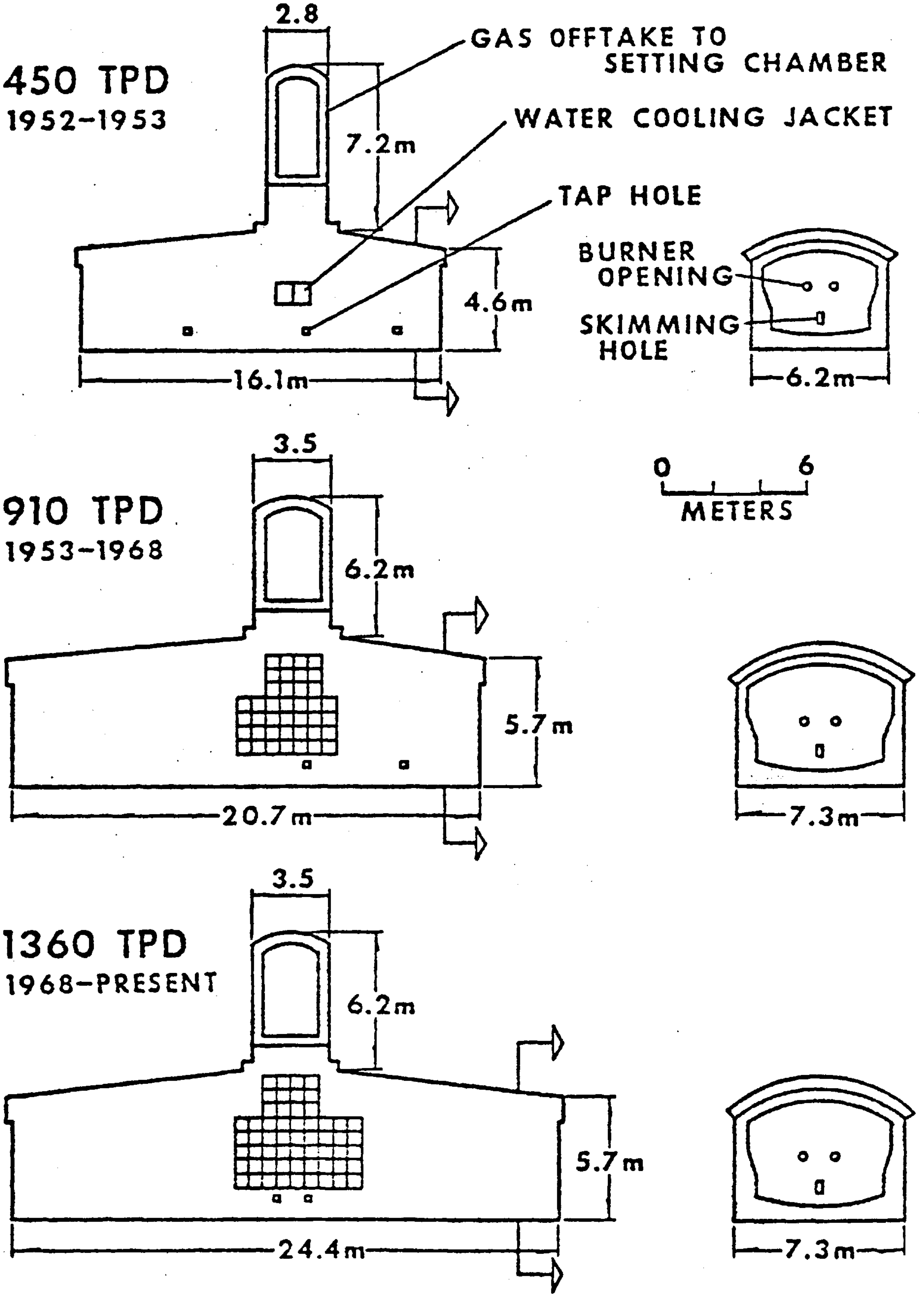


FIGURE 3
FLASH SMELTING FLOWSHEET

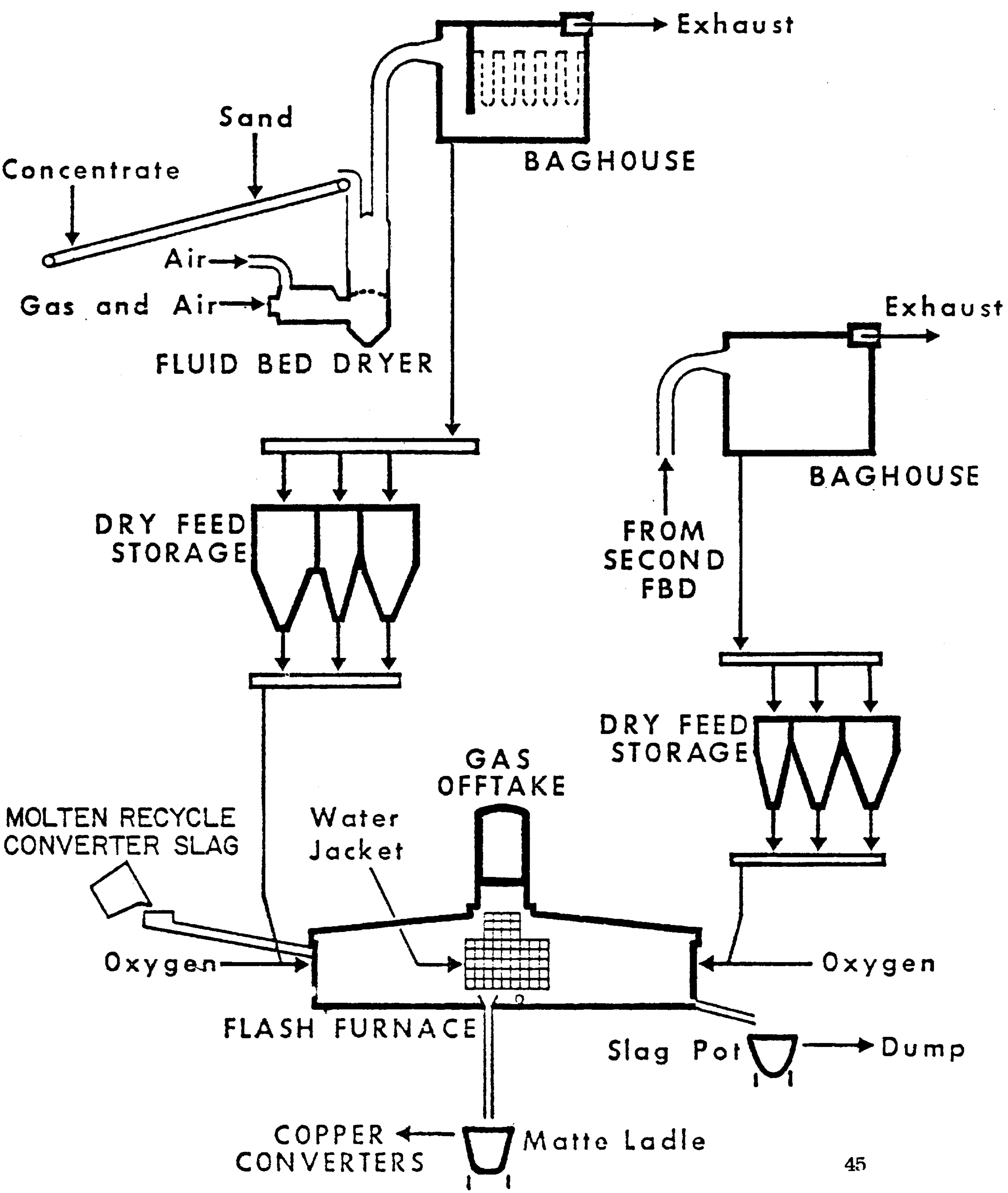


FIGURE 4
INCO FLASH FURNACE

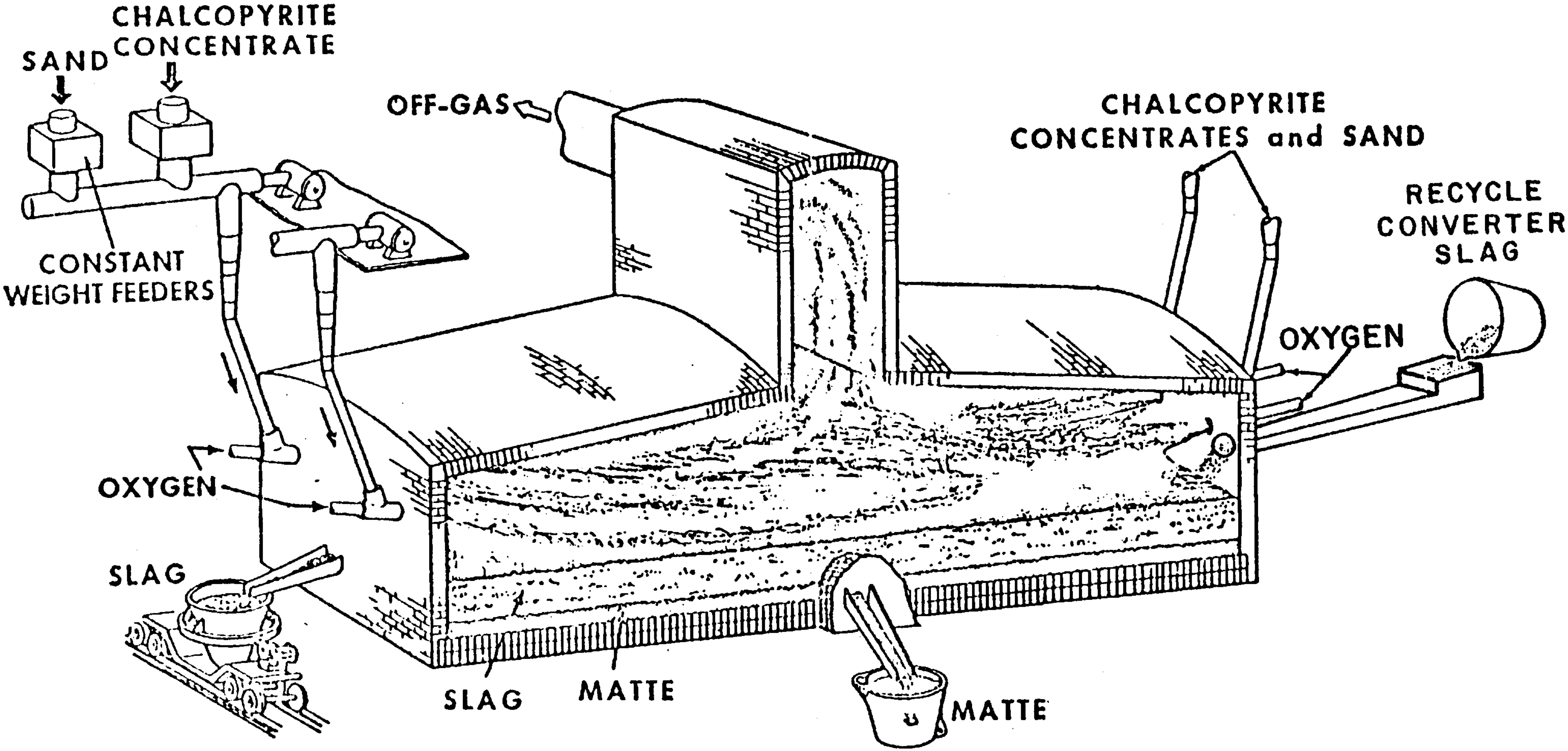


FIGURE 5
FLASH FURNACE GAS CLEANING SYSTEM

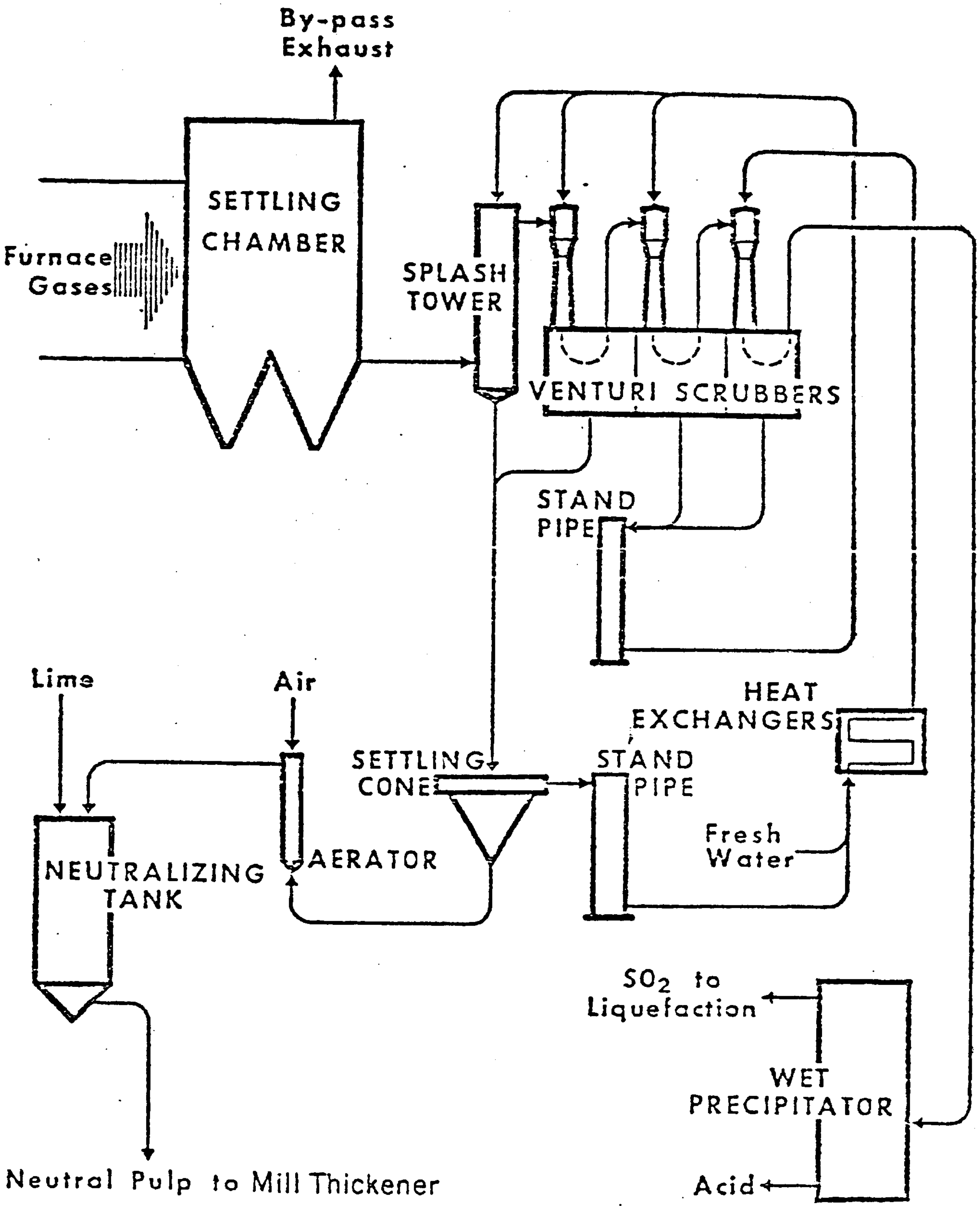
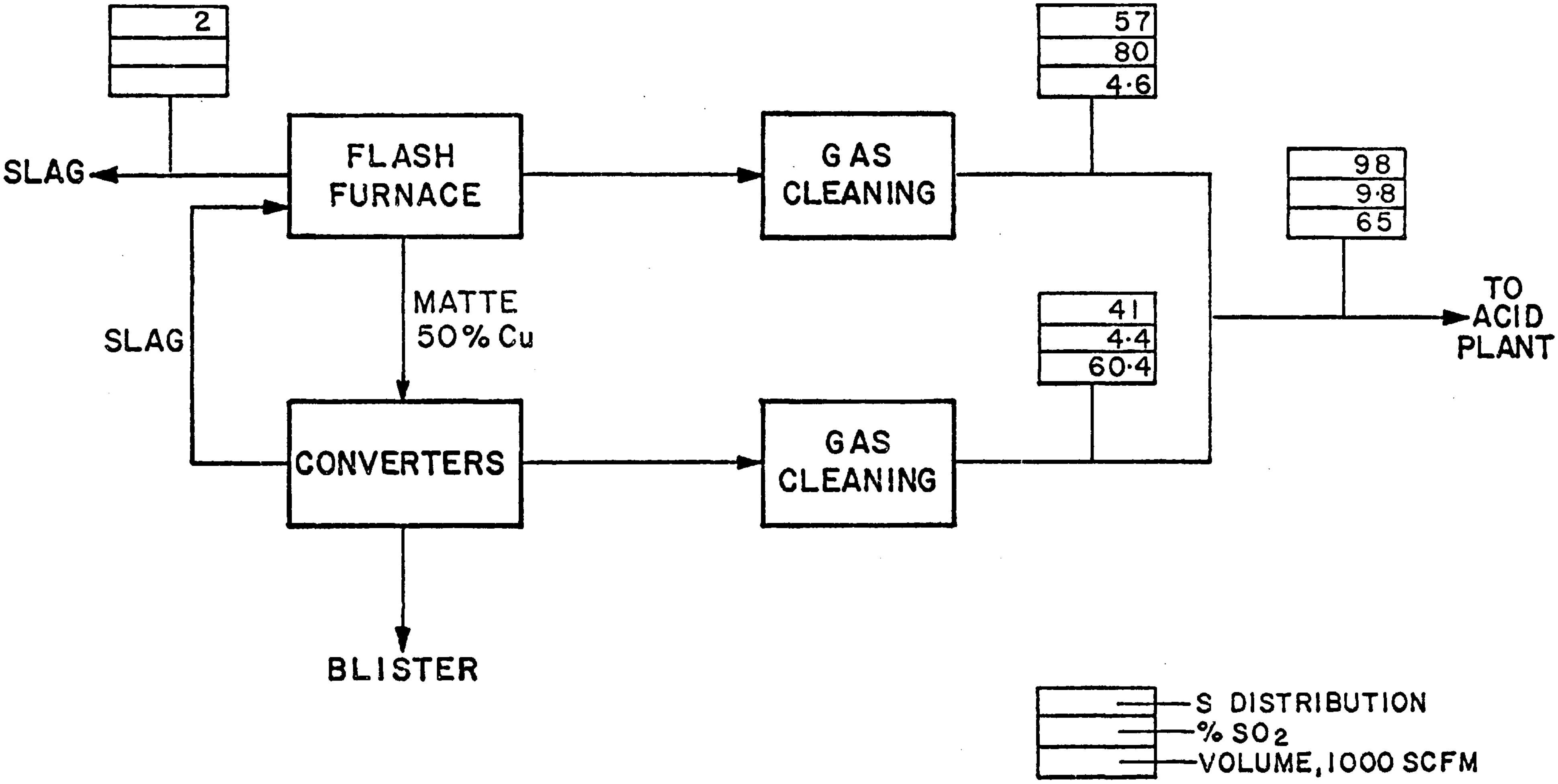


FIGURE 6

SULFUR DISTRIBUTION IN INCO OXYGEN FLASH SMELTING

THROUGHPUT = 1100 MTPD OF CONCENTRATE AT 28 % Cu 29% Fe + 32.6% S



SECTION III

THE NORANDA PROCESS

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ABSTRACT

The Noranda Continuous Smelting Process commercial plant at Noranda, Quebec has been in operation since March 1973. The operating results have demonstrated the flexibility of the process in producing either copper or high grade matte from a variety of copper feed materials. No special preparation of the feed is required. Most of the process energy is provided by the exothermic heat of the smelting and converting reactions.

The 21.3 m long by 5.2 m diameter reactor at Noranda was designed for a smelting rate of 726 tons of copper concentrate per day when producing copper without oxygen enrichment of the converting air. During a test campaign when the converting air was enriched with oxygen to 30%, and excess oxygen was added above the bath to react with concentrate and matte splash, a smelting rate of 1 560 tonnes of concentrate per day was achieved.

With an 80 tonne oxygen plant the reactor smelts 1 150 tonnes of chalcopyrite concentrate per day and produces a 70%-75% Cu matte with oxygen enrichment of the air to approximately 23%. Fuel can be supplied by oil or natural gas burners, or by the addition of coal or coke to the feed.

The low silica reactor slag is slow cooled, and copper is recovered by milling and flotation for recycle. Slag tailings contain 0.28% Cu. This loss of copper to slag tailings is lower than any other process and is equivalent to achieving 0.22% copper in slags from reverberatory or electric furnaces. Refractory consumption in the Noranda Process is less than in the conventional reverberatory smelter at Noranda.

INTRODUCTION

Work on the process began in 1964 at Noranda's Research Centre in Montreal. A pilot plant with capacity of 100 tons of concentrate per day was built at the smelter in Noranda, Quebec in 1968. This plant operated for 4 years smelting about 90 000 tons of concentrate directly to copper. In 1973, a plant went into operation that was designed to smelt 726 tons per day of concentrate directly to copper. Today with the aid of 80 tons per day of oxygen and producing a 75% copper matte rather than copper the same plant has a capacity of 1 150 tons per day of concentrate. Studies are being made to add more oxygen and increase throughput to 2 000 tonnes per day. This plant operated for 2 years producing copper directly, but operation was changed to produce a high grade matte for several reasons:

(1) Throughput would increase when producing matte allowing the shutdown of a reverberatory furnace.

(2) The refinery had to operate without the electrolyte purification circuit for a year and needed the cleanest anode copper possible.

The most recent Noranda reactors have been installed at the Kennecott Minerals Company's Utah Copper Division Smelter and brought on stream during 1978 (8). Two reactors are normally operational with a third one scheduled down for general overhaul. This plant has replaced the old smelter on the same site and has an annual design treatment capacity of 900 000 tonnes per year concentrates and copper precipitates. With a projected output of 250 000 tonnes per year of copper, the Utah Smelter is the largest copper smelter in the United States. The reactors are operating at an oxygen enrichment of 34%. The process was designed to produce copper matte containing 70%-75% Cu due to the level of arsenic and bismuth in the concentrate.

Environmental pollution in the vicinity of the smelter has been dramatically reduced by installation of the new smelting and emission control facilities.

The cylindrical reaction vessel is shown in Figure 1. Filter cake copper concentrates and silica flux are thrown onto the surface of the reactor bath, which is maintained in a highly turbulent state by air, or oxygen-enriched air, introduced through a series of tuyeres. The exothermic oxidation of sulfur and iron provides part of the heat for smelting; additional heat is supplied by a burner or by the addition of solid fuel added with the feed.

When producing metallic copper, high-grade matte co-exists inside the reactor with the copper and slag. The slag is highly oxidized, and contains 20-30% magnetite (Fe_3O_4) in solid and dissolved form, and 8-12% copper. Since the reactor bath is highly agitated, magnetite build-up is not a problem. When producing matte, the slag contains less magnetite and copper, ranging from 15-25% Fe_3O_4 and 3-7% Cu, depending on the matte grade. The lower-magnetite slag is more fluid and allows operation of the reactor at a slag temperature of 1200°C, compared to 1230°C while making metallic copper.

The metallic copper or matte is periodically tapped through a taphole in the side of the reactor. Slag is skimmed from one end of the reactor. The slag is slowly cooled, crushed, and sent to a concentrator where its copper content is recovered by flotation (5). The slag concentrate is recycled to the reactor, and the low-copper tailings are discarded.

Off-gas leaves the reactor through a water-cooled hood either to a waste heat boiler, or an evaporative water spray cooler, and then to an electrostatic precipitator. Dust collected in the gas handling system may be partially or wholly recycled to the reactor, or treated separately for by-product recovery. The continuous flow of high strength SO_2 gas facilitates the production of sulfuric acid.

REACTOR OPERATION

The reactor at the Horne smelter is 21.3 m long and has an outside diameter of 5.2 m (Figure 1). The reactor is normally rotated only through 48° to bring the tuyeres above the bath (Figure 2), but can be rotated to other positions for emptying out the reactor at the end of a campaign or while rebricking. The reactor at the Horne smelter is turned with a hydraulic piston, but at Kennecott a girth gear and pinion drive is used.

The reactor is lined with direct bonded chrome-magnesite brick, except in the high wear areas adjacent to the burners, slag line and tapholes where fused-cast magnesite-chrome brick is used.

REACTOR FEED

The reactor at the Horne smelter normally smelts chalcopyrite concentrates from about 25 different mines with a range of analysis: 20-28% Cu; 0.8-9% SiO_2; 25-35% S; 0.1-8% Pb; and 0.1-10% Zn. Chalcocite concentrates, precipitate copper, copper and zinc refinery residues and smelter reverts have also been fed to the reactor. No special blending of concentrates is required.

The reactor has also operated with wide variations in flux analysis and size. At Noranda, the flux normally contains 67% silica, 4% iron and 12% alumina and is usually crushed to -1 cm.

The feed system to a reactor is simple (Figure 3). Copper concentrate, recycled slag concentrate, flux and coal are fed from surge bins to a conveyor belt. Precipitator dust is agglomerated in a drum mixer to minimize dusting. A high speed belt (slinger) throws the charge into the reactor through a port in the feed-end wall. At Noranda a pelletizer is used to mix concentrates and dust but it is thought that future plants should only pelletize the dust.

Material sized less than about 8 cm can be handled through the feed system. Splashings from tapping, and accretions from the water-cooled reactor hood are crushed and recycled. Larger material, such as scrap anodes and ladle skulls can be charged through the mouth of the reactor, but when producing matte, these are conveniently handled in the converters.

CONVERTING AIR

The reactor at the Horne smelter is provided with sixty 5.4 cm inside diameter tuyeres, spaced at 15.9 cm. About fifty tuyeres are normally in use, blowing at air flow-rates of 0.33 to 0.40 Nm^3/s. The tuyeres are cleaned with a Gaspe puncher equipped with tuyere silencers.

In 1975, a small oxygen plant rated at 85 tons per day of 99% oxygen was purchased and re-erected at the Horne smelter. This has allowed a modest degree of oxygen enrichment of the converting air, up to 24% O_2. At Kennecott oxygen enrichment of 34% is routinely used.

FUEL USAGE

The heat required to supplement that generated by the exothermic smelting reactions can be provided by either gas, oil, or solid fuel in a variety of combinations. Initially, the reactor was fired by a natural gas/heavy oil burner at the feed-end and a natural gas burner at the slag-end of the vessel. The slag-end burner was eliminated when it became evident that it was not necessary. The feed-end burner was modified to give a longer, less intense flame as part of the program to reduce the brick wear opposite the tuyere line and it has since been used only to keep the vessel hot during long periods of standby. It has been found more effective to add solid fuel to the reactor bath and burn it in contact with the charge using the tuyere air to supply the oxygen for combustion. Approximately 30% of the fuel requirement is provided by Bunker "C" oil which is introduced into the furnace from a 0.6 cm diameter nozzle located in the feedport. No attempt is made to atomize the oil which flows as a continuous jet onto the bath surface, and off-gas analysis shows that complete combustion takes place. Alternatively, dependent upon cost, natural gas may be piped through the feedport.

The major part of the fuel requirement is supplied by solid fuel which is introduced into the furnace with the feed.

Several coals and cokes have been used and typical assays are shown in Table I.

REACTOR CONTROL

The basic control parameters of the Noranda process are (1) the oxygen/concentrate ratio, which is controlled at the value required to oxidize the feedstock sulfur and iron, and to give copper or matte of a desired grade, and (2) the flux/concentrate ratio, which is controlled to maintain the desired iron/silica ratio in the slag. The slag and metallic copper or matte are tapped so as to keep relatively constant levels in the reactor. Dipping a steel bar into the bath serves as a simple hourly level check. The fuel input regulates temperature to provide the required slag fluidity.

COMPUTER CONTROL

The reactor originally had analog instrumentation but a few years after operation began a process control computer was added. This computer monitors all process signals and provides operators and management with appropriate reports on the operation of the process. Also the routine control calculations, previously performed by the operators, are now performed by the computer. It is now possible to use more complex control loops than were feasible with the analog system. For example, the oxygen/concentrate ratio with the analog system was a straight ratio of oxygen to concentrate, but with the computer it is possible to ratio the concentrate to the net oxygen available after the requirements of all the fuels have been met. This has led to improvements in the thermal regulation of the reactor with fewer incidents of viscous slags due to low temperature.

Because of the complexities involved with on-line chemical analysis, temperature and level measurements, feed forward control is not used. The large mass of matte held in the reactor smooths out swings in the temperature or composition and allows feedback control to be used effectively. The only assays required when making copper are the Fe and SiO^2 in the slag; when matte is the product, Fe and Cu in the matte are analysed as well.

COPPER AND MATTE HANDLING

Copper or matte is tapped intermittently through one of two tapholes in the barrel of the reactor. Each taphole is faced with a copper block with a 4 cm diameter hole. The tapholes are opened with an oxygen lance and plugged with clay by a clay gun.

Reactor copper contains about 2% sulfur and additional air blowing is required in an anode furnace. A standard anode furnace would have to be fitted with extra tuyeres for this purpose.

When high grade matte from the reactor is treated in a converter, very little slag is formed and the slag tends to be dry. The standard Noranda practice of periodically adding cold pigs of copper to the converter, for temperature control, is followed. After two or three successive reactor copper charges have been blown, the accumulated slag is fluidized by allowing the temperature to rise just before the copper blow is completed. Slag is skimmed, and the converter is cleaned. The small amount of slag produced is crushed and recycled to the reactor. The high grade of the matte gives a much more constant and regular converter operation than does the low grade matte from reverberatory furnaces. With 100% air infiltrating at the converter hood, the gas contains 8.7% SO_2 during most of the blowing period.

SLAG HANDLING AND MILLING

Slag is skimmed into a short launder which directs the slag stream into a 13 m^3 cast steel ladle, which holds about 35 tons of slag.

The slag ladle is transported to the slag cooling area and water is added to the surface of the molten slag for 24 hours. The solidified slag is tipped out of the ladle and is broken with a pneumatic impact hammer. The broken slag (minus 60 cm) is taken to the crushing plant where it is crushed to minus 6 cm and delivered to the concentrator.

The crushed slag is ground in ball mills to 72-76% minus 44 microns at a power consumption of 25 kWh/tonne of slag using a 2.5 kg/tonne of grinding steel and 0.22 kg/tonne of lining steel. The slag is floated at its natural pH to produce a slag concentrate which is filtered and recycled to the reactor.

The slag lends itself well to autogenous grinding and a semi-autogenous mill is being installed at Noranda.

Table 2 shows the milling results for 1979.

Because the silica content of Noranda Process slag is about 23% compared to 37% for a reverberatory or electric furnace slag, the amount of slag tailings produced from the Noranda Process is less than slag from conventional smelting. At Noranda, these tailings assay 0.28% Cu compared to 0.41% Cu from a reverberatory furnace slag. The copper loss to slag from the Noranda Process is therefore about half that for a reverberatory furnace.

GAS HANDLING

Off-gases leave the reactor through a 3.7 m x 2.5 m mouth in the barrel of the reactor. A water-cooled hood covers the mouth whether the reactor is operating or on standby (Figure 2). This avoids the spill gas common to converter operations. To minimize air infiltration, moveable flaps are installed on all four sides of the hood. The front flaps can be opened to allow charging through the reactor mouth. The draft at the hood is adjusted to maintain a slightly negative draft at the feedport. The air that infiltrates the flaps on the reactor hood and throughout the flue system is controlled to between 50% and 100% of the volume of the off-gases from the reactor by controlling the draft at the hood.

At the Horne smelter, off-gases enter an evaporative water cooler at 815°C to 930°C and are quenched by the water sprays to about 340°C. Dust collected in the cooler is in a fine dry condition and is recycled with copper concentrate. A waste heat boiler was not installed at the Horne smelter, in order to minimize capital expenditure and because the smelter's hydroelectric power contract was fixed to a historical minimum demand and there could be little cost saving by generating additional power. At Kennecott, waste heat boilers are operating well with no problems associated with the metallurgical process.

From the evaporative cooler, the off-gases go to an electrostatic precipitator from which the dust is removed continuously and pneumatically conveyed to the pelletizer. The dust catch in the precipitator and evaporative cooler is generally 5% by weight of the high-lead, high-zinc copper concentrate smelted at Noranda.

Because of a lack of markets for sulfuric acid in the Noranda area, an acid plant was not installed when the reactor was built, and the clean gas is vented from the existing converter gas stack.

MANPOWER REQUIREMENTS

The reactor and its ancillary operations, including the delivery of concentrate, and the transfer of metallic copper or matte to the converter aisle, and cooling and breaking of slag, require a total of 42 men working 40 hours a week on 8-hour shifts as shown in Table 3.

OPERATING RESULTS

The reactor at the Horne smelter was designed to expand smelter capacity by 726 tonnes per day of copper concentrate. Following start-up in March 1973, the reactor successfully operated near its design rate for almost two years, producing metallic copper. During a special campaign late in 1973, the reactor was operated at smelting rates up to 1 650 tonnes per day, using purchased liquid oxygen to enrich the converting air. Early in 1975, copper concentrate receipts at the smelter began falling off, and one of the three reverberatory furnaces was shut down. In order to balance the smelter production rate and concentrate receipts, the Noranda Process reactor was switched to the production of high-grade matte for which there was spare converting capacity in the existing five converters. The reactor throughput was thus increased to about 900 tonnes per day of concentrate. When the 80 tonnes per day oxygen plant became available in July 1975, the smelting rate was increased to 1 150 tonnes per day.

COPPER PRODUCTION WITH AIR

Table 4 gives results for a period when over half of the new feed was a high-magnetite purchased secondary material. During this period, heavy oil and gas were supplemented by the addition of coke fines to the bath. The relative contributions of these fuels to the overall fuel ratio were: natural gas - 58.7%; oil - 21.6%; coke - 19.7%. Based on subsequent reactor operating experience, it is believed that these tonnages could have been equalled or exceeded if the maximum air blowing rate had been used. With a higher coke addition, and a lower burner fuel rate, a lower fuel ratio would have resulted.

COPPER AND MATTE PRODUCTION WITH HIGH OXYGEN ENRICHMENT

Nine months after the reactor was started up, a special campaign was conducted to test the use of a high degree of oxygen enrichment (6). This test was carried out in conjunction with Kennecott Copper Corporation who subsequently used the data to establish design parameters for their modified Utah smelter.

For the test, liquid oxygen was purchased and shipped to the Noranda smelter. The oxygen was added to the converting air, and also to the feed-end burner in amounts in excess of that required for combustion of the natural gas used during the tests; the excess oxygen introduced through the burner reacted with concentrate and matte splash above the bath.

The reactor operation was very smooth, with good heat distribution and no local overheating. Slags were more fluid than with air operation, and this allowed the slag temperature to be lowered to 1200°C. Results for one of the test periods with the reactor producing copper from a mixture of chalcopyrite concentrates and copper precipitate (15:1 weight ratio) are shown in Table 5. The average concentrate smelting rate was 1 560 tonnes per day. Smelting rates and fuel ratios as a function of oxygen enrichment for other test periods, shown in Figure 4, clearly demonstrate the advantage of using oxygen to increase reactor throughput and to decrease fuel consumption.

MATTE PRODUCTION AT LOW OXYGEN ENRICHMENT

Table 6 gives the results for 1979. The available time for smelting in the year was decreased by the smelter stopping operations whenever the weather caused SO_2 to descend on populated areas. The smelting rate was limited by the capacity of the blower for the tuyere air; winter operation when the air is dense being better than summer operation.

REFRACTORY WEAR

The tuyere line is subject to substantially greater wear during copper production than during matte production. However, because the converter is eliminated the brick wear is still less than conventional reverberatory furnace smelting. With matte production a protective build-up occurs on the tuyere line, and tuyere line wear is not a problem. In campaigns in which the reactor has been operated only to produce matte, the tuyere line thickness has ranged from 38 to 43 cm throughout the campaign.

Refractory wear in other areas of the reactor has been slight, and refractory repairs usually consist of replacing the tuyere line or burner area, plus taphole and collar maintenance.

MINOR ELEMENTS

In any smelting process producing metallic copper directly from concentrates, the metallic copper will contain more arsenic, bismuth and antimony than copper produced by converting matte in a separate converter (7). Concentrates with appreciable amounts of As, Bi and Sb cannot be smelted directly to copper and produce an acceptable anode copper. In the Noranda Process, such concentrates are best smelted to high grade matte in the reactor, followed by conventional converting or else if the copper is the reactor product it is given a special treatment in the anode furnace.

At the Horne smelter, some concentrates contain high levels of lead and zinc which give rise to a high re-circulating dust load in reverberatory smelting-converting. The more oxidizing conditions in the Noranda Process reactor promote the slagging of lead and zinc, and reduce the dust load. This is used to advantage by smelting high lead and zinc concentrates and residues in the reactor rather than in the reverberatory furnaces.

Not only does the reactor at Noranda have an excellent copper recovery but the gold and silver recoveries are higher than the reverberatory furmace recoveries.

SUMMARY

The Noranda Process has proven to be a flexible and economical method for smelting copper concentrates, and simple to operate. It makes good use of the heat generated by exothermic smelting and converting reactions and the small amount of additional heat required can be supplied in a variety of ways using gaseous, liquid or solid fuels. The process is ideally suited to the use of oxygen enrichment which gives both higher throughputs and lower fuel consumptions, and increases the strength of SO_2 in the off-gas. No special blending of the copper concentrate feed is required and wet concentrate can be used directly. Coarse secondary material can be fed to the reactor, and high magnetite reverts which would upset the operation of other types of furnace can be consumed with ease. The Noranda Process has a high inherent stability due to the large mass of matte in the reactor which acts as a buffer against temperature changes and swings in the copper content of the charge. The operation and behaviour of the Noranda process are more predictable than conventional smelting processes, and it has been found that acceptable operation can be achieved by personnel with limited smelting experience.

Table 8 shows the projections for a reactor smelting a 25% Cu chalcopyrite concentrate in a single reactor.

REFERENCES

(1) McKerrow, G.C., Themelis, N.J., Tarassoff, P., and Hallett, G.D., The Noranda Process, Journal of Metals, Vol. 24, No. 4, April, 1972, pp 25-32.

(2) Themelis, N.J., and McKerrow, G.C., Production of Copper by the Noranda Process, in Advances in Extractive Metallurgy and Refining (ed. M.J. Jones), The Institution of Mining and Metallurgy, London, 1972, p 3.

(3) McKerrow, G.C., Hallett, G.D., and Tarassoff, P., Oxygen in the Noranda Process, in The Future of Copper Pyrometallurgy (ed. C. Diaz), the Chilean Institute of Mining Engineers, Santiago, Chile, 1974, p 121.

(4) Mills, L.A., Hallett, G.D., and Newman, C.J., Design and Operation of the Noranda Process Continuous Smelter, in Extractive Metallurgy of Copper (ed. J.C. Yannopoulos and J.C. Agarwal), the Metallurgical Society of AIME, New York, N.Y., 1976, p 458.

(5) Godbehere, P.W., and Brooks, B.W., Milling of Copper Slags from the Noranda Continuous Smelting Process, Paper presented at the 106th AIME Annual Meeting, Atlanta, Georgia, March 6-10, 1977.

(6) Bailey, J.B.W., Beck, R.R., Hallett, G.D., Washburn, C., and Weddick, A.J., Oxygen Smelting in the Noranda Process, Paper presented at the 104th AIME Annual Meeting, New York, N.Y., February 16-20, 1975.

(7) Mackey, P.J., McKerrow, G.C., and Tarassoff, P., Minor Elements in the Noranda Process, Paper presented at 104th AIME Annual Meeting, New York, N.Y., February 16-20, 1975.

(8) Dayton, S., Utah Copper and the $280 Million Investment in Clean Air, Engineering and Mining Journal, April, 1979

(9) Bailey, J.B.W., and Storey, A.G., The Noranda Process After Six Years' Operation, paper presented at the 18th Annual CIM Conference of Metallurgists, Sudbury, Ontario, August 19-23, 1979.

(10) Bailey, J.B.W., and Dutton, W.A., Computer Control of the Noranda Continuous Smelting Process, paper presented at the 107th AIME Annual Meeting, Denver, Colorado, February 26-March 2, 1978.

(11) Bailey, J.B.W., and Balfour, G.C., Refractory Practice for the Noranda Process, paper presented at the 109th AIME Annual Meeting, Las Vegas, Nevada, February 24-28, 1980.

TABLE 1

TYPICAL ASSAYS OF SOLID FUELS

Fuel Type		% H_2O	% Ash	% S	% Fixed C	% Volatiles	Gross Kcal/Kg (Dry Basis)
Anthracite		2	4-5	0-1	82	6	7 500
Semi-Anthracite		5-6	10-12	0-1	76	15	7 200
Bituminous	A	6-8	4-5	1-2	71	25	8 000
"	B	10-12	15-25	3-5	50	28	6 700
Coke	A	10-12	16-18	0-1	79	5	6 700
"	B	10-12	24-28	1-2	75	4	5 300

The reactor operates efficiently with any of these materials provided that the particle size is <1 cm. Coal with the required particle size is readily available and pulverized fuel is not required.

TABLE 2

SLAG MILLING RESULTS FOR 1979
(REACTOR PRODUCING MATTE

	Tonnes	% Cu	Cu Distribution
Concentrate	31 432	39.72	95.06
Tailings	227 935	0.28	4.94
Head Slag	259 367	5.06	100.00

TABLE 3

MANPOWER REQUIREMENTS

1 General Foreman	- Supervises repair work and shift.
4 Shift Crews consisting of:	
1 Shift Foreman	- Supervises operation on shift.
1 No. 1 Operator	- Adjusts reactor conditions, keeps records.
1 No. 2 Operator	- Takes bath depths, cleans mouth and evaporative cooler.
1 Copper/Matte Tapper	- Taps and delivers copper/matte to the converter/anode furnace.
1 Slag Tapper	- Taps slag and takes it to cooling area.
1 Loader Operator	- Sends concentrate from storage building to reactor bins.
1 Feeder	- Operates feed system from bins to reactor.
1 Puncher	- Operates Gaspe puncher.
1 Dustman	- Operates precipitator - pumps dust.
1 Labourer	- Receives concentrates to bins.
1 Slag Cooling Area Operator	- Cools and breaks slag.

TABLE 4

RESULTS OF PERIOD ON COPPER PRODUCTION WITH AIR

INPUT	Dry Tonnes	H_2O, %	Composition (dry basis), % Cu	Fe	SiO_2	S	Pb	Zn	Fe_3O_4
Copper concentrate	2 681	9.0	23.5	29.0	3.0	33.0	3.2	3.8	-
Purchased reverts	3 005	5.0	29.5	22.9	16.2	13.0	0.1	0.4	22.5
Slag concentrate	1 113	10.8	40.6	19.6	15.0	8.9	1.3	2.1	-
Flux	440	4.0	0.1	4.5	66.8	-	-	-	-
Precipitator dust	281	-	11.9	4.0	2.0	13.1	19.4	13.4	-
OUTPUT									
Copper	1 492	-	96.4	0.2	-	2.0	0.4	-	-
Slag	4 513	-	11.0	37.5	22.7	2.2	2.1	3.0	-
Evaporative cooler dust	79	-	45.0	9.5	5.9	12.8	3.3	2.3	-
Precipitator dust	281	-	11.9	4.0	2.0	13.1	19.4	13.4	-

PRODUCTION DATA

Instantaneous smelting rate, tonnes/day concentrate plus purchased reverts	960
Average blowing rate, Nm^3/s	13.83
Tonnage oxygen, tonnes/day	0
Oxygen content of tuyere blast, %	21
Average fuel rate, 10^6 kcal/tonne of copper concentrate plus purchased revert	1.33
Feed-end burner firing rate, 10^6 kcal/h38	
Solid fuel rate (5.8×10^6 kcal/tonne), tonnes/h	1.66

TABLE 5

RESULTS OF PERIOD ON COPPER PRODUCTION WITH OXYGEN ENRICHMENT

	Dry		Composition (dry basis), %					
INPUT	Tonnes	H_2O, %	Cu	Fe	SiO_2	S	Pb	Zn
Copper concentrate and precipitate	4 108	7.7	28.5	27.1	4.7	30.0	1.1	4.6
Slag concentrate	623	8.4	39.2	19.9	15.3	11.1	0.1	2.9
Flux	567	9.2	-	2.1	78.2	-	-	-
Precipitator dust	210	-	22.8	6.8	3.2	11.7	5.0	16.9
OUTPUT								
Copper	1 104	-	97.5	0.4	-	2.0	-	-
Slag	3 392	-	9.6	36.7	25.0	1.2	1.2	5.7
Evaporative cooler dust	33	-	37.1	19.4	11.2	6.2	1.0	4.0
Precipitator dust	229	-	22.7	6.8	3.2	11.8	4.0	17.1

PRODUCTION DATA

Instantaneous smelting rate, tonnes/day	1 560
Average blowing rate, Nm^3/s	16.61
Tonnage oxygen, tonnes/day	325
Oxygen content of tuyere blast, %	30
Average fuel rate, 10^6 kcal tonne of copper concentrate	0.17
Feed-end burner firing rate, 10^6 kcal/h	9.1
Solid fuel rate	0

TABLE 6

1979 RESULTS, MATTE PRODUCTION WITH LOW OXYGEN ENRICHMENT

	Dry		Composition (dry basis), %					
INPUT	Tonnes	H_2O, %	Cu	Fe	SiO_2	S	Pb	Zn
Copper concentrate	311 642	7.5	25.3	29.0	5.5	31.1	1.9	3.4
Slag concentrate	31 432	10.0	39.7	16.8	11.7	13.3	2.6	2.6
Flux	54 706	4.0	-	4.2	71.5	-	-	-
Precipitator dust	15 600	-	7.8	4.4	1.1	11.4	26.1	7.5
OUTPUT								
Matte	107 887	-	72.5	3.3	1.2	21.0	2.1	0.6
Slag	259 367	-	5.06	36.4	22.6	1.6	1.7	4.2
Precipitator dust	15 600	-	7.8	4.4	1.1	11.4	26.1	7.5

PRODUCTION DATA

Instantaneous smelting rate, tonnes/day	1,089
Average blowing rate, Nm^3/s	17.54
Tonnage oxygen, tonnes/day	73.2
Oxygen content of tuyere blast, %	23.9
Average fuel ratio, 10^6 kcal/tonne of copper concentrate	0.8
Feed-end burner firing rate, 10^6 kcal/h	11
Solid fuel rate, tonnes/h	3.97

TABLE 7

REACTOR REFRACTORY CONSUMPTION

AVERAGE REFRACTORY CONSUMPTION

kg/tonne of concentrate

	Copper Production Reactor	Matte Production Reactor	Conventional Smelter
Smelting Vessel	1.55	0.55	1.35
Converter	-	0.75	0.75
Anode Furnace	0.05	0.05	0.05
TOTAL	1.60	1.35	2.15

TABLE 8

PROJECTED REACTOR OPERATING PARAMETERS

USING 376 TONNES PER DAY OF OXYGEN

	Mode of Operation			
	75% Matte	75% Matte	Copper	Copper
Copper concentrate smelting rate, tonnes/day	1 905	2 177	1 696	1 860
Tonnage oxygen to tuyeres (at 96%), tonnes/day	376	376	376	376
Tuyere blowing rate, Nm^3/s	21.24	21.24	21.24	21.24
Oxygen content of tuyere blast, %	32.6	32.6	32.6	32.6
Feed end burner firing rate, 10^6 kcal/h	0	22.7	0	15
Solid fuel rate, 10^6 kcal/h	11.3	0	7.3	0
Total fuel ratio, 10^6 kcal/tonne copper concentrate	0.17	0.28	0.14	0.23
Mouth off-gas flowrate (dry basis), Nm^3/s	21.2	27.4	21.2	25.5
SO_2 at the mouth (dry basis), %	21.1	18.7	22.8	20.8
Expected air infiltration before acid plant, Nm^3/s	15.9	21.2	15.9	19.1
Annual throughput of copper concentrate, tonnes	585 000	668 000	450 000	494 000

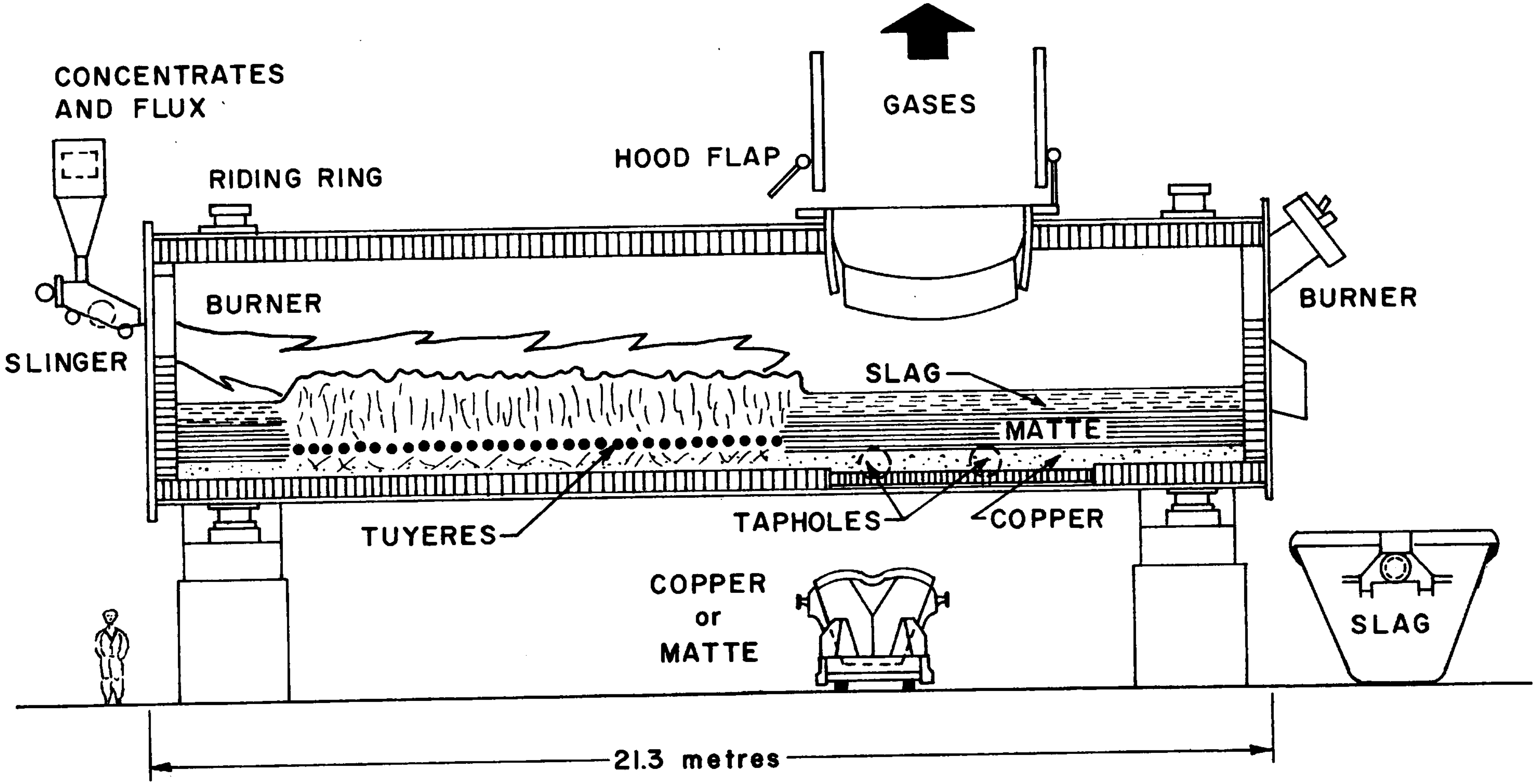

Figure 1 - Longitudinal section through Noranda Process reactor.

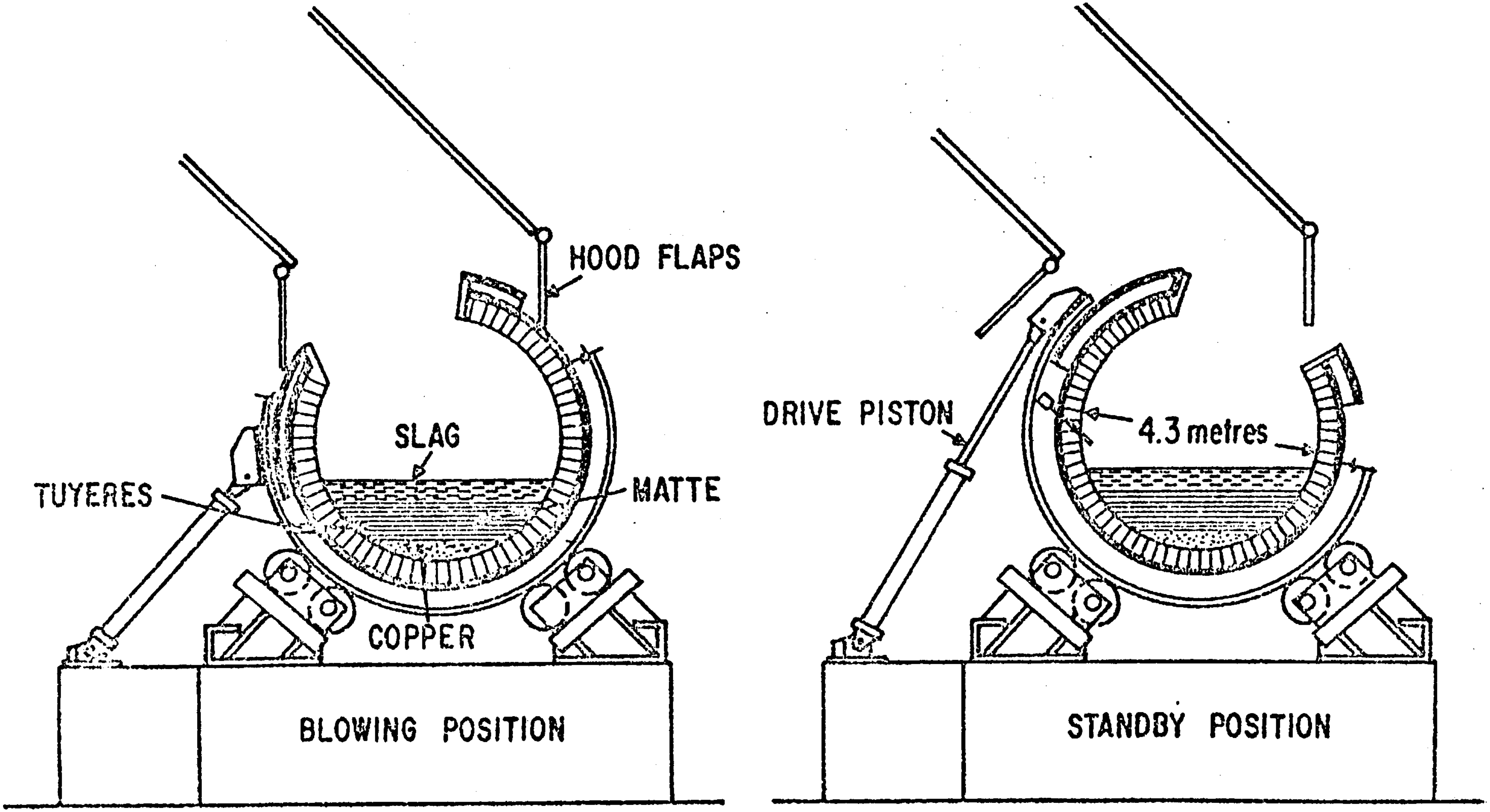

Figure 2 - Sections through reactor in blowing and standby positions.

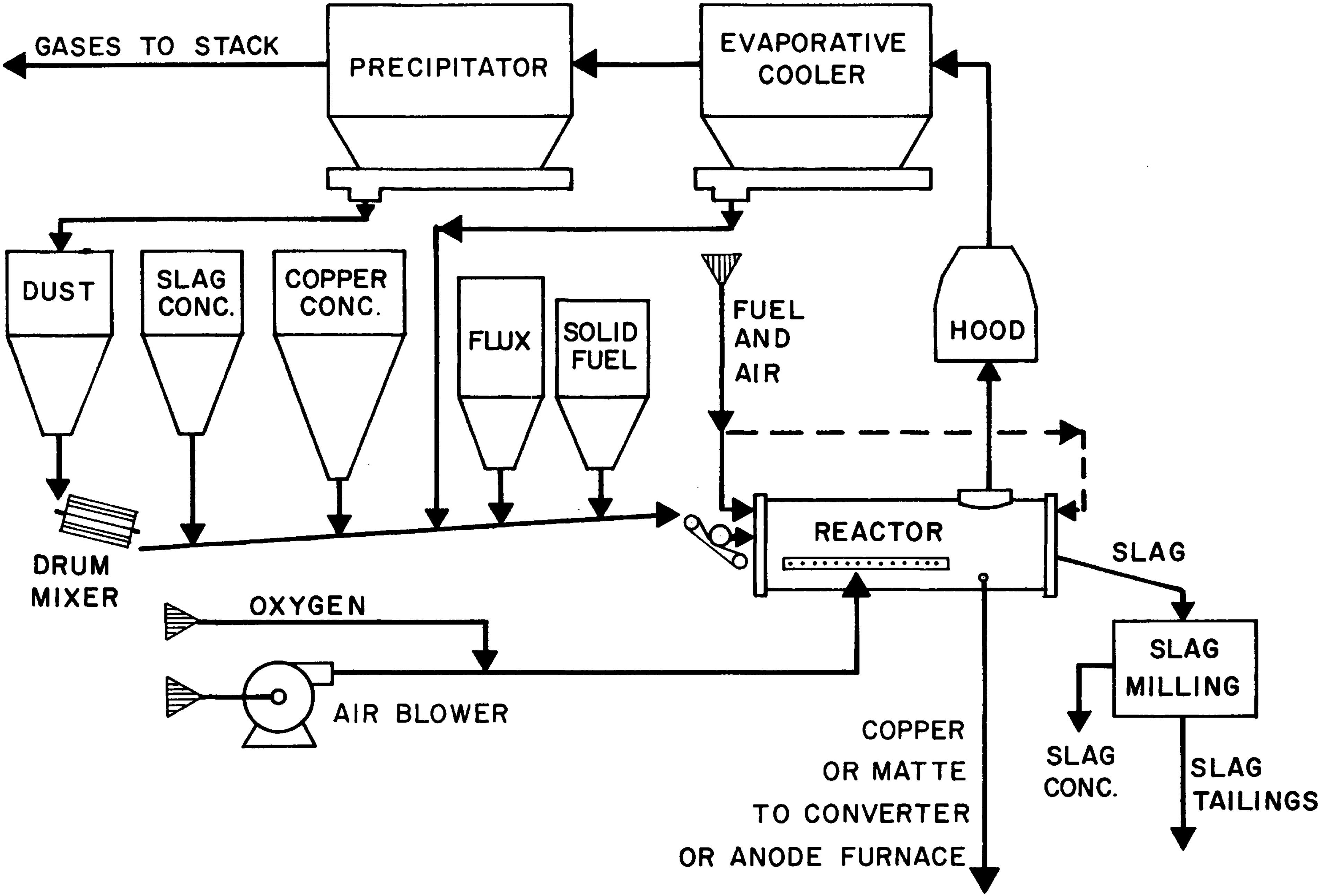

Figure 3 - Flowsheet for Noranda Process reactor.

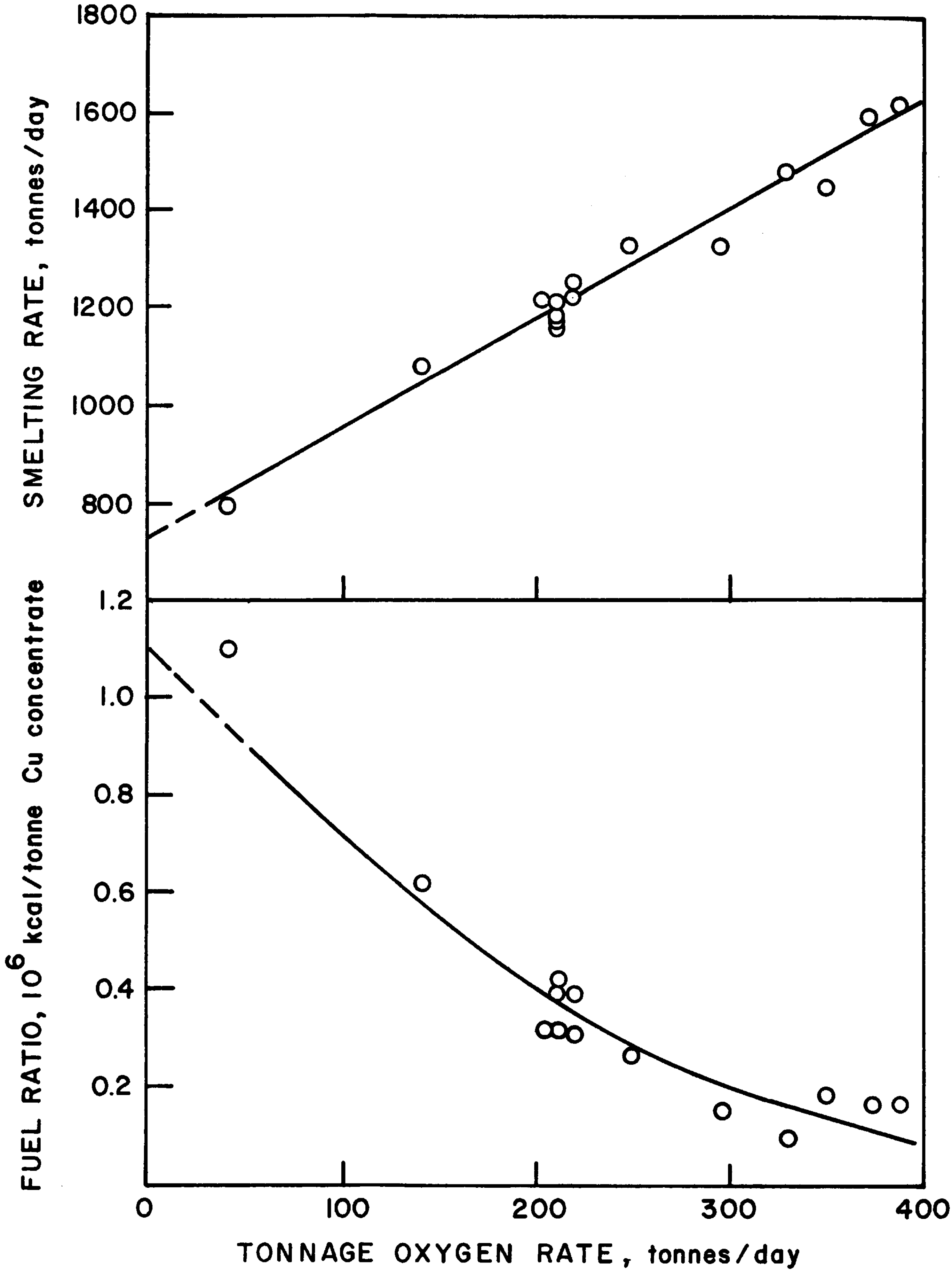

Figure 4 - Copper concentrate smelting rate and fuel ratio as a function of tonnage oxygen input (reactor producing

SECTION IV

RECENT OPERATION OF MITSUBISHI CONTINUOUS COPPER SMELTER AT NAOSHIMA

BY T. SUZUKI

Recent Operation of Mitsubishi Continuous Copper Smelter at Naoshima

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Metallurgy Department
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Paper to be presented at British Columbia Copper Smelting and Refining Technologies Seminar, Organized by the B.C. Ministry of Energy, Mines and Petroleum Resources, November 5 and 6, 1980, Vancouver, B.C.

ABSTRACT

Mitsubishi Metal Corporation has been operating Mitsubishi Continuous Copper Smelter at Naoshima for six year since 1974. During the past period, various kinds of improvement works have been directed toward attaining stable operations and lower operation cost. First of these was modification of furnace shape from oval to circular. Second was the addition of water-cooled copper jackets to the side wall of the furnaces. Third was changing the roof burners to short flame type burners, which are now installed at the side wall. Fourth was mixing of coal with copper concentrate to replace part of the fuel oil. Fifth, was higher oxygen enrichment of lance blast. As a result the furnace campaign was extended to around one and one half years, and the monthly production of copper was increased by 10 to 20% over the rated capacity. The paper discusses respective modifications, and then shows a model smelter plan with maximum utilization of oxygen.

INTRODUCTION

Mitsubishi Continuous Copper Smelting Converting Process (Mitsubishi Process) is one of the most advanced copper smelting technologies in that it is an integrated continuous process from smelting stage through converting stage (1-2). To develop the process Mitsubishi Metal Corporation (MMC) spent five years operating a pilot and a semi-commercial plant, having a capacity of 4 000 tonnes of copper per month at its Naoshima smelter. Since then various improvements and modifications were performed, and now the campaign length exceeds one year and monthly copper production exceeds the designed capacity by 10-20%.

Many of the advantageous features of the Mitsubishi Process are attributable to its continous and integrated nature of operation. For example, one of the most remarkable advantages is that the Mitsubishi Process can easily compley with most stringent regulations on sulfur oxides emission. Fugitive gases are minimized and can easily be captured, if there is any, because molten metallurgical products are transfered through launders. Furnaces and associated facilities can be designed compact thereby decreasing capital expenditure. At the same time operation is very much simplified because such jobs are eliminated as matte tapping, ladle transportation, and converter slag skimming, which are indispensable in conventional batchwise operations.

Since the new "Clean Air Act" was enacted in 1975, all the primary smelters in North America were obliged to make provisions to recover more than 90% of sulfur input, and it is expected that the situation would become more stringent in the future. With such background, the Mitsubishi Process could provide one of the best solutions to comply with the regulation.

After careful feasibility studies, Texasgulf Canada Ltd., determined in 1975 to apply the Mitsubishi Process for its new copper smelter in Timmins, Ontario, and started detail engineering works immediately followed by construction (3). The reasons for their choice of the process are that:

(1) It is proved in commercial scale at Naoshima Smelter,

(2) It is not labour intensive,

(3) A converter aisle is eliminated as molten matallurgical products are transferred through launders,

(4) The capital cost is competitive to the conventional process,

(5) SO_2 recovery is high and emission is low.

The new copper smelter was originally scheduled to be completed in 1978. However, the completion was delayed by two years due to unfavourable copper market conditions. It is expected to start-up in April, 1981.

Taking advantage of the opportunity to attend the British Columbia Copper Smelting and Refining Technologies Seminar, the latest operation data of the continuous plant at Naoshima will be introduced and various improvements and test works performed during the past six years will be discussed.

PROCESS DESCRIPTION

The detailed instructions on the process features of the Mitsubishi Process and the Naoshima plant were published on various occasions, but the essential parts are summarized below for better understanding.

The Mitsubishi Continuous Copper Smelting Process is a process to produce blister copper continuously from copper-bearing raw materials by a series of furnaces, e.g., smelting, slag cleaning and converting furnaces, which are linked together. A schematic flowsheet is shown in Figure 1. The designed capacity of the Naoshima Plant is 4 000 tonnes of copper per month.

SMELTING FURNACE (S-FURNACE)

The smelting furnace is 8.25 m in diameter, 3.3 m in height, both inside bricks, and 1.1 m in bath depth, lined with direct bonded chrome-magnesite bricks and fused cast chrome-magnesite bricks at severe wear areas. It is provided with six or seven lances which perpendicularly penetrate the furnace roof. The concentrate is dried and continuously delivered at a constant feed rate, e.g., at 25-27 tonnes per hour, and at the same time, silica and limestone flux, return C-slag and dusts are mixed at an appropriate ratio. The mixed solid charge is fed pneumatically through five feeding hoppers into inner tubes of the lances. Oxygen enriched lance blast air is introduced through outer tubes of the lances via lance heads. Lance blast and solid charge are mixed at the lower part of the lances and injected into the molten bath with high velocity. The oxygen and air supply is adjusted to maintain the matte grade at 65% copper. As matte and slag produced are taken out by overflow, slag layer in the furnace is very thin, and the molten bath is composed mainly of matte which is rich in unoxidized sulfur and iron, and highly reactive to oxygen. As a result, the smelting reactions proceed rapidly, and the oxygen utilization efficiency of the lance blast is very high although the lances are not submerged into the bath. Recently some amount of pulverized coal is mixed with concentrate so as to replace a part of fuel oil which is burned through burners installed at the sidewall. Furnace temperature is controlled by adjusting the fuel oil rate. Combustion air is not preheated.

SLAG CLEANING FURNACE

The products of the smelting furnace, e.g., slag and matte, overflow through a common overflow hole, and are transferred to the slag cleaning furnace via a heated launder. The slag cleaning furnace is an electric furnace with 4.2 m in width, 7.0 m in length and 2.2 m in height inside bricks, provided with three prebaked graphite electrodes in row. The transformer capacity is 1 200 KVA. The slag is separated from matte, cleaned to 0.5-0.6% copper while being retained about one hour, then caused to overflow at an equivalent rate with that of transfer from the smelting furnace. The slag is finally water-granulated and discarded. The matte is siphoned out and transferred to the converting furnace through a heated launder.

CONVERTING FURNACE (C-FURNACE)

The converting furnace is 6.65 m in diameter, and 2.9 m in height, both inside bricks, and 0.75 m in bath depth. Various features of the converting furnace are essentially the same as those of the smelting furnace except the dimension and tapholes arrangement. In the converting furnace, matte is continuously converted to blister copper by blowing air into the molten bath through lances. The amount of air is adjusted to oxidize not only all the iron and sulfur, but also a part of copper in the matte to copper oxide. A small amount of limestone is added through lances, thus forming a special converter slag comprising Cu_2O-CaO-Fe_3O_4 ternary system. CaO in the converter slag is controlled at about 15%, and Cu at 15 to 20%. Under these operating conditions, the sulfur content of the blister copper can be reduced to 0.1-0.5%, which is far lower than saturation. Blister copper is siphoned out, stored temporarily in a holding furnace then delivered to the anode furnace by a ladle car, refined and cast into anodes. Converter slag is tapped through a slag hole, water-granulated, dried and returned to the smelting furnace.

All the process supplies are monitored and controlled by the computer system. Molten products are sampled every hour and analysed automatically. The results are fed back to the control system to adjust feed rates of process supplies (4).

Gases from the smelting and converting furnaces are cooled to about 350°C through respective boilers. Respective draft control dampers for the smelting and converting furnaces are provided at the outlet of the boilers. Recovered steam is sent to a recently constructed power plant. The gases are then introduced into Cottrell precipitators, and the cleaned gas from the Cottrells is sent to the acid plants. The SO_2 concentration of the combined gas is 10-11%. Major features of the furnaces are summarized in Table 1, and typical recent operation data in Table 2 and 3.

The Mitsubishi process has various advantages over the conventional processes as summarized below:

(1) <u>It is easy to operate</u>:
As both smelting and converting processes are integrated and continuous, various jobs, indispensable with conventional operations, such as tapping of matte and slag, ladle transferring of matte and converter slag, converter slag skimming, are eliminated. The process is highly mechanized and automated. Therefore operation is simplified, resulting in substantial manpower saving. At the same time, in-plant inventory of copper is reduced.

(2) The furnaces can be designed compactly:
The smelting rate, e.g., throughput per unit volume of the furnaces, is very high as the charge mixture is fed through lances and injected into molten bath, where substantial heat of reactions are evolved. Therefore, the design of the furnaces and smelter building can be compact.

(3) Fuel consumption is low:
High grade matte can be obtained at the smelting furnace while keeping copper loss of slag at an economically acceptable level. The slag can be discarded without further processing. Thus, calorific value of copper concentrates can be utilized at maximum extent. With utilization of tonnage oxygen, fuel requirement can greatly be reduced.

(4) Gas trains can be designed compactly:
Off-gas volume can be minimized especially when maximum amount of tonnage oxygen is used. Therefore, the design of the gas trains can be compact.

(5) It is easy to comply with most stringent emission regulations:
All three main furnaces are of a stationary type, which are free from any complicated mechanical structures, such as furnace tilting facilities, tight fitting hood driving mechanisms, and tuyere punching mechanism. As molten metallurgical products are transferred through covered launders, sulfur dioxide and particulates emissions are minimized and can easily be captured, if any.

As a whole, the Mitsubishi Process requires smaller capital expenditure, operation cost and manpower.

MODIFICATION AND IMPROVEMENTS

Many of the modifications during the past six years were briefly discussed in a previous paper; (5)

- The furnace shape was modified from oval to circular.

- Water-cooled copper blocks were added to cool the sidewalls effectively, and bottom cooling system was also modified.

- Roof burners were changed with short flame type burners which were installed at side walls.

- Pulverized coal was mixed to copper concentrates, thereby improving calorific value of them and replacing part of fuel oil burnt through burners.

- Highly oxygen-enriched lance blast air was tested, which could reduce fuel requirement and off-gas volume substantially, or could allow higher throughput within a limited size of the furnace.

With those modifications in the facilities and improvements in operation, one year furnace campaign life has been established for both S-and C-Furnaces, and monthly production of copper exceeded by 10 to 20% over the designed capacity. As a result, Mitsubishi Metal Corporation decided to achieve the production only with the main furnaces and the stand-by furnaces were demolished.

In Mitsibushi Metal Corporation's case, stand-by S-and-C-Furnaces were installed with two main purposes:

(1) to maintain steady copper production during boiler inspection period which is associated with main furnace line; and

(2) to allow shut-down of main furnace line that is long enough to perform various possible modifications which might be required especially, for the first commercial plant of a new process.

Monthly anode productions in the latest campaign, which is still continuing, are summarized in Table 4. This campaign will be continued further after a temporary shut down for boiler inspection at the end of November.

MODIFICATION OF FURNACE DESIGN

In the Mitsubishi Process, the furnace design is one of the most important items to be studied carefully. Turbulence given to the molten bath in the furnaces is far larger than that of the other smelting furnaces, such as reverberatory or electric furnaces, due to the lancing operation. In the original design, the furnace shape was oval shaped and the shell was divided into several parts connected by bolt-nuts with springs. On the other hand in the modified design, the furnace shape is circular and the shell is a welded and mono-block structure (6-7).

With the oval shape, the lance could be located in line parallel with long axis, thereby the accessibility to the lances was assured. However, expansion of the brick works was not homogeneous in oval shape furnaces, and sometimes abnormal expansion was observed at the shell partition area. During the furnace repair after a one year campaign, the slag line and part of the brick works on the sidewalls near the burners were repaired. The hearth brick was normally sound, because it was not subjected to corrosion by either matte or blister copper. Nevertheless, the accumulated abnormal expansion made it inevitable to reline the bottom bricks even if the individual bricks were sound. Contrary to this, with a circular furnace design thermal expansion can be almost homogeneous and abnormal expansion, as observed in the oval shape, will be avoided.

With larger diameter lances, the number of lances is reduced to half of the original number as shown in Table 5. Even at the smelter with larger capacity, the number of lances could be minimized by applying higher oxygen enrichment for lance blasts. Thus, lance arrangement is not any more a limiting factor for taking a circular furnace design.

At the same time with modification of furnace shape, the cooling system was also modified. With the turbulent flow of the molten bath, the slag line of both smelting and the converting furnaces are subjected to severe wearing conditions. In the original design, water-cooled spray boxes were provided at the slag line to cool and prevent wear of the lining in these areas. The system itself worked well. However, sidewall above the bath level also suffered from wear, presumably due to splashes of molten slag and matte and direct exposure to high-temperature burner flames. For this reason, two or three layers of water-cooled copper blocks were added in the sidewall near burners in the modified design. The same provision was also made on the furnace roofs.

Bottom cooling was also intensified. Originally the furnace bottom was lined with two brick layers composed of 350 mm thick fused-cast bricks and 230 mm thick chemical bonded magnesite-chrome bricks on tamped periclase at the very bottom. When 180 000 tonnes of solid charge was injected after the commercial plant was put into operation in March, 1974, the arched bottom structure of the smelting furnace was partly damaged. Then the bath depth was increased to 1 090 mm from the original 800 mm. At the same time, the bottom brick was reduced to single layer and the bottom air cooling was intensified. With this modification, the bottom bricks were cooled effectively and permeation of matte and slag was minimized. Figure 2 and 3 show modified S and C-furnaces.

LANCE BLOWING

Lancing techniques to feed solid materials and oxygen-enriched air into the molten bath is one of the unique features of the Mitsubishi Process. Figure 4 shows the feeding system. Solid charges, such as dry copper concentrates, silica and limestone fluxes and reverts are proportioned onto a common conveyor line, which conveys them up to the top of the smelter building. Solid charges are distributed into five feeding hoppers, out of which they are delivered pneumatically and conveying tubes are regulated as low as possible to minimize erosion of the pipeline. To the lances, oxygen-enriched air is supplied separately. Solid charge and oxygen-enriched air are mixed together at the lower part of the lances and injected into the molten bath at a high velocity. Solid particles are entrained in the bath and melted rapidly. Lance blast oxidizes the iron and sulfur and maintains the matte grade at 65% copper.

The lances are installed vertically through the roof of the furnace and the tips of the lances are positioned right above the bath surface. The original two and one-half-inch lances were recently replaced with three inch lances and the number of lances was reduced as shown in Table 5. Table 6 shows a typical operating condition of the lance.

At the end of the first one year campaign, it was found that the smelting furnace bottom was partly damaged at the area just beneath the lances as mentioned previously. It was estimated that the damage was caused by the jet of air-solid charge mixture based on the following observations:

(1) There was no damage in the converting furnace, where the amount of solid charge was far smaller and the density of the molten copper bath was far larger than the matte bath in the smelting furnace.

(2) The damage was limited only in small area just beneath the lances.

By analyzing the effect of lancing on the bottom bricks utilizing the model equation developed in connection with the lancing practices in steel industry, the bath depth in the smelting furnace was increased by about 300 mm. Further, the hearth structure was modified when the furnace shape was changed. As described previously, the bottom brick lining was changed to one layer and the area beneath the lances was lined with fused-cast magnesite-chrome bricks. The furnace bottom is now air-cooled from outside and permeation of the matte into the bricks is prevented. It is known that with permeation of matte, the mechanical strength of the bricks decreases remarkably.

Further study was performed at the Research Center to analyze behaviour of the gas-solid mixture jet in the bath and an experimental equation was developed. It was found that the lance height and solid feed rate have the largest effect on the penetration depth of the particles in the bath (8). In the operation, instantaneous feed rate at an individual lance was decreased by decreasing the flow rate of the conveying air. Lance height was checked every shift by inspecting inside the furnaces visually to check the conditions of the actual lances.

MODIFICATION OF THE BURNER

As mentioned earlier, the sidewall near the burner was one of the severe wear areas. The original burner was installed at the roof with an angle so that the flame would impinge to the center of the bath. The burner itself was made with mild steel air duct and oil atomizing gun in it. In the actual operation, there was a tendency that splashes of molten slag and matte entered inside the burner and accretions were formed, especially at the mouth. As a result, the flame became unstable and it was estimated that the sidewall or roof near the burner might be exposed directly to the hot flame in such a condition. The first measure taken to reduce refractory wear was the application of copper jackets, as mentioned previously.

As the second and more important measure, modification of the burner itself was tested. The burner tested has a combustion chamber as shown in Figure 5. The combustion chamber is lined with high alumina castable material and cooled with combustion air from outside. The oil is burned in the chamber and the hot gas produced is discharged through the mouth with high velocity. As the mouth is always exposed to high temperature combustion gas, the splashes that enter the mouth are expected to flow down out of the mouth. At first one of the old long flame type burners was replaced with a short flame burner, which was installed at the sidewall instead of the roof. Then the refractory wear around the new burner was remarkably reduced compared to the other burner areas.

In August, 1978, two of the old burners were replaced and in September, 1979, all the burners were replaced. Figure 5 shows the modified burner arrangement. With this modification, refractory wear was remarkably reduced as being demonstrated by the pictures of the brick works. Figure 6 shows brick condition around the slag-matte outlet after a one year campaign with the old type burner. Swelled parts are rows of copper jackets, and hollow parts are worn bricks. Figure 7 shows brick condition at the same area as Figure 6 after a one year campaign with the new type burners. The surface is flat, which means that brick wear was very small even after one year operation. With such improvements the repair became no longer necessary for these areas.

UTILIZATION OF COAL

The extraordinary increase in oil price since 1974 gave enormous economical impacts on the smelters. A preliminary test to replace part of the oil with coal was started at the continuous smelter to relieve the impact.

In the normal mode of coal combustion, coal is finely ground and burned with a coal burner. As the Naoshima Smelter is designed to utilize only the liquid fuel, none of the facilities have been provided to prepare and burn fine-ground coal. Thus, it was intended to burn purchased pulverized coal without further grinding. At first, coal was fed into the furnace through a drop chute. However, the combustion rate was limited. Then, it was considered to feed pulverized coal into a mixture with copper concentrates through lances by bedding the wet coal with concentrates and drying with a concentrate dryer.

To check the susceptibility of coal for firing in the dryer, the igniting temperature was measured for coal as well as for various copper concentrates. The igniting temperature for chalcopyrite type copper concentrates was 200°-250°C, 180°C when pyrrhotite was contained, 150°C for pulverized coal. However, the igniting temperature of the mixed concentrates containing 2% pulverized coal was 226°C which was higher than that of coal itself. Encouraged with these measured data, the actual test was performed.

In the first stage, 2-3 wt % of coal was added to concentrates. During treatment of the coal containing concentrates, fuel oil consumption could be decreased by the amount calorifically equivalent to coal. About 30-40% of fuel oil consumption of the smelting furnace was thus replaced with coal. By the temporary test it was found that even up to 70% of fuel oil consumption could be replaced with coal if enough oxidation air could be supplied through lances. Table 7 shows typical operation data with and without coal utilization. Calorific value was 10 000 Kcal per kg for fuel oil and 7 100-7 300 Kcal per kg for coal. Then, the calculated equivalent fuel rate for an oil-coal dual system was 1,300 l/h, which was slightly smaller than 1,370 l/h in the case when only oil was used.

By feeding coal through lances, the refractory wear is expected to decrease, because of the reduced fuel rate at the burners and the smaller effect of radiation of the flame on the wall. Coal will burn in the molten bath or at the surface of it. The heat evolved may be effectively transferred to the molten bath without making hot flame.

It was thought that the addition of coal to concentrates, among the various improvements, might have given a large contribution to improve monthly copper production. Major advantages are as follows:

(1) The bath temperature was maintained at slightly higher, by 20°C, and the molten bath had a higher fluidity as judged by visual inspection. Hourly throughput was increased as a result.

(2) The slag loss of copper was reduced.

A key to such improvements can be attributed to the improvement of local heat balance in the areas where charge materials are injected into the molten bath. Combustion of coal is expected to take place at the surface of the molten bath, then the heat evolved in this area is increased with coal, which is effectively transferred to the molten bath or to the solid charge. This direct heat transfer mechanism is one of the major advantages of the Mitsubishi Process. With the high copper—low sulfur concentrates this advantage would greatly be offset if coal is not added. This item will be discussed later.

HIGH OXYGEN ENRICHMENT OF LANCE BLAST

The lance blast for the smelting furnace of the continuous plant is normally oxygen-enriched up to 30%—40%. Utilization of tonnage oxygen in the smelting process has various advantages as follows:

(1) It decreases fuel consumption.

(2) It decreases off-gas volume and consequently the size of facilities for gas treatment.

(3) It makes it possible to smelt higher throughput within a smaller furnace.

Figure 8 shows calculated off-gas volume, fuel requirement and concentrate feed rate at different levels of oxygen enrichment. Calculations were based on the existing Naoshima plant. To confirm the calculated results, test works were performed in stages.

At the first stage, one lance was chosen and oxygen concentration was increased stepwise by five per cent at a time and finally to sixty percent. This test proved that it was free from problems such as quicker burn-out of lance and combustion of concentrates during transfer in the lance pipe.

The second stage was a full scale test, in which oxygen concentration of the lance blast for all the S-furnace lances was increased up to about fifty per cent. The following were confirmed by the test:

- There was no change in lance consumption.
- Fuel consumption and off-gas volume were reduced. At a plant with a larger capacity, autogeneous operation could be possible.
- It was easier to keep optimum bath temperature.
- Concentrate feed rate could be increased proportionally to the total oxygen content of the lance blast.

Typical test data are shown in Table 8. Oxygen concentration of S-furnace lance blast was increased from 30%-50% while concentrate feed rate was maintained unchanged. Off-gas volume was reduced almost as expected.

The range of operating conditions during high throughput tests is shown in Figure 8. Average concentrate smelted in an operating day was 580 tonnes in normal case and 760 tonnes during the test. The smelting rate was improved to 14 tonnes per square metre of hearth per day from 11 in regular operation. The limitation for the test was the capacity of existing feeding system and oxygen supply. There was no metallurgical limitation found to increase the throughput further.

MODEL PLANT WITH 100 000 TONNES ANNUAL CAPACITY

With all the improvements discussed so far, the smelting operation could be now thermally autogeneous in a larger plant by applying higher oxygen enrichment for the S-furnace lance blast.

The fuel requirements for a plant with 100 000 tonnes annual capacity are shown in Figure 9 at varying oxygen enrichment levels for three different cases. With normal chalcopyritic concentrate the smelting operation is autogeneous at 60% oxygen enrichment (type A). The composition of type A concentrate is identical with what was applied in an evaluation study of a metallurgical complex in British Columbia (9) except that the content of sulfur was revised to 35% from the original figure of 38%.

With the higher grade concentrate (type B), fuel requirement is still relatively high even at a higher level of oxygen-enrichment as shown in the figure by the line B. For such a case, apparent calorific value of the concentrate could be increased by mixing small amounts of pulverized coal as shown with line C in the figure. In this case the coal mixing ratio is 3.4%. By increasing the coal ratio, the fuel oil requirement could be reduced to zero.

Taking the autogenous point as a model case condition, a set of process calculations was made and attached to this paper as an appendix. Based on this model, energy consumption was evaluated and shown in Table 9. In the table, fuel consumption includes dryer for concentrates, anode furnaces, and launder heating-up. Power consumption includes electric furnace, production of tonnage oxygen, material handling, compressors and blowers including those for fugitive gas collection, excluding the acid plant.

With higher oxygen enrichment, the furnaces can be compact. The smelting rate, which is copper production per day per unit furnace volume including roaster, smelting furnace, converters, and electric furnace, was calculated and compared with various processes. As shown in Table 10, the smelting rate is largest for the Mitsubishi Process. With compact furnaces and gas trains, capital expenditure for the continuous plant could be reduced.

Another remarkable advantageous feature of the process is that it can easily comply with the most stringent SO_2 emission regulations. To show such a situation, the fugitive gas handling systems of various types of smelters are summarized in Table 11. The respective features were taken from published papers as far as possible, (10-12) or estimated if not available in a published material. Fugitive gas volume to be collected vary widely depending on the type of process and also on the location of the plant (different regulation). However, it is obvious that the Mitsubishi Continuous Process can easily reduce emissions without applying any extensive fugitive gas collection system.

It should also be pointed out that the Mitsubishi Process can greatly save manpower requirements as shown in Table 12.

SUMMARY

During the past six years of operation, various improvements on the furnace design and operating procedure were performed. The major items are:

(1) Furnace shape was modified from oval to circular.

(2) Water-cooled copper jackets were added at the sidewall of the furnace.

(3) Roof burners were changed to short flame type burners which were installed at the sidewall.

(4) Coal was mixed with copper concentrate to replace part of fuel oil.

As a result, the furnace campaign was extended from one to one and a half years and the monthly production of copper was increased by about 20%. It was then decided to achieve the production only with the main furnaces and demolish the stand-by furnaces.

By applying higher oxygen enrichment for S-furnace lance blast, furnaces can be designed compact and smelting can be performed thermally autogenous. Taking as a model case a smelter with 100 000 tonnes annual capacity, the results of the process calculation were shown. Various specific features such as smelting rate, fugitive gas collection, manpower requirements were compared with the other processes.

REFERENCES

(1) T. Nagano and T. Suzuki, "Commercial Operation of Mitsubishi Continuous Copper Smelting and Converting Process", in Extractive Metallurgy of Copper, Ed. J. C. Yannopoulos and J. C. Agarwel, the Metallurgical Society of AIME, New York, N.Y., 1976, pp. 439-457.

(2) T. Nagano and T. Suzuki, "Process Development for Continuous Copper Smelting and Converting", paper presented at the 15th Annual Conference of Metallurgists, the Metallurgical Society of The Canadian Institute of Mining and Metallurgy, August 22-26, 1976, Ottawa, Canada.

(3) M. P. Amsden, R. M. Sweetin and D. G. Treilhard, "Selection and Design of Texasgulf Canada's Copper Smelter and Refinery", J. Metalls, Vol. 30, July 1978, pp. 16-26.

(4) T. Shibasaki and Y. Tosa, "Application of Computer Control to Mitsubishi Continuous Copper Smelting Process", ISA (Instruments Society of America) Transactions, Vol.17, No. 2, pp 93-100, 1978.

(5) T. Suzuki and M. Goto, "Recent Improvements in Mitsubishi Process Operation", paper presented at the Annual Meeting of Arizona Section, The Metallurgical Society of AIME, December 3, 1979, Tucson, Arizona, USA.

(6) M. Goto and K. Kanamori, "Converting Furnace Operation of Mitsubishi Process", in Copper and Nickel Converters, Ed. R. E. Johnson, The Metallurgical Society of AIME, New York, N.Y., 1979.

(7) M. Goto and T. Echigoya, "Refractory Practice and Application of Water Jackets in Mitsubishi Process", Paper presented at the 109th Annual Meeting of AIME, February 1980, Las Vegas, Nevada, USA.

(8) E. Kimura, "Lancing Mechanism in Mitsubishi Continuous Smelting Furnace", paper presented at the Symposium of 140th Committee (Physico-Chemical Properties of Substances at Metallurgical Temperature), Japan Society for the Promotion of Science, October 31, 1979, Tokyo (Japanese).

(9) "Evaluation of a Metallurgical Complex in British Columbia", A study by Wright Engineers Ltd. and H. N. Halvorson Consultants Ltd., March 1979, Province of British Columbia, Ministry of Economic Development.

(10) H. Uchida, N. Kikumoto and M. Hayashi, "Processing of Copper Smelting Gases at Naoshima Smelter", paper presented at the Symposium on Control of Particulate Emissions in the Primary Nonferrous Metals Industries, March 18-21, 1979, Monterey, California, USA, (EPA Organized).

(11) H. Kohno, K. Sugiyama and Y. Sugawawa, "Fugitive Gas Treatment at Onahama Smelter", paper to be presented at the Fourth Joint Meeting of MMIJ-AIME, November 4-8, 1980, Tokyo, Japan.

(12) T. Kawakita, T. Watanabe and H. Yonei, "Emission Control at Tamano Smelter", paper presented at the Fall Meeting of the Mining and Metallurgical Institute of Japan (MMIJ), October, 17-19, 1977, Fukuoka, Japan.

TABLE 1

MAJOR FEATURES OF NAOSHIMA CONTINUOUS PLANT

Design Capacity	4 000 Tonnes Cu/m
Concentrate Grade	27.5%
Furnace Design	
S-Furnace	
Diameter	8.25 m inside bricks
Height	3.31 m inside bricks
Bath Depth	1.09 m inside bricks
Slag Cleaning Furnace	
Diameter	7.00 m x 7.20 m inside bricks
Height	2.21 m x 7.20 m inside bricks
Bath depth	1.11 m x 7.20 m inside bricks
Transformer capacity	1 200 KVA
C-Furnace	
Diameter	6.65 m inside bricks
Height	2.94 m inside bricks
Bath depth	0.75 m inside bricks

TABLE 2

TYPICAL OPERATION DATA (JANUARY 1980)

S-Furnace	
Concentrate, tonnes	18 000
Silica flux, tonnes	3 200
Limestone flux, tonnes	140
Return C-Slag, tonnes	1 300
Lance blast air, Nm^3/Hr	22 400
Oxygen concentration, %	29.0
Fuel oil, kl	710
Pulverized coal, tonnes	460
C-Furnace	
Matte (calculated), tonnes	8 000
Limestone flux, tonnes	720
Lance blast air, Nm^3/Hr	10 000
Oxygen concentration, %	25.6
Fuel oil, kl	125
Steam recovered, tonnes	13 900
Anodes produced, tonnes	5 100

TABLE 3

TYPICAL ANALYSIS, %

	Cu	Fe	S	SiO_2	CaO	Al_2O_3
Concentrate	27.5	27.5	31.0	5.5	0.5	1.5
Silica flux	-	-	-	90.0	-	-
Limestone flux	-	-	-	-	53.0	-
Matte	65.0	11.0	22.0	-	-	-
Discard slag	0.5	42.0	0.7	30.2	4.2	3.3
C-Slag	15.0	44.0	< 0.1	< 0.2	15.0	< 0.2

	Cu	Pb	Ni	Bi	As	Sb
Anode	99.4	0.21	0.03	0.01	0.02	0.01

TABLE 4

ANODE PRODUCTION

	Concentrate Treated		Anode Produced, Tonnes
	Tonnes	Cu %	
October 1979	15 100	27.6	3 470
November	18 000	28.7	5 090
December	18 800	28.2	5 100
January 1980	18 100	27.5	5 150
February	16 500	27.3	4 470
March	16 400	26.9	4 770
April	14 400	26.5	3 490
May	18 100	27.0	4 750
June	17 500	27.2	4 650
July	17 100	25.7	4 360
August	17 600	24.9	4 040
continuing			
Average	17 050		4 480

TABLE 5

NUMBER OF LANCES

	Lance Diameter	S-Furnace Lances	C-Furnace Lances
Original Design	2 1/2 cm	10	6
Recent Practice	3 cm	6	4

TABLE 6

TYPICAL OPERATING CONDITION OF LANCES

Lance Diameter	3 inch
Lance Blast	3000 Nm^3/Hr/Lance
Solid Charge (Average)	7 MT/Hr

TABLE 7

TYPICAL OPERATION DATA WITH AND WITHOUT COAL UTILIZATION

		MODE OF OPERATION	
		Fuel Oil Only	Oil & Coal
Concentrate feed rate	MT/H	26.2	27.0
Fuel oil	l/H	1,370	850
Coal	kg/H	0	645
Equivalent fuel rate	l/H	1,370	1,300

TABLE 8

RESULTS OF HIGH OXYGEN ENRICHMENT TEST

	O_2% of S-Furnace Lance Blast	
	30%	50%
Throughput tonnes per day	500	500
Fuel rate		
Fuel Oil-l/H	1,100	400
Coal -kg/h	300	290
Off-gas Volume (S+C) Nm^3/min.	830	580

TABLE 9

ENERGY CONSUMPTION AT 100 000 TONNE PLANT

	Consumption Per Tonne Copper
Fuel Consumption, - including dryer, anode furnaces, launder heating.	73.3 l
Power Consumption,	
- Materials handling, compressors and blowers including fugitive gas treatment, excluding acid plant	308 KWH
- Production of tonnage oxygen	162 "
- Electric furnace	323 "
Total	793 KWH
Steam Recovered, 40 kg/cm^{2}G, saturated	1.50 MT

TABLE 10

SMELTING RATE

	MITSUBISHI CONTINUOUS (Model)	NAOSHIMA (Reverb)	TAMANO (Flash)	SAGANOSEKI (Flash)
Annual capacity, tonnes	100 000	96 000	94 000	110 000
Operating days per year	325*[1]	350*[1]	325*[1]	325*[1]
Furnace volume, M^3	670	1,520*[2]	860	960
Smelting rate T/M^3/D	0.46	0.18	0.34	0.35

Note:

(1) Assumption
(2) Roaster is included

TABLE 11

FUGITIVE GAS COLLECTION

	Naoshima Continuous Plant	Naoshima(10) Reverb (Calcive)	Ohahama(11) Reverb (Green)	Tamano(12) Flash
Annual Capacity, tonnes	48 000	96 000	146 000	94 000
Gas Neutralization, Nm^3/M				
Acid plant tail-gas	0	2,600	2,700	0
Anode f'ce off-gas	0*1	500	800	500*2
Reverb off-gas	-	0	2,000	-
Secondary hood, etc.	0	0	7,100	2,800
Total	0	3,100	12,600	3,300
Fugitive Gas Collect Nm^3/M*3	500	4,500	5,100	2,700
SO_2 Emission				
Allowed, $Nm^3 SO_2/H$	50	135	84	87
Equivalent emission, $Nm^3 SO_2/H/100,000T$ Cu	104	141	58	93

Note: 1) Anode furnace off gas during oxidation are mixed to burner air. In autogenous mode of operation, this part will be sent to neutralization plant or high stack depending on degree of allowable SO_2 emission.

2) Estimated figure.

3) These are additional to what are neutralized normally, filtered and vented off.

TABLE 12

MAN POWER REQUIREMENT (UNIT; MAN-SHIFT/DAY)

	Mitsubishi Continuous Process 100 000 T/Y		Naoshima Conventional Reverb 96 000 T/Y	
	Foreman	Operator	Foreman	Operator
Feed preparation	1	2 + 1 x 3	2	2 + 3 x 3
Smelting furnace	1 x 3	7 x 3	1 x 3	6 x 3
Converting furnace	1 x 3	7 x 3	1 x 3	7 x 3
Anode furnace	1 x 3	5 x 3	1 x 3	5 x 3
Converter slag treatment	-	-	1	4 + 3 x 3
Oxygen plant	-	1 x 3	-	-
Miscellaneous	1	5	1	11
Total	8	55	13	89

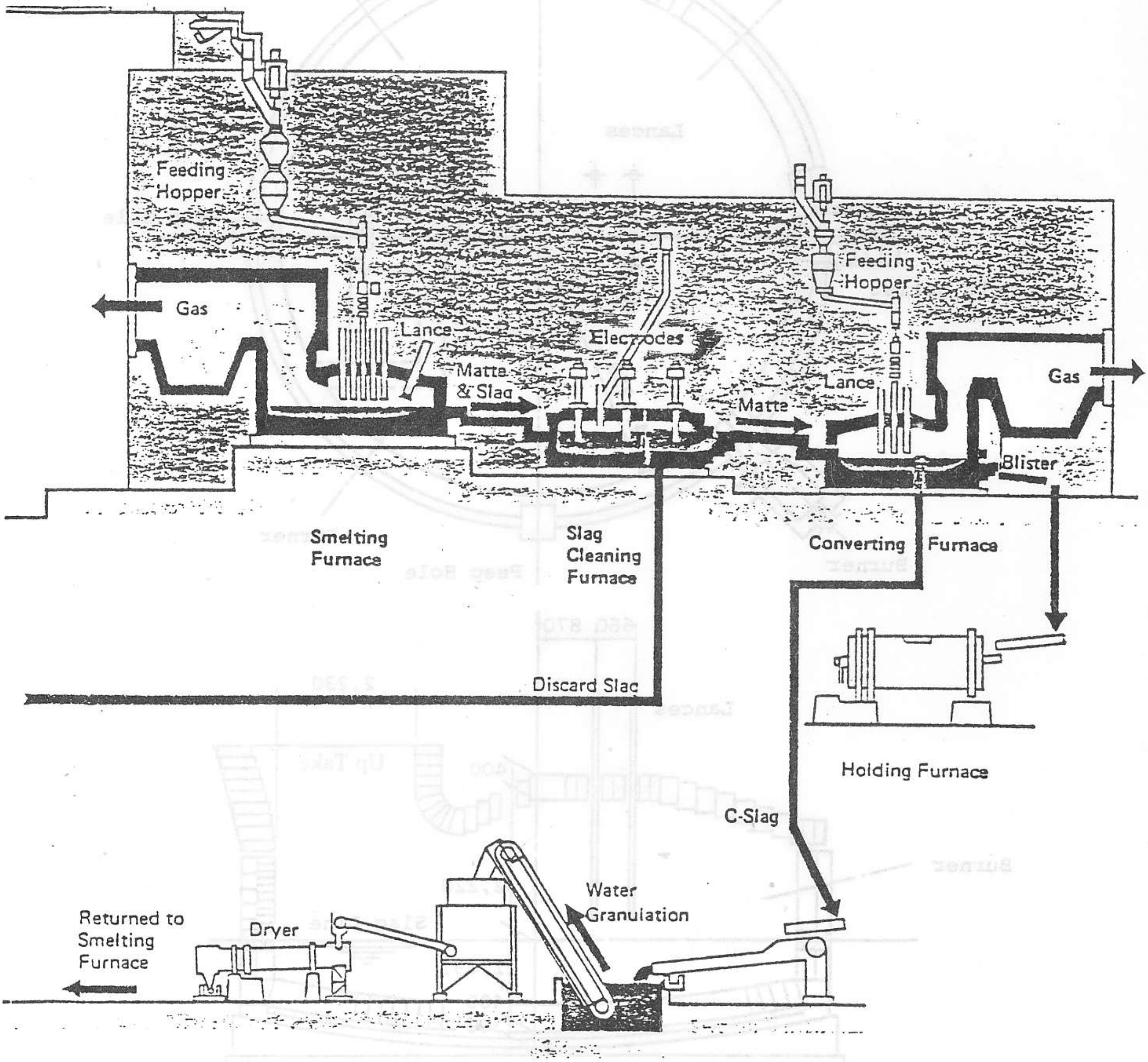


Figure 1. Schematic Flowsheet of Mitsubishi Process

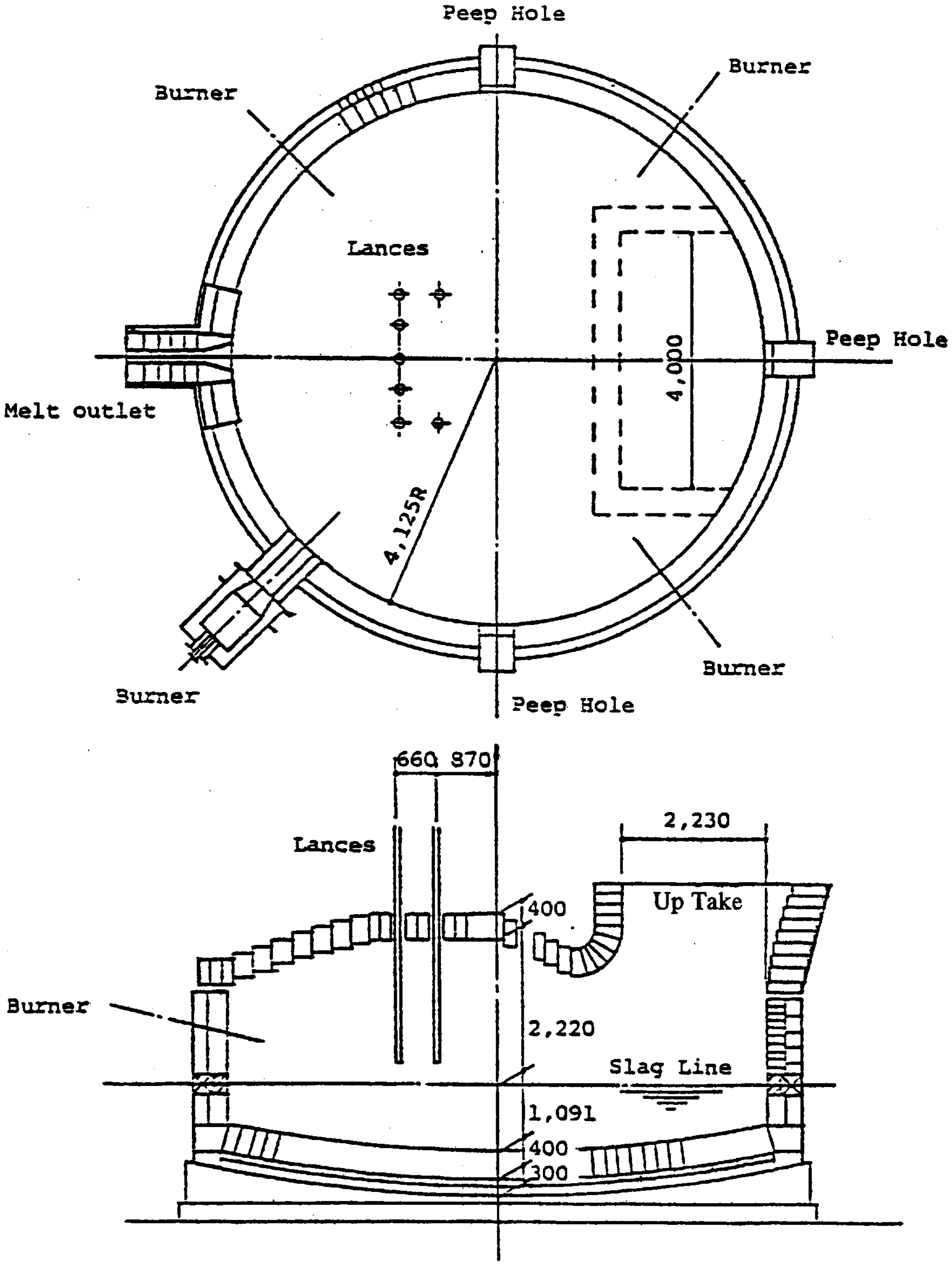


Figure 2. Smelting Furnace

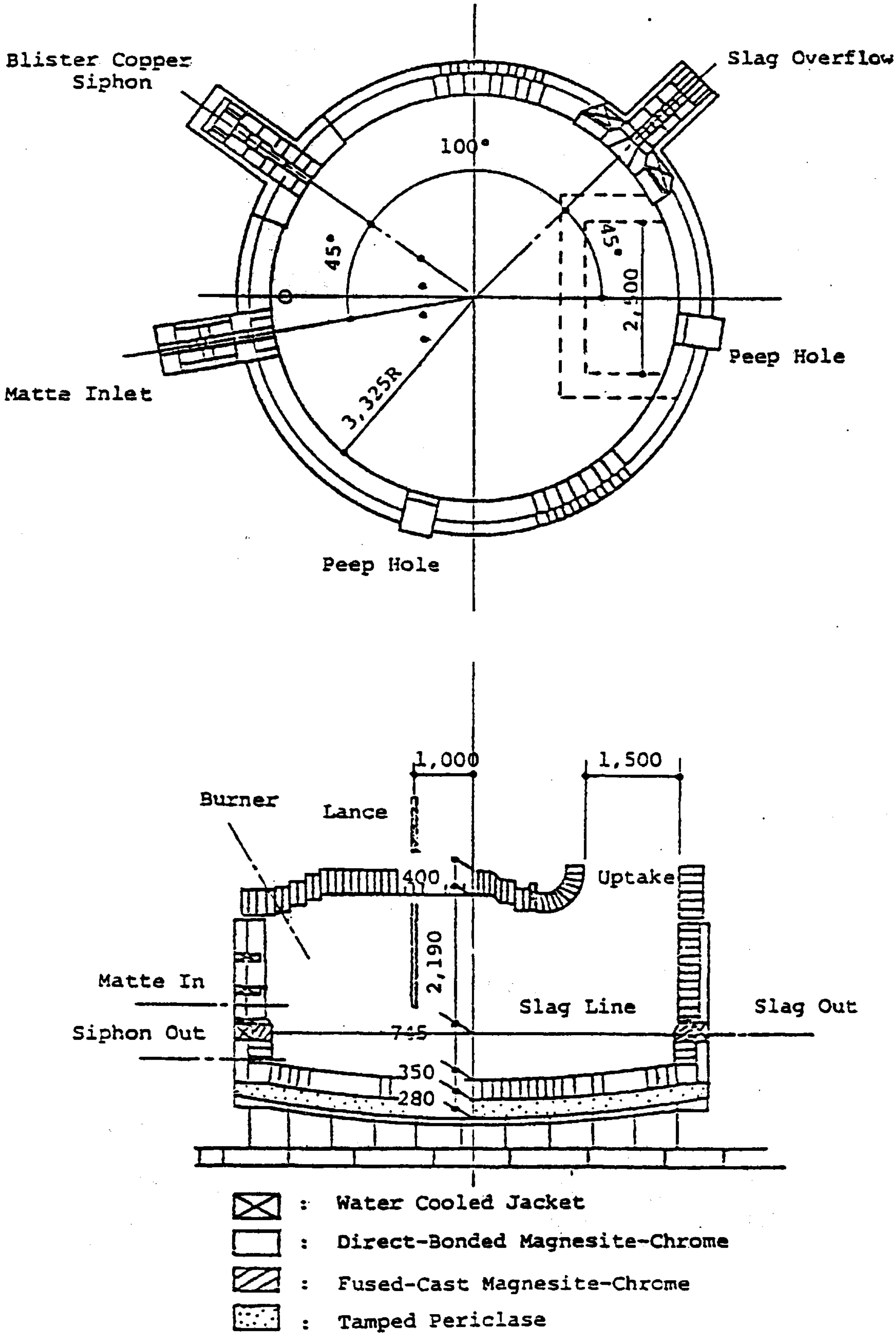


Figure 3. Converting Furnace

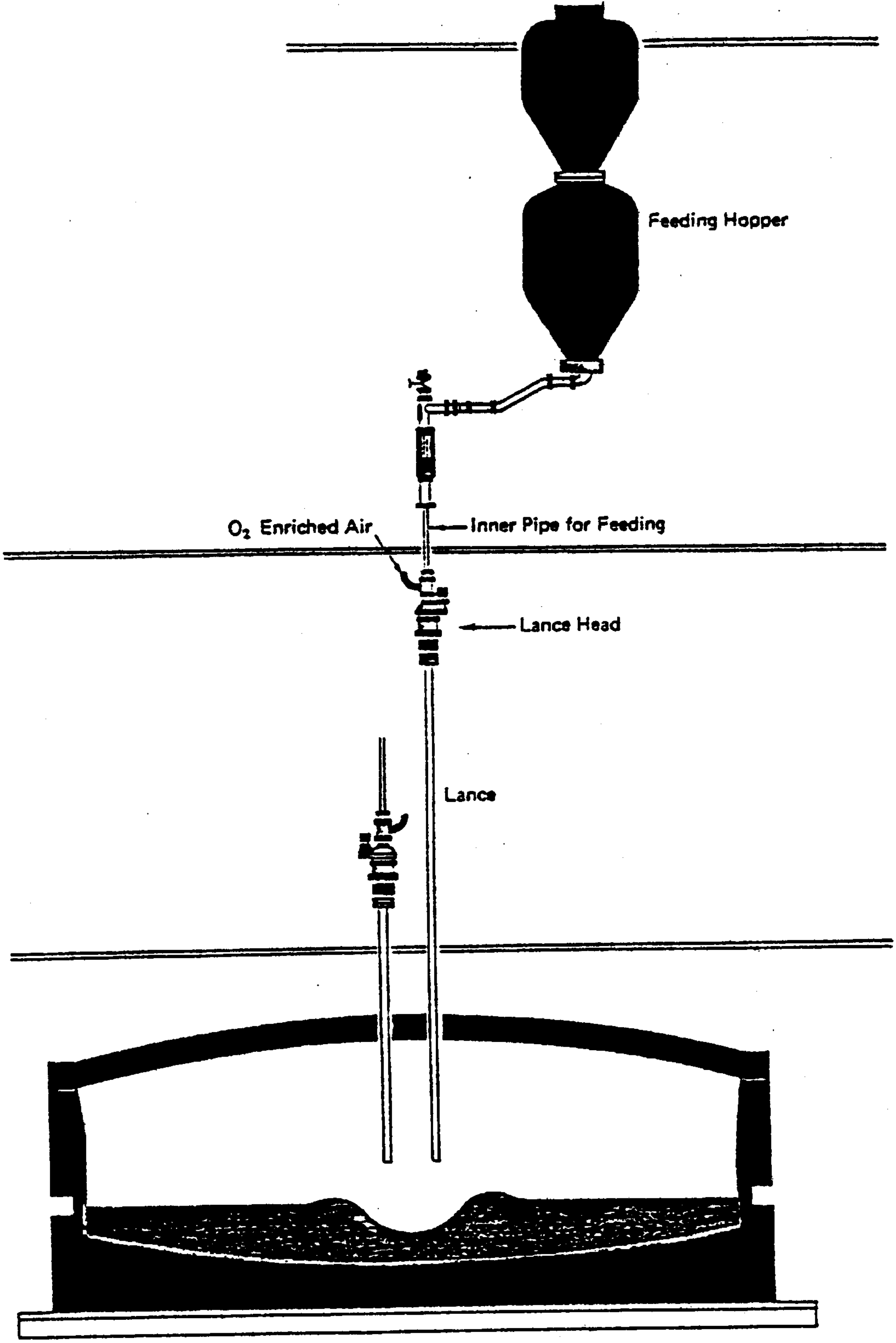
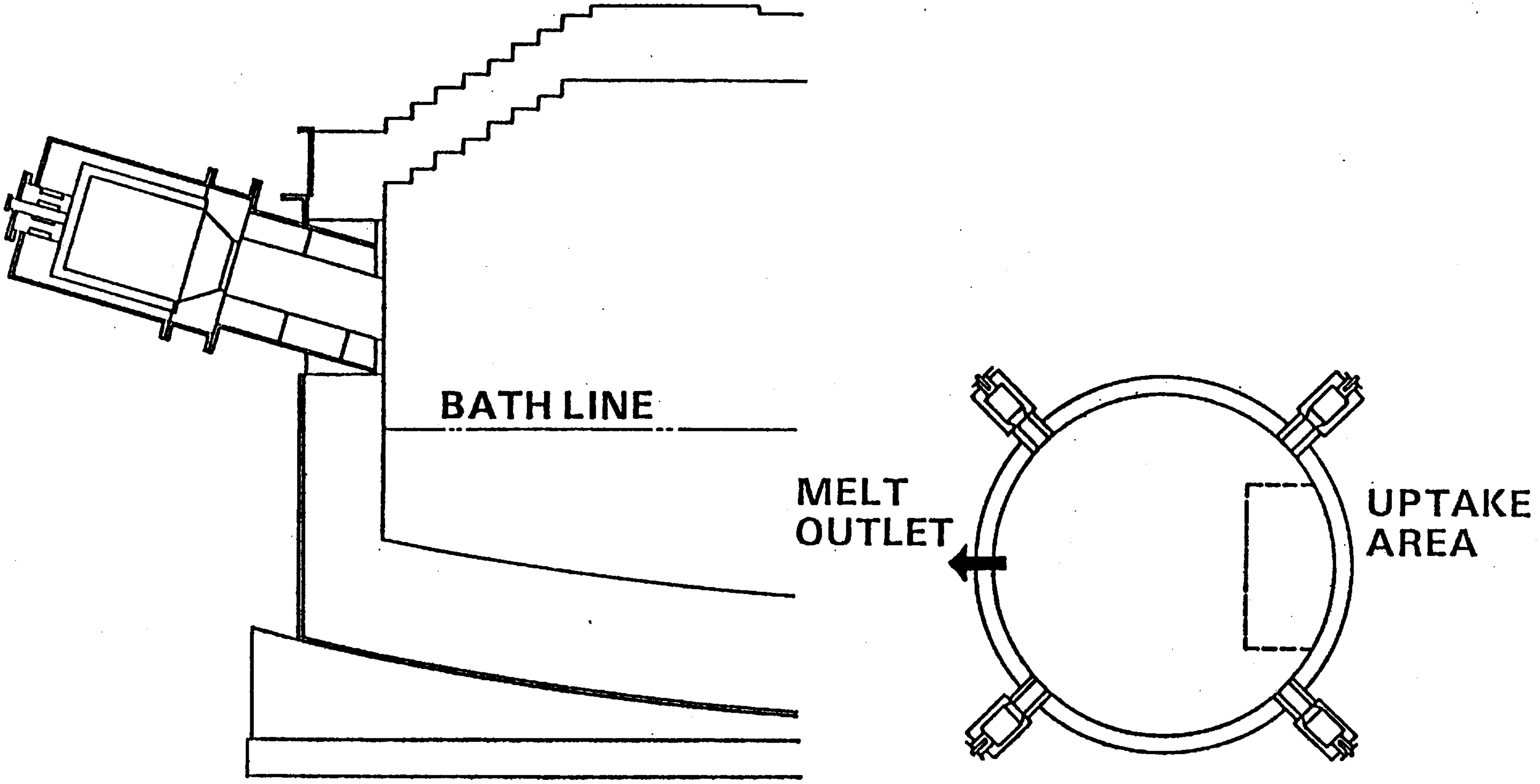


Figure 4. Schematic drawing showing feeding and lancing system

Figure 5.

MODIFIED BURNER ARRANGEMENT



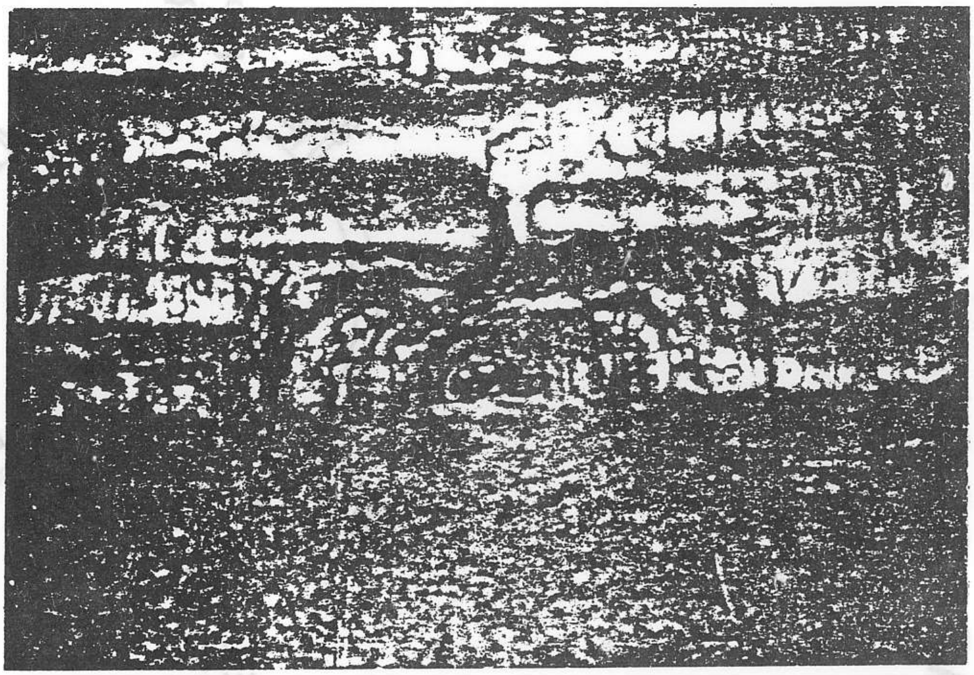


FIGURE 6. Brick condition around the slag/matte outlet after one year campaign, before modification.

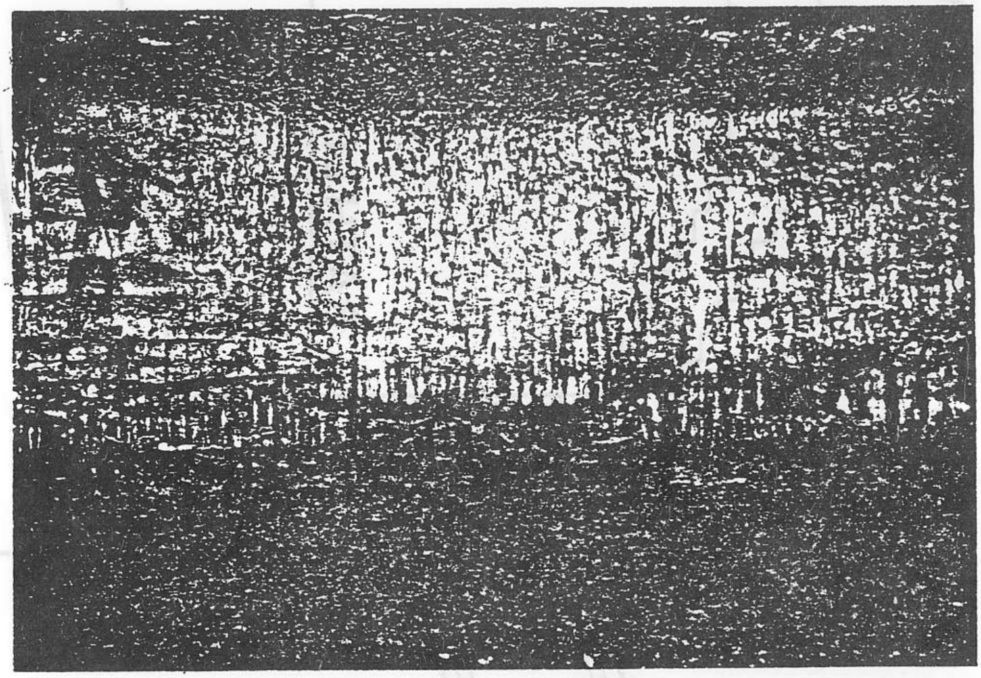
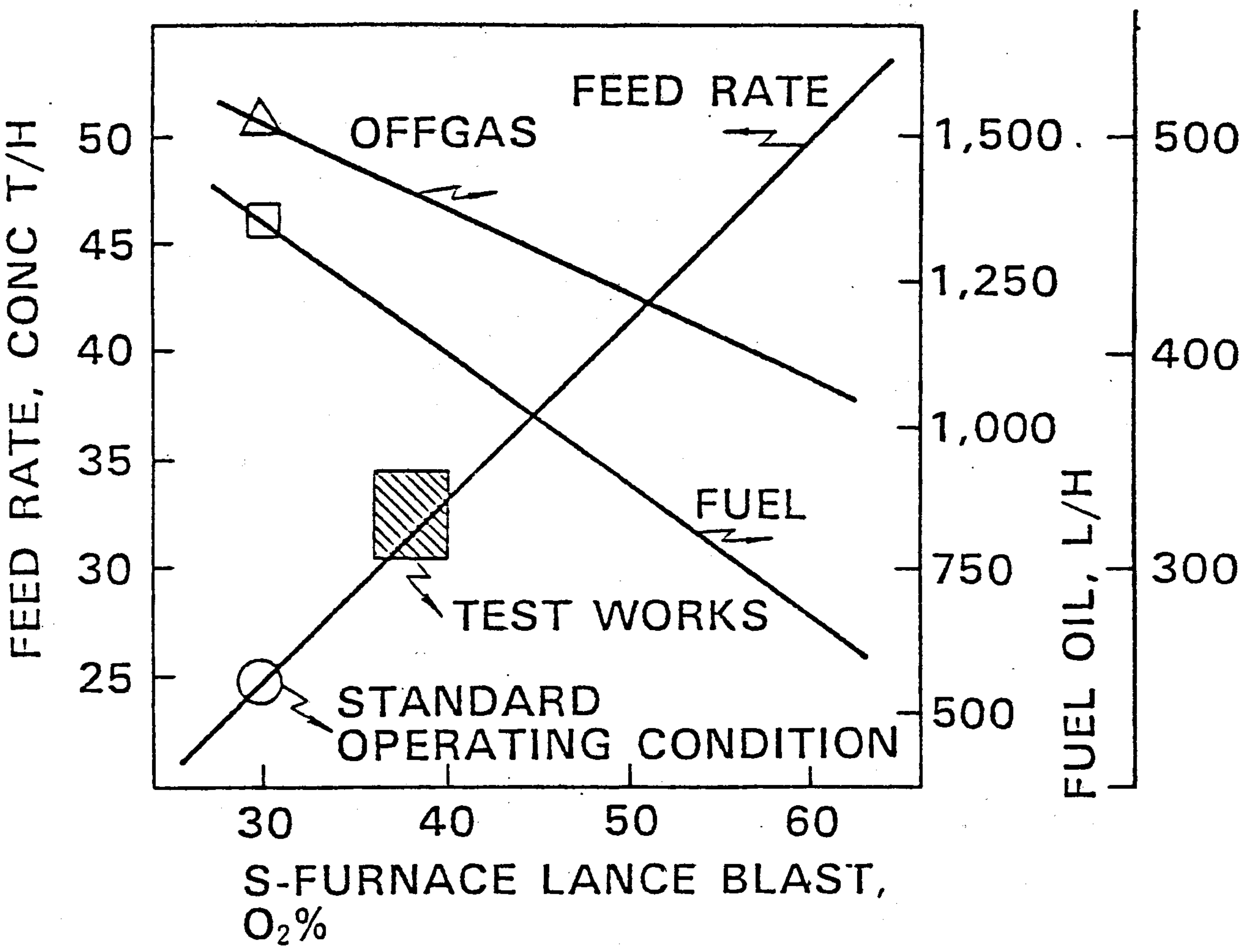


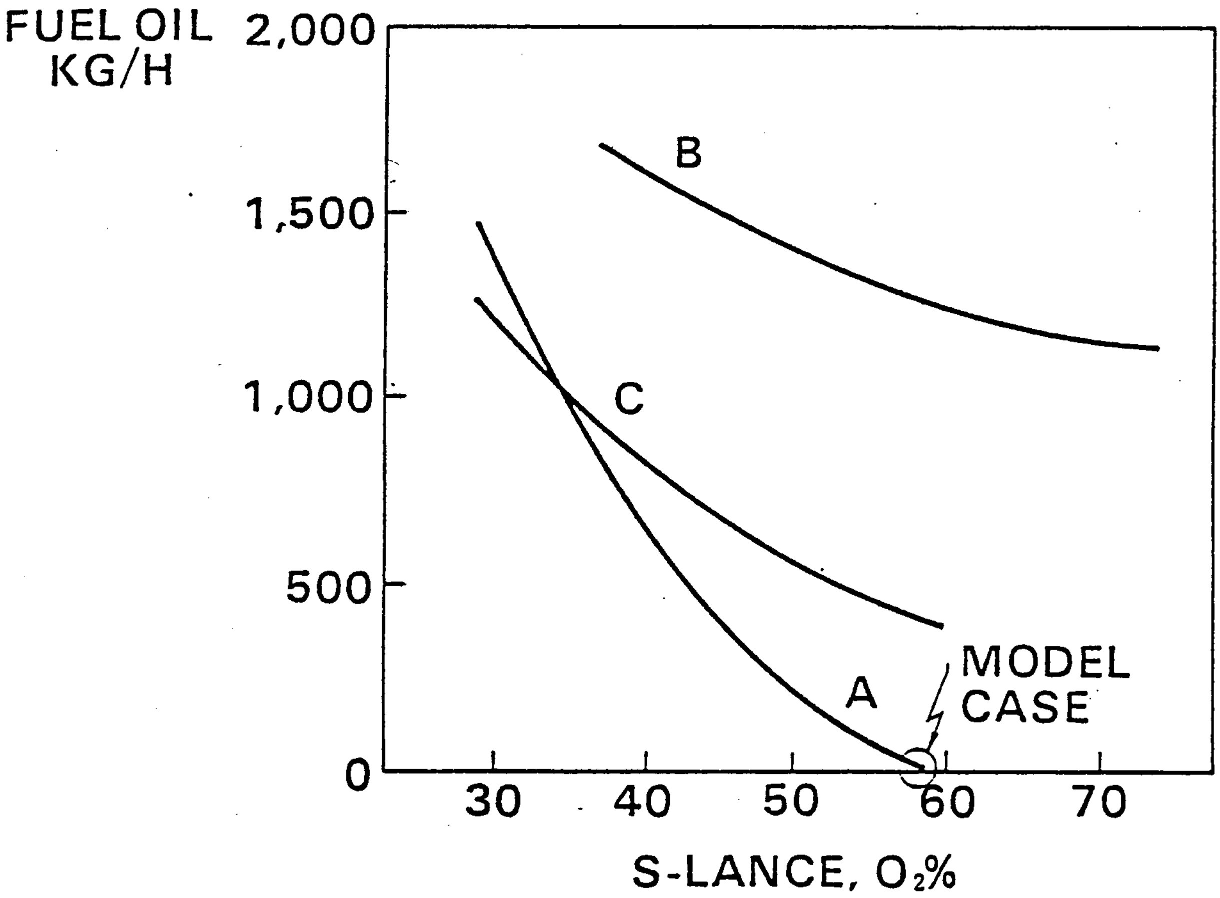
FIGURE 7. Brick condition around the slag/matte outlet after one year campaign, after burner modification.



Condition;

- Based on the existing Naoshima plant,
- S-Furnace lance blast being fixed at 15,000 Nm_3/H.
- Concentrate analysis being assumed as Cu 28%, Fe 27%, S 32%, SiO_2 5%.

Figure 8. Increase of throughput by higher oxygen enrichment.



Concentrate	Cu%	Fe%	S%	SiO %
Type A	28	28	35	5
Type B	35	25	27	8
Type C	Type B + 3.4% coal			

Figure 9. Fuel requirement at 100,000 MT annual capacity

APPENDIX

PROCESS CALCULATION OF MITSUBISHI CONTINUOUS COPPER SMELTING AND CONVERTING PROCESS

PROJECT NAME: COPPER SMELTING/REFINING TECHNOLOGIES
SEMINAR, VANCOUVER, B.C., NOVEMBER 1980

ANNUAL CAPACITY: 101 000 TONNES COPPER PER YEAR

NOVEMBER 5, 1980

MITSUBISHI METAL CORPORATION

APPENDIX 1

FEED MATERIALS AND FUEL

COMPOSITION OF FEED MATERIALS ($)

	CU	NI	CO	FE	ZN	CD	PB	AS	SB	BI	SI02	CAO
Concentrates	28.000	0.0	0.0	28.000	0.0	0.0	0.0	0.0	0.0	0.0	5.000	0.0
Other S-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Silica Flux	0.0	0.0	0.0	2.000	0.0	0.0	0.0	0.0	0.0	0.0	90.000	0.0
Limestone Flux	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.000	50.000

	MGU	AL203	GANGUE	TOTAL $	S/S03	C	H	C02	F.CL	FIXH20	IGLOSS	MOIST
Concentrates	0.0	1.000	0.0	35.000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.500
Other S-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Silica Flux	0.0	2.000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.500
Limestone Flux	1.000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.500

COMPOSITION OF FEED MATERIALS ($)
(REVISED)

	CU	NI	CO	FE	ZN	CD	PB	AS	SB	BI	SI02	CAO
Concentrates	28.000	0.0	0.0	28.000	0.0	0.0	0.0	0.0	0.0	0.0	5.000	0.0
Other S-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Silica Flux	0.0	0.0	0.0	2.000	0.0	0.0	0.0	0.0	0.0	0.0	90.000	0.0
Limestone Flux	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.000	50.000

	MGU	AL203	GANGUE	TOTAL $	S/S03	C	H	C02	F.CL	FIXH20	IGLOSS	MOIST
Concentrates	0.0	1.000	3.000	35.000	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.500
Other S-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Furnace Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Silica Flux	0.0	2.000	5.140	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.500
Limestone Flux	1.000	0.0	7.658	0.0	0.0	0.0	0.0	40.342	0.0	0.0	40.342	0.500

TYPE OF FUEL **BUNKER-C-OIL**

COMPOSITION OF FUEL

	S	C	H2	02	N2	ASHES
PHASE CODE 1	3.000	85.000	11.000	0.500	0.500	0.0

*PHASE CODE 1: Liquious or Solidus (WT $)
2: Gaseous (VOL $)

APPENDIX 2

MATERIALS TO BE TREATED (DRY BASE)

	M.T. PER YEAR	M.T. PER YEAR PER LINE	M.T. PER DAY PER LINE	M.T. PER HOUR PER LINE
S-INPUT				
Concentrate	365000.	365000.	1123.	52.00
Other S-Charge	0.	0.	0.	0.0
Silica Flux	66842.	66842.	206.	9.52
Limestone Flux	7151.	7151.	22.	1.02
Converter Slag	41658.	41658.	128.	5.93
S-Boiler Dust	3806.	3806.	12.	0.54
C-Boiler Dust	900.	900.	3.	0.13
S-EP Dust	5008.	5008.	15.	0.71
C-EP Dust	1502.	1502.	5.	0.21
TOTAL INPUT	491867.	491867.	1513.	70.07
S-OUTPUT				
Matte	167126.	167126.	514.	23.81
Discard Slag	245641.	245641.	756.	35.00
S-Boiler Dust	3850.	3850.	12.	0.55
S-EP Dust	5089.	5089.	16.	0.73
S-Off Dust	0.	0.	0.	0.0
TOTAL OUTPUT	421706.	421706.	1,298.	60.08
C-INPUT				
Matte	167126.	167126.	514.	23.81
Other C-Charge	0.	0.	0.	0.0
Limestone Flux	12504.	12504.	38.	1.78
TOTAL INPUT	179630.	179630.	553.	25.59
C-OUTPUT				
Blister Copper	101685.	101685.	313.	14.49
Converter Slag	41658.	41658.	128.	5.93
C-Boiler Dust	785.	785.	2.	0.11
C-EP Dust	1340.	1340.	4.	0.19
TOTAL OUTPUT	0.	0.	0.	0.0
C-OFF GAS	145468.	145468.	448.	20.72
BLEED OFF				
Discard Slag	245641.	245642.	756.	35.00
S-Boiler Dust	0.	0.	0.	0.0
S-EP Dust	0.	0.	0.	0.0
S-Off Gas	0.	0.	0.	0.0
TOTAL S-BLEED OFF	245641.	245641.	756.	35.00
C-Boiler Dust	0.	0.	0.	0.0
C-EP Dust	0.	0.	0.	0.0
C-Off Gas	0.	0.	0.	0.0
C-Slag	0.	0.	0.	0.0
TOTAL C-BLEED OFF	0.	0.	0.	0.0

APPENDIX 3
METAL BALANCE - UNIT: KG PER 100 KG CONC.

	MASS	CU 1	NI 2	CO 3	FE 4	ZN 5	CD 6	PB 7	AS 8	SB 9	B1 10	S102 11	CAO 12	MGO 13	AL203 14	GANG 15
S-INPUT																
Concentrates	100.00	20.00	0.0	0.0	28.00	0.0	0.0	0.0	0.0	0.0	0.0	5.00	0.00	0.00	1.00	3.00
Other Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.0	0.0
Silica Flux	18.31	0.0	0.0	0.0	0.37	0.0	0.0	0.0	0.0	0.0	0.0	16.48	0.00	0.00	0.37	0.94
Limestone	1.96	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.02	0.98	0.02	0.0	0.15
C-Slag	11.41	1.71	0.0	0.0	5.37	0.0	0.0	0.0	0.0	0.0	0.0	0.05	1.71	0.03	0.00	0.26
S-Boiler Dust	1.04	0.66	0.0	0.0	0.14	0.0	0.0	0.0	0.0	0.0	0.0	0.02	0.000	0.00	0.00	0.00
C-Boiler Dust	0.25	0.18	0.0	0.0	0.02	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00
S-EP Dust	1.37	0.69	0.0	0.0	0.34	0.0	0.0	0.0	0.0	0.0	0.0	0.02	0.00	0.00	0.00	0.00
C-EP Dust	0.41	0.21	0.0	0.0	0.11	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00
TOTAL INPUT	134.76	31.45	0.0	0.0	34.35	0.0	0.0	0.0	0.0	0.0	0.0	21.59	2.69	0.05	1.37	4.37
S OUTPUT																
Matte	45.79	29.76	0.0	0.0	5.50	0.0	0.0	0.0	0.0	0.0	0.0	0.01	0.00	0.00	0.00	0.00
Slag	67.30	0.34	0.0	0.0	28.37	0.0	0.0	0.0	0.0	0.0	0.0	21.54	2.69	0.05	1.37	4.35
S-Boiler Dust	1.05	0.66	0.0	0.0	0.14	0.0	0.0	0.0	0.0	0.0	0.0	0.02	0.00	0.00	0.00	0.00
S-EP Dust	1.39	0.69	0.0	0.0	0.34	0.0	0.0	0.0	0.0	0.0	0.0	0.02	0.00	0.00	0.00	0.00
S-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL OUTPUT	115.54	31.45	0.0	0.0	34.35	0.0	0.0	0.0	0.0	0.0	0.0	21.59	2.69	0.05	1.37	4.37
C INPUT																
Matte	45.79	29.76	0.0	0.0	5.50	0.0	0.0	0.0	0.0	0.0	0.0	0.01	0.00	0.00	0.00	0.00
Other Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00
Limestone	3.43	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.03	0.0	0.0	0.0	0.26
TOTAL INPUT	49.21	29.76	0.0	0.0	5.50	0.0	0.0	0.0	0.0	0.0	0.0	0.05	1.71	0.03	0.00	0.27
C OUTPUT																
Blister	27.86	27.66	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00
C-Slag	11.41	1.71	0.0	0.0	5.37	0.0	0.0	0.0	0.0	0.0	0.0	0.05	1.71	0.03	0.00	0.26
C-Boiler Dust	0.22	0.18	0.0	0.0	0.02	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00
C-EP Dust	0.37	0.21	0.0	0.0	5.11	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00
C-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00
TOTAL OUTPUT	39.85	29.76	0.0	0.0	5.50	0.0	0.0	0.0	0.0	0.0	0.0	0.05	1.71	0.03	0.00	0.27
BLEED-OFF																
S-Boiler Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S-EP Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL S-BLEED	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Boiler Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-EP Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Slag	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL C-BLEED	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0				

APPENDIX 4

COMPOSITION OF MATERIALS AND PRODUCTS (%)

	MASS	CU 1	NI 2	CO 3	FE 4	ZN 5	CD 6	PB 7	AS 8	SB 9	B1 10	S102 11	CAO 12	MGO 13	AL203 14	GANG 15	S 16
S-INPUT																	
Concentrates	100.00	28.00	0.0	0.0	28.00	0.0	0.0	0.0	0.0	0.0	0.0	5.00	0.00	0.00	1.00	3.00	35.00
Other Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.0	0.0	0.0
Silica Flux	18.31	0.0	0.0	0.0	2.00	0.0	0.0	0.0	0.0	0.0	0.0	90.00	0.00	0.00	2.00	5.14	0.0
Limestone	1.96	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.00	50.00	1.00	0.0	7.66	0.0
C-Slag	11.41	15.00	0.0	0.0	47.07	0.0	0.0	0.0	0.0	0.0	0.0	0.43	15.00	0.30	0.01	2.32	0.0
S-Boiler Dust	1.04	63.35	0.0	0.0	13.18	0.0	0.0	0.0	0.0	0.0	0.0	2.07	0.03	0.00	0.13	0.42	7.18
C-Boiler Dust	0.25	72.39	0.0	0.0	8.93	0.0	0.0	0.0	0.0	0.0	0.0	0.02	0.49	0.01	0.00	0.11	2.19
S-EP Dust	1.37	50.43	0.0	0.0	25.04	0.0	0.0	0.0	0.0	0.0	0.0	1.57	0.02	0.00	0.10	0.32	2.80
C-EP Dust	0.41	50.63	0.0	0.0	26.76	0.0	0.0	0.0	0.0	0.0	0.0	0.01	0.25	0.01	0.00	0.06	1.79
TOTAL INPUT	134.76	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.0	0.00	0.00	0.00	0.00
S OUTPUT																	
Matte	45.79	65.00	0.0	0.00	12.02	0.0	0.0	0.0	0.0	0.0	0.0	0.03	0.00	0.00	0.00	0.01	22.24
Slag	67.30	0.50	0.0	0.0	42.15	0.0	0.0	0.0	0.0	0.0	0.0	32.00	4.00	0.08	2.03	6.47	0.13
S-Boiler Dust	1.05	62.62	0.0	0.00	13.03	0.0	0.0	0.0	0.0	0.0	0.0	2.05	0.03	0.00	0.13	0.41	20.04
S-EP Dust	1.39	49.62	0.0	0.00	24.64	0.0	0.0	0.0	0.0	0.0	0.0	1.55	0.02	0.00	0.10	0.31	20.57
S-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL OUTPUT	115.54	0.0	0.0	0.00	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00	0.00
C INPUT																	
Matte	45.79	65.00	0.0	0.0	12.02	0.0	0.0	0.0	0.0	0.0	0.0	0.03	0.00	0.00	0.00	0.01	22.24
Other Charge	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00
Limestone	3.43	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.00	50.00	1.00	0.0	7.66	0.0
TOTAL INPUT	49.21	0.00	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.00	0.00
C OUTPUT																	
Blister	27.86	99.30	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.00	0.0	0.00	0.00	0.0	0.50
C-Slag	11.41	15.00	0.0	0.0	47.07	0.0	0.0	0.0	0.0	0.0	0.0	0.43	15.00	0.30	0.01	2.32	0.0
C-Boiler Dust	0.22	83.04	0.0	0.0	10.24	0.0	0.0	0.0	0.0	0.0	0.0	0.02	0.56	0.01	0.00	0.12	0.0
C-EP Dust	0.37	56.75	0.0	0.0	29.99	0.0	0.0	0.0	0.0	0.0	0.0	0.01	0.28	0.01	0.00	0.07	0.0
C-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL OUTPUT	39.85	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
BLEED-OFF																	
S-Boiler Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S-EP Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL S-BLEED	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Boiler Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-EP Dust	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Off Gas	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
C-Slag	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL C-BLEED	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

APPENDIX 5

LANCE BLAST AND BURNER AIR , NM3/NR

		S-FURNACE		C-FURNACE		TOTAL
		Amount	02 PCT	Amount	02 PCT	Amount
Lance Blast	1	19,115.0	57.6	17,113.8	29.8	36,229.0
Air	2	9,610.0	20.6	15,002.1	20.6	24,612.4
Tonnage Oxygen	3	9,505.0	95.0	2,111.7	95.0	11,616.6
Fuel Requirement	4	0.0	0.0	0.0	0.0	0.0
Burner Air	5	0.0	25.0	0.0	25.0	0.0
Air	6	0.0	20.6	0.0	20.6	0.0
Tonnage Oxygen	7	0.0	95.0	0.0	95.0	0.0

TOTAL TONNAGE OXYGEN	11,616.6/ NM 3/HR
LANCE DIAMETER	3.0/IN.
MAXIMUM LANCE BLAST	3,000.0/ NM 3/HR
NUMBER OF LANCES, S-Furnace	7.0
C-Furnace	6.0

APPENDIX 6

GAS VOLUME AND COMPOSITION (WET BASIS)

Gas Components			S FURNACE OFF GAS					
			Furnace Off-Gas	Boiler Inlet	Boiler Outlet (Normal)	EP Outlet (Normal)	Boiler Outlet (Max)	EP Outlet (Max)
SO2	PCT	1	50.45	48.05	44.65	42.52	37.89	36.08
O2	PCT	2	0.00	0.98	1.45	2.36	4.35	5.12
CO2	PCT	3	1.19	1.13	1.04	0.99	0.88	0.84
H2O	PCT	4	3.32	3.26	3.16	3.11	2.99	2.94
S2	PCT	5	0.00	0.00	0.00	0.00	0.00	0.00
H2	PCT	6	0.00	0.00	0.00	0.00	0.00	0.00
CO	PCT	7	0.00	0.00	0.00	0.00	0.00	0.00
N2	PCT	8	45.04	46.58	49.70	51.02	53.90	55.02
TOTAL	PCT	9	100.00	100.00	100.00	100.00	100.00	100.00
TOTAL	NM3/HR	10	17574.33	18453.05	20170.78	21179.31	23770.77	24959.31

Gas Components			C FURNACE OFF GAS					
			Furnace Off-Gas	Boiler Inlet	Boiler Outlet (Normal)	EP Outlet (Normal)	Boiler Outlet (Max)	EP Outlet (Max)
SO2	PCT	1	22.10	21.05	19.13	18.22	16.13	15.36
O2	PCT	2	3.95	4.74	6.09	6.78	8.37	8.95
CO2	PCT	3	2.18	2.08	1.89	1.80	1.59	1.52
H2O	PCT	4	1.86	1.86	1.88	1.88	1.90	1.90
S2	PCT	5	0.00	0.00	0.00	0.00	0.00	0.00
H2	PCT	6	0.00	0.00	0.00	0.00	0.00	0.00
CO	PCT	7	0.00	0.00	0.00	0.00	0.00	0.00
N2	PCT	8	69.91	70.27	71.00	71.31	72.01	72.27
TOTAL	PCT	9	100.00	100.00	100.00	100.00	100.00	100.00
TOTAL	NM3/HR	10	16764.55	17602.77	19340.01	20307.01	22940.01	24087.01

Gas Compnents			FURNACE OFF GAS TOTAL (S+C)	
			EP Outlet (Normal)	EP Outlet (Max)
SO2	PCT	1	30.63	28.07
O2	PCT	2	4.53	5.87
COS	PCT	3	1.39	1.27
H2O	PCT	4	2.51	2.47
S2	PCT	5	0.0	0.0
H2	PCT	6	0.0	0.0
CO	PCT	7	0.0	0.0
N2	PCT	8	60.95	62.33
TOTAL	PCT	9	100.00	100.00
TOTAL	NM3/HR	10	41486.32	45266.32

FURNACE OFF GAS TEMPERATURE	
S-FCE	1250.°C
C-FCE	1250.°C

APPENDIX 7

S-FURNACE HEAT BALANCE

	UNIT AMOUNT KG(NM/3) /100KG	UNIT AMOUNT KCAL/ KG(NM/3)	HEAT VALUE KCAL/ 100KG	HEAT VALUE MCAL/ HR
INPUT				
Metallurg Reactions	0.0	0.0	60701.4	31564.7
Solid Feed	134.758	4.0	539.0	280.3
Molten Feed	0.0	0.0	0.0	0.0
Fuel Combustion	0.0	10194.2	0.0	0.0
TOTAL INPUT	0.0	0.0	61240.5	31845.0
OUTPUT				
Matte/Blister	45.788	216.0	9890.2	5142.9
Slag/C-Slag	67.299	319.0	21468.4	11163.5
Dusts	2.449	500.0	1224.5	636.0
Off Gases				
SO2	17.051	685.0	11680.0	6073.0
O2	-0.000	451.2	0.0	0.0
CU2	0.402	683.7	275.0	143.0
H20	1.121	533.7	598.5	311.2
S2	0.0	451.2	0.0	0.0
H2	0.0	402.5	0.0	0.0
CO	0.0	431.2	0.0	0.0
N2	15.222	426.2	6488.4	3374.0
TOTAL OFF GASES	33.797	0.0	19042.0	9901.8
HEAT LOSSES	0.0	0.0	9615.4	5000.0
TOTAL OUTPUT	0.0	0.0	61240.5	31845.0
BALANCE	0.0	0.0	0.0	0.0

APPENDIX 8

C-FURNACE HEAT BALANCE

	UNIT AMOUNT KG(NM/3) /100KG	UNIT AMOUNT KCAL/ KG(NM/3)	HEAT VALUE KCAL/ 100KG	HEAT VALUE MCAL/ HR
INPUT				
Metallurg Reactions	0.0	0.0	22676.4	11791.7
Solid Feed	3.426	4.0	13.7	7.1
Molten Feed	45.788	196.0	8974.5	4666.7
Fuel Combustion	0.0	10194.2	0.0	0.0
TOTAL INPUT	0.0	0.0	31664.6	16465.6
OUTPUT				
Matte/Blister	27.859	182.0	5070.3	2636.6
Slag/C-Slag	11.413	325.0	3709.3	1928.8
Dusts	0.5823	500.0	291.1	151.4
Off Gases				
S02	7.126	685.0	4881.0	2538.1
02	1.273	451.2	574.6	298.8
CU2	0.703	683.7	481.0	250.1
H20	0.598	533.7	319.4	166.1
S2	0.0	451.2	0.0	0.0
H2	0.0	402.5	0.0	0.0
CO	0.0	431.2	0.0	0.0
N2	22.539	426.2	9607.2	4995.7
TOTAL OFF GASES	32.240	0.0	15863.1	8248.8
HEAT LOSSES	0.0	0.0	6730.8	3500.0
TOTAL OUTPUT	0.0	0.0	31664.6	16465.6
BALANCE	0.0	0.0	0.0	0.0

SECTION V

THE OUTOKUMPU FLASH SMELTING METHOD

BY K. MURDEN
J. SULTANO

THE OUTOKUMPU COMPANY

Outokumpu is the largest mining and metallurgical company in Finland. With an annual turnover of about 600 million dollars in 1979 the company ranks among the country's top industrial enterprises.

Two major shareholders are the Finnish Government and the Social Insurance Institute of Finland. Employees at twenty-one individual units throughout Finland total nearly 9,500.

Activities first began when the Outokumpu copper ore body in eastern Finland was discovered in 1910. Today Outokumpu operates nine mines, four metallurgical plants, the Technical Export Division, a Metallurgical Research Institute, an Institute of Physics and an Exploration Department.

The main metal products are copper and copper alloys, stainless steel, zinc, cobalt, nickel and ferrochrome.

OUTOKUMPU FLASH SMELTING PROCESS

The basis of the Outokumpu Flash Smelting Process for sulfidic ores is the replacement of the conventional methods separate roasting and smelting by a combined roasting, smelting and part-converting process carried out in one unit. The heat generated by the exothermic reactions can thus be used for smelting the ore, so that only a small amount of additional energy is needed. The recovery of waste heat from the furnace off-gases further improves the energy efficiency of the process. Smelting of dried, fine-grained concentrate in suspension, which offers the best conditions for rapid reactions, is a characteristic feature of the process.

HISTORY OF THE OUTOKUMPU FLASH SMELTING PROCESS

The shortage of electrical energy in Finland in World War II caused Outokumpu to consider replacing the electric furnace smelting of copper concentrates by another method. After a careful study Outokumpu decided to develop the flash smelting process, which gives a considerable saving on fuel and yields a high recovery of sulfur. The pilot plant testing of the new process was carried out in 1947-1948.

The process was adopted for the smelting of copper concentrates on a commercial scale in 1949 and applied successfully to the smelting of nickel concentrates at the Harjavalta Works in 1959.

In 1962 Outokumpu adopted a variant of the process, in which elemental sulfur is recovered from the gases after thermal decomposition of pyrite in the flash smelting furnace. This process and the normal flash smelting process can be combined to produce elemental sulfur from smelter gases.

In 1971 oxygen enrichment in flash smelting was adopted in the Harjavalta copper and nickel smelters which gave considerable improvement in capacity and energy efficiency.

A commercial scale flash smelter producing blister copper in flash smelting furnace was started in 1978 in Poland.

PROCESS DESCRIPTION

The main process steps at a modern flash smelter are:

(1) Feed preparation and drying.
(2) Flash smelting and gas treatment.
(3) Converting.
(4) Anode casting.
(5) Slag cleaning.

FEED PREPARATION AND DRYING

When the feed to the smelter consists of several concentrates, these should be mixed to make the feed as homogenous as possible. The homogeneity of the feed makes the flash smelting process easier to control. Fluxing agents are normally mixed with the concentrate before drying.

Fine grained feed has to be dried before going to the flash smelting furnace in order to ensure sufficiently rapid ignition and oxidation of the sulfides. There are several drying methods, the most common of them being co-current rotary drying. Heat needed for drying is provided by burning fossil fuels. Hot gases from the anode furnace or other sources can also be used for drying.

FLASH SMELTING

Dried feed and flue dust obtained by cleaning the furnace gas are fed together with the process air at a controlled rate into the concentrate burner at the top of the vertical, cylindrical reaction shaft.

Process air is either oxygen-enriched air preheated up to 250° or ambient air preheated up to 400°-1000°C. Oxygen-enriched air is preheated with steam, whereas high preheating calls for an oil or gas fired preheater.

The concentrate oxidizes in the reaction shaft in the concentrate-air suspension and flows towards the horizontal settler. Heat generated by the exothermic oxidation reactions smelts the solid particles. Molten material falls to the bottom of the settler, where it separates into matte and slag layers.

Part of the feed flows with process gases as flue dust and is circulated from the gas line back to the flash smelting furnace feed. Impurities which have a high vapour pressure will concentrate in the flue dust and can be removed from the process and recovered by handling the dust in a separate process.

CONVERTING AND ANODE CASTING

Flash smelting furnace matte is taken to the converter to be blown with air or with oxygen-enriched air, thus producing blister copper. Converter slag is transported to the slag cleaning process or poured back to the flash smelting furnace. The converter gases are conducted after cleaning to the sulfuric acid plant.

Sometimes blister copper is the end product of a smelter, but usually it is cleaned in the anode furnace and cast into anodes. Anode furnace gases can be used in a concentrate dryer.

SLAG CLEANING

The slags from the flash smelting furnace and the converter contain so much copper and other valuable metals that their recovery is economically worthwhile. Recovery is carried out either by cleaning slag in an electric furnace or by flotation.

In the electric furnace cleaning slag is tapped either continuously or batchwise into the electric furnace to be reduced with coke. This reduces the solubility of copper in slag, and the copper thus settles as matte to the bottom of the furnace to be taken to the converter.

When using the flotation process the slags are cooled slowly, crushed, ground and flotated. Flotation concentrate is fed through the dryer back to the flash smelting furnace.

GAS TREATMENT

Gas from the flash smelting furnace flows through the uptake shaft into the waste heat boiler, where it cools to about 350°C. Waste heat is recovered as steam. Gas from the boiler is conducted through electrostatic precipitators to the sulfuric acid plant. Flue dust is recovered in the boiler and in the electrostatic precipitator and fed back into the flash smelting furnace.

REQUIREMENTS FOR A MODERN SMELTER INSTALLATION

To make a profitable smelter investment, the process selected must meet several, partly contradictory, requirements; the most important of these are reliability, flexibility, low investment and operating costs, free from pollution and occupational safety and health hazards.

RELIABILITY

The process must use proven techniques and be completely understood, so that the equipment can be designed and dimensioned to meet the requirements of the process. Over 30 years' experience in flash smelting, together with intensive research and development work at Outokumpu's plants and Research Centre, guarantees the reliability of the modern Outokumpu Flash Smelting Process. The theoretical basis of the process is well-known, thus a reliable plant can be planned for different kinds of conditions and a wide variety of raw materials.

Outokumpu has developed an effective training system for customers' staff to ensure proper operation of licensed flash smelting processes. The best evidence of the reliability is that Outokumpu has sold twenty-six copper flash smelting licences all over the world.

INVESTMENT COSTS

The requirement of low investment costs conflicts with many of the other demands made of the process, for example with reliability and environmental protection. A compromise has to be made on the different requirements and this always depends on local conditions. In comparison with other smelters, an Outokumpu-type flash smelter is fully competitive in terms of investment costs.

The flash smelting furnace can be designed for high feed rates especially when high oxygen enrichment in process air or pure oxygen is used for oxidation. Some flash smelters have a production capacity of 200 000 tonnes per year of copper in one flash smelting furnace and the limit has been put by other factors than smelter capacity like availability of concentrates, marketing of products etc. The high unit capacity in flash smelting decreases the investment cost in relation to the produced copper amount.

When using oxygen-enriched process air, especially in flash smelters of higher capacity, investment costs are as much as 10% lower than in a flash smelter with the same capacity using preheated process air; at the same time other requirements for the smelter are more readily met.

FLEXIBILITY

The flexibility of a smelter is of the utmost importance in maintaining the planned production and its quality, even with raw materials of varying quality. Maximum production can be achieved by stabilizing the matte grade in the smelting furnace at the optimum level; in other words, the degree of oxidation of the concentrate must be regulated according to the varying composition of the concentrate, so that fluctuations do not disturb process stages after smelting.

The flexibility of the flash smelting process is based on the fact that in suspension smelting the degree of oxidation can be rapidly and easily regulated by changing the ratio of concentrate to oxygen in the process air. The only requirement is that the quality of the feed mixture is known in advance so that the process parameters can be calculated.

Effective elimination of impurities at the various process stages is a prerequisite for maintaining a product of uniform quality. Flash smelting makes it possible to remove most of the impurities in the concentrate into the slag or flue dust in the smelting furnace by setting the temperature and the degree of oxidation of the reaction zone to a suitable level. Temperature regulation is especially easy when a smelter uses oxygen-enriched process air.

RECOVERY OF VALUABLE METALS

Recovery of valuable metals from the raw materials greatly affects the profitability of a smelter.

At an Outokumpu-type flash smelter the losses due to material handling are minimized because dry concentrates and dusts are conveyed in completely closed systems and process gases from the smelter are efficiently cleaned by electrostatic precipitators.

Losses of valuable metals in the waste slag are minimized by cleaning the slags from the flash smelting furnace and the converter either in an electric furnace or in a slag concentrator. The copper content of the waste slag at a copper flash smelter varies from 0.3 to 0.6%, depending mainly on the cleaning method.

Due to high matte grade the usage of fluxes can be minimized in flash smelter which despite saving energy and material also minimizes the waste slag amount and thus increases the copper recovery.

OPERATING COSTS

The operating costs of a smelter, i.e. labour, energy and material costs, play a decisive role in profitability. At an Outokumpu-type flash smelter they are low compared to other types of smelters.

To reduce labour costs, a flash smelter can be highly automated. In the flash smelting furnace area, process staff are needed only for controlling and tapping purposes. The consumption of maintenance and other materials is low. The consumption of brick-lining materials, for example, is only about 1.0-1.5 kg per tonne of copper for the entire smelter. Refractory consumption in flash smelting furnace is low due to a protective cooling system in the reaction shaft and settler. Converter throughput during campaign is high because of high matte grade obtained in the flash smelting furnace.

ENERGY REQUIREMENTS

The greatest influence on smelting techniques has been the decreasing use of hydrocarbon fuels to stem the rise in operating costs. External energy requirements can be decreased by:

(1) Better utilization of the concentrate energy by raising matte grade.

(2) Recovery of heat.

(3) Efficient utilization of heat.

Many papers have been published on the consumption of energy in copper production including the assessment of flash smelting, indicating that oxygen enrichment with high grade matte production reduces energy requirements.

The use of oxygen reduces the gas volume, decreasing the heat carried out of the system and also fuel requirements. Since the gas volume decreases it means that a given unit can increase throughput. It enables a furnace to operate without any fuel addition.

Increasing the matte grade increases the heat produced in burning the concentrate, as shown in Table 1, and also improves the heat recovery in the system by decreasing the latent energy passing to the converters.

Process air preheating produces an effect similar but not as influential as oxygen enrichment. As steam is available, it is an obvious choice for temperatures up to 250°C with saturated steam and higher with super-heated steam. Oil heaters are also used, normally up to 500°C, but in one or two cases up to 1000°C, the economics and control do not compare with oxygen-enrichment used. Table 2 shows a heat balance for a flash smelting furnace with oxygen-enriched air.

Table 2 also shows the heat recovery for Pierce Smith converters. The difference in efficiency of heat recovery of the two systems is evident from the figures. Under normal circumstances it is not economical to recover heat from converter gases because of the low recovery and the cyclic nature of the process, which in practice reduces efficiency further and increases capital costs.

Another factor to be considered is the use of the recovered heat as steam. In many studies the assumption has been made that steam is converted into electrical energy. From the viewpoint of a purely theoretical study of energy usage this may be satisfactory, but for an economic study this is not an optimum solution. At the Harjavalta Smelter and two other smelters (one in operation and the other under construction) power generation is not the most economic alternative and steam has been used in the saturated form. Saturated steam can be used for low temperature preheating, drying, oil heating, local (district heating) and in a copper tankhouse. The oxygen plant can be run by electric or steam power (super-heated steam).

The flash smelting process is flexible in regard to energy sources. All hydrocarbon fuels can be used in the flash smelting furnace; fuel consumption can be minimized by using high oxygen enrichment in flash smelting process air, the oxygen being produced by electric energy or a superheated process steam. The price of electric power has not increased as rapidly as the price of hydrocarbon fuels and therefore fuel-free smelting has become even more attractive than before.

Outokumpu's Harjavalta smelter has been operated so that the consumption of fuel oil is only 25 kg per tonne of concentrate, and electric power is 255 kwh per tonne concentrate for the whole smelter, including the oxygen plant, acid plant and infrastructure. The steam credit at the same time is 0.6 t/t concentrate of saturated 40 bar steam. In a larger scale smelter, energy consumption is relatively lower.

ENVIRONMENTAL ASPECTS

The march of progress has brought with it a great awareness of the environment. Less is being taken for granted and more is being carried out to diminish the impact of industry on nature.

What is being done in the smelter today?

Figure I shows the sulfur recovery throughout the life of the flash smelting complex in Harjavalta, Finland. One can summarize the results in that the sulfur recovery has been in excess of 90% since the installation of the acid plant.

Is that enough?

Let us examine the flash smelter from the following viewpoints: (i) Atmospheric Emissions, (ii) Operational Conditions, (iii) Water Pollution.

(i) ATMOSPHERIC EMISSIONS

Metallurgical Processing

At the smelter in Harjavalta approximately 300 000 tonnes of sulfide material is treated annually, containing copper, nickel, iron, sulfur, lead, zinc, arsenic, and antimony, and additional chemical precipitates containing copper and nickel originating from the Kokkola cobalt and zinc plants and the Harjavalta nickel refinery.

Raw Material Handling

A characteristic of the metallurgical plants is that they process raw materials containing various amounts of valuable metals. The concentrates are usually fine-grained and the amounts processed annually come up to several hundred thousand tonnes.

The transportation, storing, drying, and feeding of the concentrates are associated with considerable problems caused by dusting. These problems are not always easy to overcome and they require disproportionately large investments. A great deal of attention has been paid to the matter in the last few years, resulting, for example, in the construction of large covered concentrate storages. This has proved to be economical if the valuable metal content in the concentrate is high.

In most processes the concentrates must be dried to zero-moisture content before further treatment. At the plants of Outokumpu, the concentrate is dried in co-current rotary driers heated by heavy fuel oil. Dust is removed from the tail gases in electrostatic precipitators or wet scrubbers and fed back to the process.

Dust and Gases in Smelting

The flash smelting method uses fine-grained dry sulfide concentrate both as a principal fuel and as a raw material for metal production. The process generates SO_2-bearing gases, which contain considerable amounts of solid particles. Owing to the nature of the process, the dust should not constitute a direct environmental problem because it has, in any case, to be removed from the gases before the SO_2 is converted to sulfuric acid or in some cases to elemental sulfur.

Converting

The copper converter matte contains between 65%-70% Cu and the nickel matte approximately 50% nickel and copper. Gases and particulates are collected in double hood systems.

Slag Treatment

The real waste from the smelting process is slag. At the Harjavalta copper smelter, the slags from both the flash smelter and the converter are cast into big blocks, which are slowly cooled. During cooling, the copper crystallizes as a sulfide mineral, which, after the crushing and milling of the moulds, is separated by flotation. The copper concentrate is fed back to the process and the fayalite slag is pumped to a tailing pond.

The slags from the Harjavalta nickel smelter are cleaned in an electrical furnace to recover nickel, and the molten fayalite slag is then granulated and stored in big heaps. The storing of the slags in the neighbourhood of the smelter causes some environmental problems.

Sulfur Dioxide

In the older copper smelting methods, SO_2 has been vented directly into the atmosphere but this practice is now prohibited in many parts of the world. The regulations vary from one country to another, but representative requirements have been presented by the E.P.A. (U.S. Environmental Protection Agency):

- Not more than 10% of the sulphur entering the smelter may be emitted to the atmosphere.
- The yearly average SO_2 content anywhere at ground level must not exceed 0.03 ppm (by volume).
- The average SO_2 content anywhere at ground level on any day must not exceed 0.14 ppm.

The two main methods for fixing SO_2 from smelter gases are: manufacture of sulfuric acid and reduction to elemental sulfur.

The recovery in both systems is dependent on the sulfur dioxide content of the gas in question. The manufacture of sulfuric acid is economically feasible when SO_2 content of the gases is more than 4%.

The flash smelting process was originally developed to reduce fuel requirements, but it has incidentally led to other process efficiencies and to more effective and economical pollution control. The SO_2 content of the flash smelting off-gases is in general very high, and as the gas volume is very stable, the gases are most suitable for producing sulfuric acid.

The distribution of sulfur in the flash smelting system differs from the conventional smelting methods in that the sulfur is confined essentially to the gas lines of the furnace and converter as a rich SO_2-bearing gas, the gas lines carrying the main load. This is illustrated in Figure II for a concentrate containing 25% Cu, 29% Fe, and 32% S, smelted to four different matte grades.

The combination of these gases from the Outokumpu flash smelter and the converter gases enables a higher recovery of sulfur with standard operating units. The sulfur losses in the smelter are outlined in Figure II.

The total recovery of sulfur at the smelter is 97.8%. The average recovery of sulfuric acid from the gases at Harjavalta is 99.7%, giving an overall recovery of sulfur as acid of 97.5%. The total emission to the atmosphere is 1.3% of the total feed.

The sulfur discarded with the slag is not dangerous to the surroundings, because it does not oxidize into gaseous products nor dissolve in water.

The ventilation gases from the flash smelter furnace section, matte launders and ladles in a modern plant can be recycled to the process air fans to be utilized as process air in the reaction shaft. At Harjavalta these gases and the fugitive gases from the converter hood are vented to the stack. Using a high stack, environmental damage can be minimized. For example, the theoretical ground-level concentrations for 70,000 Nm^3 tail gas containing 500 ppm SO_2 have been calculated in Table 3 using different wind velocities and stack heights.

Ground Level Measurements

According to ground level measurements carried out in 1975 the following approximate amounts of metals were found in the surroundings of the Harjavalta Works (measuring radius 18 km): copper 230 tonnes per annum, nickel 35 tonnes per annum, zinc 350 tonnes per annum, lead 400 kg per annum, cadmium 40 kg per annum, mercury 250 kg per annum, arsenic 6 tonnes per annum.

The amounts present in the soil in the area, plant protection agents, other future industrial emissions, and global background settlement have not been distinguished from these observed values.

The measured settlement of SO_4 was 5300 tonnes per annum. The results of total emission calculated from the measured settlements in such a vast area (1017 km^2) cannot be reliable, but conclusions on the magnitude of the emissions can be made.

The results of the measurements are probably too high, for when the wind strength rises even a little above normal on the arenaceous soil typical of the area it raises settled dust again into the air.

Ground level measurements were carried out in a smaller area in 1976 and 1977 giving dust emissions of the same magnitude as in 1975. Measurements show that zinc spreads considerably wider than other metals. The high zinc load in the years 1975-1977 is due to the fact that some of the slag from the copper furnace had to be reduced in an electric furnace in those days. Today the 9 MVA electric furnace has been replaced and zinc emissions from the plant have gone down to half of that shown above.

Measuring the Quality of Air

The company has not carried out any measurements on the quality of the air. The oxygen plant in the plant area uses an on-stream analyzer to observer SO_2 contents. The analyzer controls the quality of inlet air and should the SO_2 contents rise too high the air inlet is changed to another pipeline. Thus the equipment in question is not used for measuring higher contents. The highest SO_2 content is 860 per ug per m^3. The background content has been about 50 per ug per m^3. Airborne particulates have not been measured. A fundamental survey of the research into air protection in the town of Harjavalta was started towards the end of 1977, and the town together with industry will cover the costs. The study should be completed by the end of 1979.

Impact of Emission on the Environment

When considering the impact on the environment we should remember that the copper smelter in Harjavalta was started under very exceptional circumstances in 1945 right after the end of World War II. Copper had to be produced, but the plant for treating the gases could not be built by then and SO, SO_2-bearing gases were emitted into the atmosphere for about a year and a half. This caused practically total destruction of the coniferous forest in the immediate vicinity of the plant.

Present Emissions

Owing to the nature of the process, emissions are partly continuous and partly periodic. Drying of concentrates and steam superheating at the power plant are processes with fairly constant emissions. Both use heavy fuel oil for energy. The SO_2 contents of the gases are too low to merit piping the gases to the sulfuric acid plant. The emissions from the converters and from the reduction of nickel slag are periodic. Some converter emissions may occasionally settle on the surrounding land in the event of a disturbance.

(ii) OPERATIONAL CONDITIONS

Within the Harjavalta plant measurements have been made over a number of years and at many different locations. The data for metalliferous particulates are shown in Table 5 and for the SO_2 level in Table 6. For comparison the figures are presented at the top of each table. One can safely say that within the smelter the situation is satisfactory.

Table 4 shows the emissions for various process stages.

(iii) WATER POLLUTION

In conclusion one may say that environmental disturbances are under control, although they do exist in areas like materials handling and converter aisles, where the situation could and will be improved.

Metallurgical Plants

As mentioned before, one problem in mining is that you cannot select the location of a mine, and this can cause difficulties in handling wastes and waste waters. The location of a metallurgical plant can be selected, but in that case there are many important factors, e.g. communications, power and water supply, etc., which have to be taken into account. In most cases the plant must be built in a surrounding area with a dense population and this may add to environmental problems in the waste and waste-water treatment at such plants.

Waste Waters from Smelters

The main use of water in a smelter is for cooling purposes. Most of this water circulates in pipelines and there is no risk of contamination, but some water is used for direct cooling and collected in an open system, where impurities such as dust, etc., may be picked up.

At the Harjavalta smelter area, there are three separate drain systems:

(a) for clean cooling water,

(b) for contaminated waters and rain water;

(c) for sewage.

Waters which do not come into direct contact with the process flow into the clean cooling water drain. Part of this water contains solids, e.g. copper oxide scale, and is therefore treated in a settling tank before it enters the main drain. About 80 per cent of all the water utilized at the Harjavalta Works passes from this drain into the river.

Waters which may be contaminated, and rain water from the factory area, enter drain No. 2. These waters also flow through a settling tank, but are not treated before they enter the river.

The idea of having these waters in a separate drain is based on the principle that - should waste water treatment be needed - the water volume to be treated will be relatively small.

Sewage is pumped to the communal sewage plant.

Outokumpu's nickel refinery is also situated at the Harjavalta Works, and the cooling waters from this plant also enter drain No. 2.

The actual process waters from the Harjavalta Works are not discharged through the drains because the nickel and flotation plant waters and the slag granulation water are in a closed circuit. The most important reasons for water contamination in the plant area are the uncovered raw material stores and dusting by raw material handling. The metals are partly leached from the rain water.

Table 7 shows an example of an average monthly analysis (March 1979).

The permission granted by the Water Court allows the following metal discharges from the plant:

Cu	1000 kg per month
Ni	1500 kg per month
Zn	1500 kg per month
Pb	200 kg per month

During rainy periods the plant may have difficulties in meeting these requirements.

In summary, one could say that, under the present operating conditions prevailing in Harjavalta, it is possible to operate smelters with little environmental impact on the work area or immediate vicinity.

TABLE 1

ENERGY UTILIZATION FROM CONCENTRATE

Matte grade %	50	55	65
Energy "	52	60	69
Sulfur in gases % of input	56	64	71

CONCENTRATE ANALYSIS

Cu %	25
Fe %	29
S %	32

TABLE 2

FLASH SMELTING HEAT BALANCE

INPUT	kg, Nm^3/h	MJ/h	%
Concentrate	58 300	241 650	95.1
Flux	7 800	-	-
Flue Dust	5 150	-2 870	-1.1
Slag Concentrate	2 850	5 750	2.3
Air	35 200	8 100	3.2
Oxygen (95% O_2)	6 300	1 400	0.6
		254,030	100.0
OUTPUT			
Matte	23 670	69 300	27.3
Slag	36 040	84 550	33.3
Flue Dust	4 570	5 200	2.0
Off-gases	38 100	79 980	31.5
Heat Losses		15 000	5.9
		254,030	100.0

Heat recovered in FSF waste heat boiler	64 900 MJ/h
Heat into boiler	79 980 MJ/h
Heat recovered in converter's waste heat boiler	20 200 MJ/h
Heat into boiler	39 600 MJ/h
Energy consumed in FSF	168 930 MJ/h
Concentrate	165 100 MJ/h
Copper distribution to waste slag	0.9%

TABLE 3

Wind Velocity (km/h)	Stack Height (m)	Distance from stack (km)	SO_2 Ground level concentrates ppm (volume)
2.0	100	1.0	0.025
2.0	100	2.0	0.025
2.0	100	2.8	0.010
10.0	100	2.1	0.004
10.0	100	2.9	0.002
18.0	100	2.1	0.003
18.0	100	2.9	0.001
2.0	70	2.0	0.025
2.0	70	3.2	0.008
10.0	70	2.0	0.004
10.0	70	2.8	0.003

TABLE 4

Process Stage	Raw Material	Emissions Quality	Quantity t/a	N.B.
Drying	Cu concentrate	SO_2	700	measured
		Dust	100	measured
		- Cu 20 t/a		
	Nickel concentrate	SO_2	700	measured
		Dust	400	measured
		- Cu 20 t/a		
		- Ni 30 t/a		
Flash smelting	Copper and nickel matte and slag	SO_2	10	measured
	Operational disturbances	SO_2	300	estimated
Converting	Copper matte Nickel matte	SO_2	4000	estimated
Anode furnaces	Blister	SO_2	15	measured
	Copper	C	6	measured
Slag cleaning	Nickel slag	Dust	300	measured
Steam generation	Heavy fuel oil	SO_2	250	calculated

TABLE 5

LEVEL OF PARTICULATES IN OUTOKUMPU HARJAVALTA FLASH SMELTER

Location	Total Particulates mg/m^3	Cu mg/m^3	Zn mg/m^3	Pb mg/m^3	As mg/m^3
Finish TLV (1972) 8 h/d	10	1/0.1	5	0.15	0.5
Cu-flash smelting furnace	1.0	0.08	0.13	0.02	<0.005
Converter aisle Floor level	1.4	0.25	0.08	0.02	<0.05
Converters Control level	3.5	0.83	0.20	0.05	<0.005
Ni-slag cleaning Electric furnace	3.0	0.49	0.27	0.06	0.005

Figures are average weekly concentrations based on 10 measurements per week. Sampling time 4 hours.

TABLE 6

SULFUR DIOXIDE CONCENTRATION IN OUTOKUMPU HARJAVALTA FLASH SMELTER

Location	Week average (1) ppm	Number of measurements	Highest peak concentration ppm	8-hour average (2)
Finnish TLV (1972) 8 h/d	5			
Cu-flash smelting furnace	1.1	36	25	0.4-2.1
Converter aisle Floor level	0.2	36	5	0.1-0.4
Converters Control level	1.3	43	25	0.2-2.3
Ni-slag cleaning Electric furnace	0.9	36	25	0.3-2.3

(1) Geometric mean
(2) Min. and max. of 8-hour average during one week, geometric mean.

TABLE 7

	Drain for clean water 1	Drain for other waters 2	Total loss of metals into the drains
Cu mg/l	0.04	1.2	320 kg/month
Ni mg/l	< 0.04	3.1	< 880 kg/month
Zn mg/l	0.06	1.7	530 kg/month
Pb mg/l	< 0.01	0.04	11 kg/month
Conductivity m S/m		19	
Amount of water cubic metres/day	37000	8600	

FIGURE I

SULPHUR DISTRIBUTION AT THE HARJAVALTA WORKS

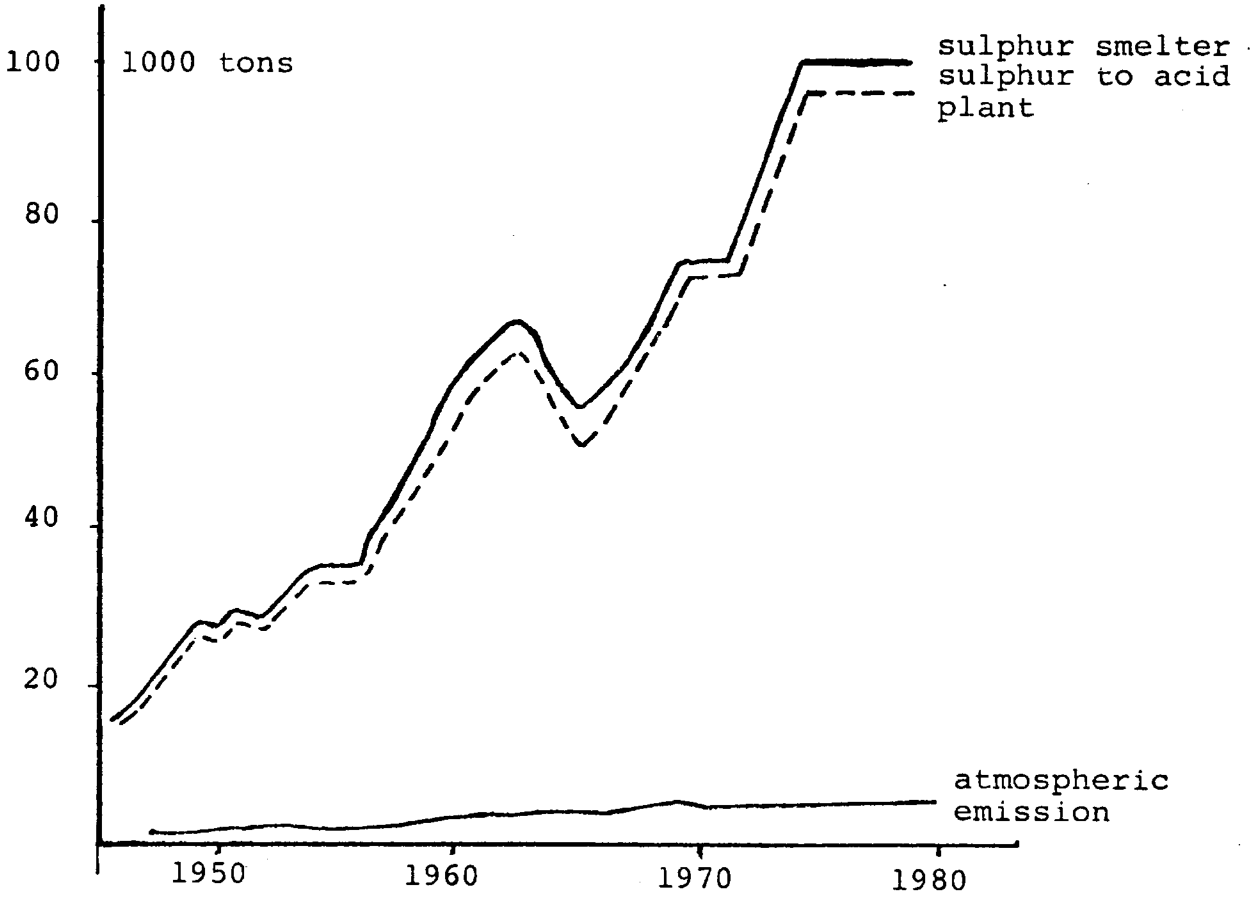
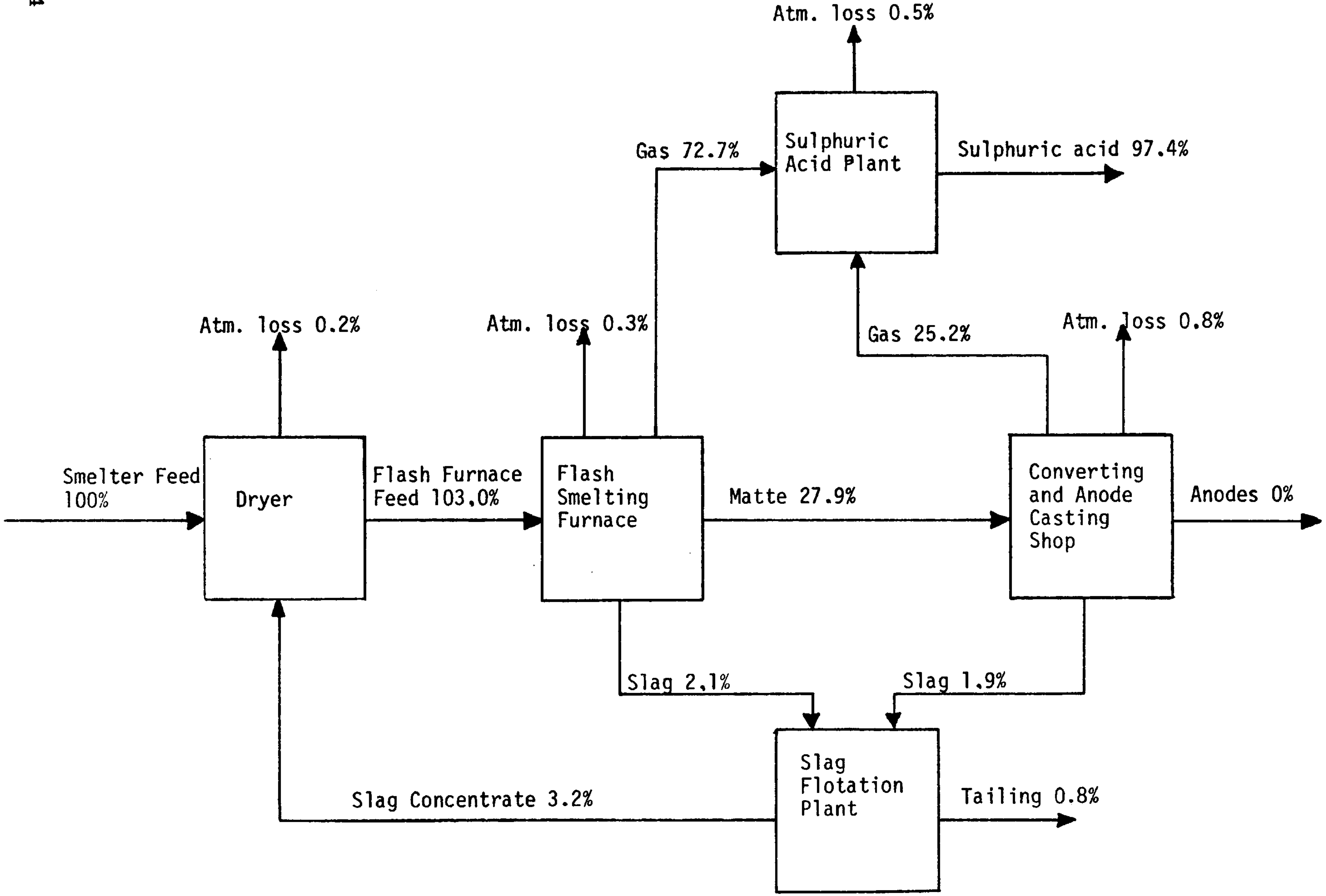


FIGURE II

SULPHUR DISTRIBUTION DIAGRAM



SECTION VI

BANQUET SPEECH

ADDRESS BY THE HON. R.H. McCLELLAND
MINISTER OF ENERGY, MINES & PETROLEUM RESOURCES

NOVEMBER 5, 1980

Good evening, ladies and gentlemen:

On behalf of the Government of British Columbia, I'd like to welcome you to our Province, to the City of Vancouver and to this Smelting and Refining Technologies Seminar.

We have guests here this evening from many parts of the world; from other parts of Canada; from the United States; from Australia and the Phillipines, Europe and Japan ... welcome. In the course of your stay with us, I hope you will be able to see some of the sights and take part in activities that are uniquely British Columbian ... especially those best experienced under the protection of an umbrella.

My Ministry is pleased to have had the opportunity to organize this seminar, the goals of which are to increase awareness in the mining community of the potential for copper smelting in this province, of the technologies applicable to British Columbia ores, and of our energy, economic and environmental circumstances which have a bearing on smelting and refining.

This seminar is very much a learning experience for us all. My Ministry is interested to hear of your plans and prospects, so as to be better equipped to lend assistance if and when proposals are submitted.

I note from your program that the seminar will include a number of highly technical talks on the merits of various smelting processes. I have been asked to give a sort of furnace-side chat to accompany these learned discussions. Well, this evening I would like to present in general terms the B.C. Government's policy framework for copper smelting and refining, outlining some of the potential benefits as well as the possible risks — as we see them.

First, I would like to digress for a moment, to review something of British Columbia's past history in copper smelting. For the lessons of history contain many of the issues facing us today, as we consider the prospects of new smelting and refining facilities in this province.

Shortly after the turn of the century, small copper smelters were in operation at Crofton and Ladysmith on Vancouver Island. Much of the copper B.C. produced was sent there for smelting, until 1914, when the new smelter of the Granby Company was 'blown in' at Anyox, a town on the North Coast of the province. The Anyox smelter transformed the North Coast region from a remote well-hole of natural resources into a self-centralized entity operating within the mining industry as a whole. New exploration ventures were keyed toward the Anyox smelter, and the whole region benefitted from additional activity and new economic strength.

The period which followed was boom time for the North Coast. By 1925, the area within a thirty-mile radius of Anyox was responsible for 55 per cent of the copper, 5 per cent of the silver and 4 per cent of the gold produced in British Columbia. Anyox had become the largest copper mine and smelter in the British Empire, producing almost 30,000 tons of copper that year.

Then came the Depression, and a general downturn in metal prices and in the mining industry in general. Copper markets dried up, and copper was stockpiled at Anyox, in hope of a reversal in the economy. That reversal came too late for Anyox. In 1934, the Anyox mines closed down and by 1937 the smelter had ceased operation. The town was deserted and then the buildings were destroyed by fire.

In the years that followed, there was no copper smelting facility in British Columbia — rather surprising, when you consider how much copper metal this province had produced over the years, and how much copper continued to be mined. That copper was sent abroad in concentrate form, to the ASARCO smelter in Washington State, to the Noranda smelter in Quebec, and more recently to smelters in Japan.

That was how matters stood until the early 1970's, when Afton Mines Ltd. established a copper smelter at its new mine near Kamloops. Afton had a particular need for smelting facilities: its copper was high grade and contained a number of impurities which made it difficult to sell on the prevailing concentrates market. Afton was assisted substantially by the provincial government's copper smelting incentives program, which provides 5 cents a kilogram for blister copper. Unfortunately, the Afton process is not suitable for smelting the bulk of British Columbia ores. This is a pity considering how close the smelter is to the Highland Valley, this province's principal copper producing region.

This historical review brings us to the critical question: should British Columbia undertake to establish new copper smelting and refining facilities to up-grade our ores? That is the question I would like to consider with you now.

Let's look first at the economics of copper smelting in British Columbia, in general terms. Traditionally, we have held a strong position in the world copper concentrates markets. This position is based on our sizeable copper production and our access to tidewater and the Pacific Rim markets. We export approximately one quarter of a billion kilograms of concentrates a year, which makes us one of the free world's largest concentrate exporters.

However, in the next few years, world copper smelting facilities will be approaching full capacity utilization. We expect the Japanese smelters to be at capacity by 1985, and I understand that Japan is not planning substantial expansions. I am told that in the United States some smelter capacity is being retired. So, by mid-decade, the world may well be looking for new smelter sites.

In the Pacific region, a number of sites present themselves. Of these, British Columbia possesses several distinct advantages: stable economic and political climates; ample energy resources; good labour relations; and most importantly, immediate access to substantial reserves of copper. So, now might be a very good time for serious consideration of new copper smelting capacity in British Columbia.

The chief benefit to British Columbia of such a move would be the economic impact, both locally and internationally. New smelting facilities would result in additional activity in construction and other industries, additional employment in many sectors, regional development and a better outlook for the province's mining industry. Indeed, we would realize many of the same gains and advantages, on a province-wide scale, as were felt at Anyox many years ago.

But the chief economic gain would be made in the international copper markets. Prices tend to fluctuate on both the world concentrate and world metal markets; and usually, when the price for copper concentrate is relatively low, the price for copper metal is relatively high. These prices move in relation to one another, and right now, British Columbia is not in a position to take advantage of this interlocking market for our copper. By developing smelting capacity in this province, we would be able to make some direct economic gain, but more importantly, we could provide greater stability for British Columbia mining industry by participating in both concentrates and metals markets.

The next link in the copper chain, the copper fabrication industry, would be that much closer to realization, assuming the right economic climate. Copper smelting facilities in British Columbia would contribute to our copper self-sufficiency, as well as improving our position on world copper markets.

For years, smelting activity has been located close to the markets for finished products, with little processing activity in the producing regions. That is changing. The clear and absolute orientation in both areas has been breaking down for some time now. The diversification of our role in the world copper market seems in order — a broadening of our industry's base, and an opportunity to provide greater economic stability.

Now, what about the potential risks involved in new smelting ventures in British Columbia?

One of the chief concerns about copper smelters anywhere has been their impact on the environment. Any new smelter in British Columbia would be subject to the government's Guidelines for Intensive Development. We have enabling legislation in place and are in the process of finalizing the guidelines themselves, which will serve to assess the environmental and socio-economic impact of major industrial projects such as smelters.

Now, government regulation is rarely popular with proponents of industrial development. However, this government believes in streamlining the regulatory process as much as possible. Such is the case in our major energy projects review process, and such will, I am sure, be the case with the Guidelines for Intensive Development. In each case we want to ensure maximum benefit to the province from the development of its resources, while at the same time making sure that other less tangible resources, such as quality of life or the integrity of the environment in which we live, are not lost along the way.

I am told that contemporary smelting and refining technologies are becoming more and more sophisticated, with less pollution and better sulfur recoveries. Never again will we see in this province the denuded hillsides and slag heaps and polluted rivers and streams that were present at Anyox in the early years. The Afton operation is a case in point. It is one of the cleanest copper smelting plants in the world. The company went to some lengths, in response to local concerns and government regulations, to make sure the environmental impact of the smelter would be minimal. Smelters have a bad name among environmentalists, but Afton is doing its part to change that. And so are we all. Our site-specific environmental guidelines are one of the ways to achieve this end.

The other risk involves the economics of copper smelting. We must ask ourselves whether the investment of risk capital in smelting and refining in this province is worthwhile. After all, British Columbia copper interests already have a strong place in the concentrates markets, as I have said. However, we would hope to increase our total copper sales, by fulfilling our present concentrate contracts and selling copper metal as well. I wish to emphasize that government policy is not intended to promote further mineral processing at the expense of our presently attractive concentrate markets. Our industry is viable at the moment; but we are always on the lookout for new opportunities which will enhance the value of our resources and broaden the base of our economy.

Whatever decisions are taken on smelting and refining in British Columbia, I am sure that this province will remain a reliable supplier of concentrates to the world market. We will honour our contracts and continue to play a role in filling the world's copper needs. We certainly have enough copper to enter the copper metals markets; we have the location, the economic and industrial base, and access to advanced copper smelting and refining technologies, as this seminar clearly shows. I guess it comes down to a matter of timing and economics. The government is cautiously optimistic about British Columbia's copper smelting prospects. We look forward to hearing from the industry on this subject, when they think the time is right.

Meanwhile, I wish you fruitful discussions at this conference, as you explore the technological side of British Columbia's smelting prospects.

Ladies and gentlemen, I would be remiss if I did not take advantage of this my first opportunity to respond in a considered manner to the federal governments recently announced budget and energy program.

I can't tell you, in absolute terms, exactly how much the budgetary moves and energy proposals are going to cost British Columbia. Officials in my ministry and in the Ministry of Finance are still working out the dollar figures. But preliminary estimates indicate that these moves will have a direct and detrimental impact on various sectors of our economy, and that they will seriously undermine our efforts to achieve energy security for this province.

I <u>can</u> tell you that this budget, with its centre-fold spread energy program, will have the immediate effect of setting industry against government, and government against government; and that as soon as the impact of the new energy taxes begins to be felt by consumers, <u>that</u> will set individual Canadians from all parts of this land against the government in Ottawa. They will demand to know, in clear terms, why they are being taxed, and exactly how their money is being spent.

This is a deceptive budget. On the surface, it looks good for the individual taxpayer. By means of this deception, the federal government hopes to buy some time. It hopes to distract the attention of Canadians from the pressing matters of the economy and energy security, by giving them the impression Ottawa has these matters well in hand. In fact, that's just not so.

This so-called budget is no more than a money-grab and a power-grab. It shifts a disproportionate amount of the economic burden of energy self-sufficiency from the federal government onto Canadian consumers, onto the producing provinces and the energy industries. It does so in an underhanded manner. Ottawa hopes to suddenly discover a whole pile of what appears to be extra revenues, which it will then happily use to pay off its deficit — the product of more turn a decade of liberal fiscal mismanagement — and then to pay off its political debts in central Canada, thereby ensuring re-election.

Most seriously of all, this budget is being used by the federal government to gather more and more power unto itself, changing this country constitutionally, through non-constitutional means. There seems no limit to what this government is prepared to do to get its way.

Let's consider for a moment some of the specific proposals contained in the budget and the energy program.

(1) Blended Price for Oil

Until now, Ottawa has paid the difference between the domestic price and the world price for oil through the oil import compensation program. They have used normal taxation revenues to do so. The new blended price for oil is a continuation of this policy. However, once the blended price is phased in, the cost will no longer be borne by the federal government. Rather, it will be paid by the Petroleum Compensation charge, which will be levied on refineries, and then passed directly on to the consumers. By this move, the federal government will save itself an estimated 4.3 million dollars per day and do next to nothing to bring a fair return to the producing provinces on their depleting oil reserves. The net result is that Canadian consumers will be paying more for oil, producing provinces will not be getting anywhere near replacement value for their oil, and Ottawa will find itself with a lot of extra money to play around with.

(2) Natural Gas Excise Tax

The federal government says it wants Canadians to switch off oil and onto other more plentiful energy sources, such as natural gas. They say price will be a prime incentive. Yet they turn around and slap an excise tax and a special charge on that very commodity. Canadian consumers, trying to help make Canada more energy secure, pay again.

In addition, Ottawa has decided to place an export tax on natural gas. They won't call it an export tax, per se, but that's what it is. And British Columbia remains unalterably opposed to it.

They justify these taxes on natural gas in a number of far-fetched ways. For instance, they say the federal government has a right to a share of the western energy resource revenues, which they call unwarranted windfall profits. That's pious rationalization. The real reason for the tax comes a few pages further on in their energy document: "The federal government is seriously short of the revenue it requires to manage the Canadian economy." They could raise these revenues through normal taxation means, and cut their own expenses at the same time. But that might be unpopular, especially in central Canada.

So Ottawa needs revenues, which it intends to capture through devious means. That seems to be the bottom line to all their energy tax proposals:

- The 8 per cent (plus) tax on net operating revenues of oil and gas companies;

- The Canada-lands incentives, which will encourage oil and gas activity there, at the expense of operators in other regions and at the same time will bring revenues from the north and the off-shore directly back to Ottawa;

- The Canadianization of the oil and gas industries, which will be paid for by consumers, from which Ottawa will gain dividends, and which will not produce one more barrel of oil or Mcf of natural gas for Canada;

- The Natural Gas bank, which will allow Ottawa to buy gas cheap from the small companies it has driven to the brink of bankruptcy, and then sell it at a profit.

I haven't time this evening to go into all these programs in detail. But you get the general drift. The impact of Ottawa's moves, in terms of direct costs, will fall first on the petroleum and natural gas and related industries, then on the producing provinces, and throughout the whole process, on Canadian consumers of energy. Never in the history of Canada has there been such direct and unilateral intrusion by government into the activities and income of specific sectors of the economy. And who says it will necessarily stop there. The budget papers include provisions to remove from mining exploration companies certain tax depletion allowances — a signal of intended federal intrusion into other resource sectors. And never has there been proposed, by any truly federal government, anywhere, such a complete and relentless raid on the revenues of regions within the nation.

As I say, we don't yet know the full impact of these policies on British Columbia's economy or on our energy security. We won't know until we see the full extent of Ottawa's raid on provincial treasuries. We won't know until we see how much money the federal government plans to take out of the pockets of individual British Columbians. We won't know until we have a sense of the damage done in the oil and gas industries and the private sector as a whole, or until we learn how these policies will affect energy pricing and supply, the two chief elements of our energy security. Unfortunately, when we do know, it may be extremely difficult to repair the damage.

I'd like to conclude by talking for a moment about sharing. Mr. MacEachen said in his budget speech that "the concept of sharing seems to be close to the distinctive fibre of the Canadian tradition." The federal government admits, with respect to natural resource revenues, that there is "no legislatively-defined arrangement under the B.N.A. Act for sharing". In other words, as we have been saying all along, the provinces own their resources.

Nevertheless, we have shared with our fellow Canadians. British Columbia has paid into the federal equalization program almost since its inception. And we have, through tariffs on imported goods, paid to support central Canada's manufacturing industry for many years. And we are not alone in our sharing. In July, the province of Alberta, which also pays into equalization, offered to share more of its new-found wealth with the rest of Canada, through transportation, energy and other initiatives, to the benefit of all Canadians. If only the federal government would allow them a fair price for their depleting energy resources. But they were turned down flat. Obviously, as far as Ottawa was concerned, they didn't offer to share enough. We haven't offered to share enough. How much is enough?

In the federal government's energy document, they say that "the revenue share accruing to each level of government is a function of a mixture of fiscal instruments that has evolved over time." Well, the fiscal instruments imposed unilaterally by Ottawa's current proposals defines clearly their concept of sharing. To them, sharing means grabbing as much revenue as they can possibly get their hands on.

The west has shared willingly, over the years, because we agree that sharing _is_ what Canada is all about. Now we are not being asked to share, we are being told to share. We are being told we are the bad boys of confederation. We are being told that, while we own our resources, they are not really ours to control. They are Ottawa's to take, whether we like it or not.

It's as if we are sitting in on a card game where the stakes are high and where the dealer makes up the rules as he goes along, to serve his own advantage. Our resource ownership has been turned into a worthless pile of chips. If we bet and win, our winnings go to Ottawa; if we bet and lose, we take the losses.

By its budgetary and energy proposals, the federal government has put British Columbia and the other energy producing provinces, and the energy companies, and the Canadian consumer into a no-win situation. Either we fold what we know is a good hand, and pay the dealer, or we get up from the table and walk away from the game. That's a terrible choice for Canadians to have to make. And as British Columbians and good Canadians, we don't want to be forced to make it.

Thank you very much.

SECTION VII

APPLICATION OF THE SHERRITT-COMINCO COPPER PROCESS TO BRITISH COLUMBIA COPPER CONCENTRATES

BY P. KAWULKA
C.R. KIRBY

SHERRITT GORDON MINES LIMITED
AND COMINCO LTD.

APPLICATION OF THE SHERRITT-COMINCO COPPER PROCESS TO BRITISH COLUMBIA COPPER CONCENTRATES

BY P. KAWULKA *
C. R. KIRBY **

This paper is to be presented at the British Columbia Copper Smelting and Refining Technologies Seminar to be held in Vancouver, B.C. on November 5 and 6, 1980. The seminar is organized by the B.C. Ministry of Energy, Mines and Petroleum Resources.

If and when this paper is published, it may embody certain changes made by the authors so that the form in which it appears here is not necessarily that in which it may be published later.

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ABSTRACT

The S-C Copper Process has been adapted to treat a typical British Columbia copper concentrate. A process flow diagram, a schematic flow diagram and process description are presented for 75 000 tonnes per year copper rod production from copper concentrate. Basic operating requirements are outlined and a preliminary estimate of operating and capital costs for a Western Canada location are summarized. The costs are compared with those of an equivalent modern flash smelter/electrorefinery.

INTRODUCTION

The S-C Copper Process was developed by Sherritt Gordon Mines Limited and Cominco Ltd. to fill a need for an improved alternative to smelting.

Laboratory work was conducted jointly by both companies. Technical feasibility was demonstrated in a 9 tonnes per day integrated pilot plant (1-2). Commercial evaluation showed the S-C Copper Process to have substantial advantages over flash smelting (3). The salient features of the process are:

- 98% or better recovery of copper with quality equal to electro-refined.
- Gold and silver recoveries comparable to smelting.
- Recovery of 60-90% of the sulfur in elemental form, the remainder being in a discardable residue.
- Recovery of associated metal values such as molybdenum and zinc as saleable by-products.
- A considerable measure of relief obtained from problems of hygiene because of total containment of the process.
- Applicability to a wide range of concentrate grades and compositions.
- Competitive costs as compared to a modern flash smelter/electrorefinery.

The process was developed for a low grade Manitoba copper concentrate, chalcopyrite with substantial pyrite, also containing recoverable zinc values in addition to gold and silver. For this presentation the process was adapted to treat a bornite-chalcopyrite concentrate of a composition more typical of British Columbia. The concentrate analyses are shown in Table 1. This analysis does not refer to a specific concentrate but is a copper grade obtained by a calculated weighted average representing present and future concentrates. The British

Columbia concentrate contains insufficient zinc to warrant separate recovery and has less sulfur and iron per unit of copper which makes it easier to treat than the Manitoban concentrates by the S-C Process.

To illustrate the economics, a preliminary design and cost estimate have been prepared for a 75 000 tonnes per year S-C Copper Plant located in Western Canada. These costs, which are adapted from a previous study (12), compare favourably with those of an equivalent modern flash smelter/electro-refinery.

CONCENTRATE DESCRIPTION

Representative analysis of Manitoba and British Columbia copper concentrates are compared in Table 1.

PROCESS DESCRIPTION

The following main steps are required for processing B.C. concentrates:

(a) Iron Removal Circuits

Thermal activation.
Acid leach.
Jarosite precipitation.

(b) Copper Winning Circuits

Oxidation leach.
Purification.
Electrowinning or hydrogen reduction.

The flowsheet is shown in Figure 1. It is a simplification of the basic flowsheet which was designed to treat concentrate of much greater complexity than the B.C. concentrate being considered. The design is based on key laboratory tests and on a computerized mass balance. Typical analyses of the main process streams are shown in Table 2.

THERMAL ACTIVATION

A key feature of the S-C Copper Process is a thorough removal of iron from the concentrate prior to the solubilizing of copper. For the B.C. concentrate this is achieved through thermal activation of the concentrate to make iron adequately and selectively soluble in a subsequent non-oxidizing sulfuric acid leach.

The activation takes place in two stages in a multi-hearth roaster. In the first stage about 12% of the concentrate sulfur is removed by thermal decomposition of pyrite and chalcopyrite at 680°C. In the second stage a further 4% is removed as hydrogen sulphide by counter flowing hydrogen. The calcine so produced corresponds to a mixture of bornite (Cu_5FeS_4) and troilite (FeS).

The reactions are:

Delabilization (First Stage)

$$5CuFeS_2 \xrightarrow{\text{Heat}} 5CuFeS\,1.8 + 1/2\ S_2 \qquad (1)$$

$$7FeS_2 \xrightarrow{\text{Heat}} Fe_7S_8 + 3\ S_2 \qquad (2)$$

Reduction (Second Stage)

$$5CuFeS1.8 + H_2 \xrightarrow{\quad} Cu_5FeS_4 + 4FeS + H_2S \qquad (3)$$

$$Fe_7S_8 + H_2 \xrightarrow{\quad} 7FeS + H_2S \qquad (4)$$

ACID LEACH

The thermally activated calcine slurry is leached with sulfuric acid solution to dissolve iron to the reaction:

$$FeS + H_2SO_4 \xrightarrow{\quad} FeSO_4 + H_2S \qquad (5)$$

A two-stage countercurrent leach is necessary to satisfy the process requirements of a product liquor with a high iron tenor (63 g/L) and low acidity (30 g/L) in the first stage, and a high degree of iron dissolution in the second stage. To extract 90% of the iron, an acid concentration of 200 gpl is required in the second stage. If a concentrate with higher iron or with a substantial zinc content were to be treated, the acid leach would be followed by a leach in recycled copper sulfate solution to dissolve more iron and part of the zinc by substitution for copper. This step, called the Activation Leach, is not necessary for the B.C. concentrate.

JAROSITE PRECIPITATION

After liquid/solid separation of the final acid leach slurry, iron is precipitated from the solution. The objectives are the precipitation of iron as a solid with good filtering characteristics and regeneration of acid to 50-60 g/L for recycle to the acid leach. To achieve this, iron is precipitated as an ammonium jarosite according to the equation:

$$6FeSO_4 + 2NH_4OH + 3/2\ O_2 + 7H_2O \underline{\qquad\quad} 2NH_4Fe_3(SO_4)_2(OH)_6 \\ + 2H_2SO_4 \quad (6)$$

Alternatively, iron could be rejected as a sodium jarosite by providing a source of sodium ions instead of ammonium ions.

OXIDATION LEACH

The acid leach residue is leached in acid solution under oxygen pressure to convert the copper sulfides to copper sulfates. The overall reaction is:

$$Cu_5FeS_4 + 6H_2SO_4 + 3O_2 \underline{\qquad} 5CuSO_4 + FeSO_4 + \\ 4S^\circ + 6H_2 + 0 \quad (7)$$

The major objectives of this step are:

- To obtain greater than 98% extraction of copper and maximum conversion of sulphidic sulfur to elemental sulfur.

- To obtain a product liquor containing only 10-15 g/L H_2SO_4 while avoiding hydrolysis of ferric iron.

- To carry the iron in solution mainly as ferrous in order to meet the requirements of the subsequent solution purification.

SOLUTION PURIFICATION

Impurities such as Te, As, Bi, Sb, Pb and Se are removed by coprecipitation with Fe_2O_3. To precipitate Se its valence of six must first be reduced to four by ferrous iron at 200°C, followed by coprecipitation with iron oxide.

ELECTROWINNING

Copper is electrowon from the purified solution at high current density in air-sparged cells. The use of air sparging makes it possible to operate satisfactorily when the ratio of current density to copper content in the electrolyte is between 16 and 24 amperes/metre^2/g/L. Above this range, the deposits become rough.

HYDROGEN REDUCTION

As an alternative to electrowinning, recovery of copper powder by continuous hydrogen reduction of purified solutions was proven technically feasible in the laboratory and mini pilot plant. Plastering and plating of copper onto equipment internals during reduction can be overcome by addition of high molecular weight polyethylenimine. The additive also promotes the growth of dense powders which except for sulfur are comparable in purity to electrowon copper. The physical properties such as electrical conductivity and spring elongation are equivalent to electro-refined copper.

COMMERCIAL PLANT DESIGN

The brief description that follows reflects the design of the 75 000 tonnes per year S-C copper plant. The schematic flowsheet is presented in Figure 2.

copper concentrates. Copper recovery for medium and high grade concentrates will be equivalent to that from smelting and is better than smelting for lower grade pyrite feeds. This permits a mill to attempt higher recovery at a cost of lower concentrate grade, with beneficial effects on the overall economics in addition to making better use of resources.

Treatment steps can be incorporated to recover zinc and molybdenum in addition to gold and silver if these metals are present in economically significant quantities. Molydenum and precious metals report to the oxidation leach residue from which these can be separated by techniques such as flotation, solvent extraction or hot filtration for sulfur removal, resulting in a final residue that contains the precious metals or molybdenum in concentrated form.

For a high pyrite copper concentrate that frequently also contains 2-5% zinc, pelletization may be necessary to minimize sintering of the charge during thermal activation as slightly higher temperatures are required. In addition, the extra iron and the need to dissolve zinc require an activation leach after the acid leach. This is a displacement leach using recycled copper sulfate solution at 150°-160° in a neutral atmosphere. Zinc is precipitated with H_2S as a clean sulfide from the activation leach solutions.

ECONOMICS

BASIS OF COMPARISON

The economic comparison with Flash Smelting prepared for this presentation is based on grass roots plants producing 75 000 tonnes per year of copper rod, in a 24 hour per day, 340 days per year operation. Recovery of copper from concentrate is taken to be 98% for the S-C and smelter plants.

The Outokumpu flash smelter was selected for this comparison because published data from many operating plants were available (5-11).

No credits have been allowed for the elemental sulfur recovered in

THERMAL ACTIVATION, SULFURIC ACID PRODUCTION

About 800 t/d of copper concentrate (30% copper) is charged to two 16-hearth Herreshoff-type roasters. Roaster internal steel is Alonized and refractories are conventional. In the upper eight hearths concentrate is heated to about 680°C to drive off labile sulfur from pyrite and chalcopyrite, and the sulfur is burned in admitted air to provide most of the necessary heat. On entering the reduction section, the preheated solids are contacted with counter-flowing hydrogen gas from a reform plant and recirculation system.

Additional sulfur is removed as H_2S. The minor amount of heat required by the reduction section is supplied by combustion of a small part of the hydrogen. About 90% of the reduction gases are drawn off the top reduction hearth for cleaning and recycle; the remainder is burned in the upper hearths.

Roaster top gases containing over 10% SO_2 pass through a dust removal system to a double contact sulfuric acid plant with tail gas absorption, which produces 235 tonnes of H_2SO_4 daily for use in the leach plants. The activated roaster product is quenched and stored for acid leaching.

ACID LEACHING, JAROSITE PRECIPITATION

The acid leach of the activated concentrate is conducted at 70°C in closed rubber-lined tanks under slight suction in sulfuric acid returned from the jarosite section together with make up acid. About 60% of the contained iron dissolves in the first stage, a further 30% in the second stage. Most of the hydrogen sulphide gas evolving is drawn off to a small Claus plant to recover about 50 tonnes per day of elemental sulfur. A lesser part of the H_2S is diverted to be burned to SO_2, joining roaster gases for acid production. A still smaller part is used to precipitate copper in the bleed circuit.

The copper-free ferrous sulphate liquor from the first stage of the acid leach is pumped through a flash steam preheater to a four compartment titanium-clad, mild steel autoclave. Ammonium or sodium ions are added and oxygen is injected. The reaction at 190°C and 2160 kPa absolute pressure precipitates iron as a jarosite, a form of complex basic ferric sulphate, and regenerates some sulfuric acid. After flashing and solids separation, the acid solution is contacted with SO_2 to reduce any residual ferric iron in solution to the ferrous state. The jarosite slurry is pumped to a lined pond from which the water is recycled. The solution flow in this loop is approximately 90 m^3/h (333 imperial gallons per minute).

OXIDATION LEACHING

Acid leach residue, return acid from copper winning and tonnage oxygen (95% O_2) are reacted in a two-stage oxidation leach in which copper and residual iron and zinc are dissolved and elemental sulfur is formed. The leach vessels are four-compartment autoclaves, lead clad and brick lines. The first stage employs four autoclaves and the second stage two. Over 84% of the copper is extracted in the first stage continuous leach, which operates at 100°C and 390 kPa pressure. The second stage is a batch leach at 100°C and 1370 kPa and brings the overall copper extraction to over 98%.

After solids separation, the second stage leach liquor recycles to the first stage and the residue goes to a storage area for possible future recovery of elemental sulfur and precious metals. Copper leaves the oxidation leach section in the first stage oxidation leach liquor, passing on to purification and electrolysis. Oxidation leach flows are 154 m^3/h (563 imperial gallons per minute) in the first stage and 528 m^3/d (1970 imperial gallons per day) handled in batch operation in the second stage. Oxidation leach residue contains about 72 tonnes per day of sulfur together with gangue and a small amount of residual sulfides.

SOLUTION PURIFICATION

To prepare the solution for recovery of copper, the solution from oxidation leaching is given a rigorous purification to remove elements that would contaminate the final copper product or lower the current efficiency. Some of these elements are selenium, tellurium, iron, bismuth, and arsenic. Selenium is brought to the tetravalent state by a combination of high temperature (200°C), and a reducing agent, most conveniently the solution's ferrous iron. This takes place in a standard autoclave, titanium clad, and may apply to all or only part of the electrolyte depending on selenium content and valence. Heat is supplied to the solution by exchange with flash steam and through direct steam injection. Following this, in a second autoclave of the same type, at 190°C and 1720 kPa total pressure, iron is oxidized and precipitated as a hydrated ferric oxide, taking with it the major portion of the arsenic, selenium, bismuth, and other impurities. These procedures leave an extremely pure solution, far cleaner than copper refinery electrolyte, which after pressure filtration passes to copper winning. The flow through the purification section is about 140 m^3/h (516 imperial gallons per minute).

ELECTROWINNING

The design of the electrowinning plant incorporates most of the recent advances (4-5) in this area, including high current density operation (650 and 430 A/m^2) in covered air-sparged cells using lead-alloy anodes and strippable titanium cathodes instead of starting sheets. These innovations make for a compact and cheaper tank room. Feed to electrowinning is 122 m^3/h (447 imperial gallons per minute). Cell electrolyte analysis is 30 g/L copper, 140 g/L sulfuric acid. Return acid with 20 g/L Cu, 150 g/L H_2SO_4 recycles to the oxidation leach. Cell potential drop is 2.5 volts with a direct current flow of 144 000 amperes, plating 1.15 tonnes of copper daily per operating cell. Current efficiency of over 90% was demonstrated in the pilot plant. Cathode copper is carefully washed, stripped and then melted and processed to rod. The purity and physical properties of the product meet the existing standards of the copper industry.

BLEED FOR IMPURITY CONTROL

About 7 m^3/h (25 imperial gallons per minute) are bled from the return electrolyte stream to purge out the zinc and magnesium. The contained copper is first precipitated with H_2S and the resulting CuS recycled to the oxidation leach. The stripped acid liquor is then neutralized with limestone and lime and the slurry stored in a tailings pond.

GENERAL DESIGN FEATURES

To conclude the plant description a few final points should be added. The design that has been costed employs at every point materials of construction and linings that pilot testing has shown to be satisfactory in that service. The autoclaves have already been mentioned. Other process vessels, tanks, thickeners, filters are fibre reinforced plastic or rubber lined mild steel; pumps are nearly all stainless steel or Hastelloy; autoclave agitators, coils and heat exchangers are generally of titanium.

Process buildings are of standard industrial design, requiring only a fraction of the ventilation and dust handling facilities required in pyrometallurgical plants. The unit operations embodied in the process design, such as pressure leaching, liquid/solid separation, heat exchange and sulfide flotation are standard in hydro-metallurgical practice and are part of existing Sherritt-Gordon and Cominco plant operations. The auxiliary plants, mainly sulfuric acid, oxygen, hydrogen and Claus, are standard.

S-C COPPER PROCESS EVALUATION

HYGIENE AND ENVIRONMENTAL CONTROL

Turning now to an evaluation of this process as it has been described, a number of aspects in relation to copper metallurgy in general will be mentioned and finally the economics will be presented.

Hydro-metallurgy provides distinct advantages to the copper industry in its struggle to meet tightening hygiene and environmental regulations. The S-C Copper Process offers the following relevant features which are basic to the technology:

- It has no gas emission problems.
- Heat and dust are minimized.
- It does not generate highly arsenical intermediates.
- Process and cooling water are recirculated.
- Threshold limit values for Pb, As, SO_2 can be easily met (as was demonstrated during piloting operations).
- Jarosite is impounded in a lined pond.
- Lead and mercury report to the tailings.

The process employs H_2S as a chemical reagent. In twenty-five years of trouble-free operation, Sherritt has demonstrated the techniques and safety procedures for handling this gas.

SULFUR DISPOSAL

To recover 95% of the input sulfur to conform with emission regulations a smelter must produce about 2.5 tonnes per day of sulfuric acid per tonne of copper. Historically there has been a surplus of this acid which, in relation to its value, is very expensive to transport to market. A common integration is the associated production of phosphoric acid: this may prove unprofitable or inconvenient. Even less attractive alternatives are to neutralize the acid to gypsum or to reduce SO_2 to elemental sulfur. The first requires about 4 tonnes of limestone per tonne of copper, creating large quantities of gypsum to be disposed of while the second is expensive and uses much energy. The S-C Copper Process produces elemental sulfur directly which can be easily stored and recovered for possible future use.

ENERGY REQUIREMENTS

Hydro-metallurgical copper processes generally consume more energy than pyrometallurgical processes and the S-C is no exception. Its consumption of energy in various forms is contrasted with flash smelting in Table 3.

The higher energy requirement for the S-C Copper Process is mainly due to the electrowinning step which requires the same current flow as electrorefining but at ten times the voltage. It should be noted, however, that the elemental sulfur produced by the S-C Copper Process has a potential energy credit of 4100 kJ/kg of copper. Despite the higher energy consumption, the operating costs for the S-C Copper Process are not markedly higher than those for smelting. On the other hand, smelter operation in the future may involve higher energy and operating costs if SO_2 must be disposed of as sulfur or neutralized sulphate.

PROCESS FLEXIBILITY

The S-C Copper Process can be adapted to treat a wide range of copper concentrates. Copper recovery for medium and high grade the S-C Copper Process nor for the by-product acid produced in a flash smelter.

OPERATING REQUIREMENTS

The operating requirements for the S-C copper hydro-metallurgical plant and the flash smelter-electrorefinery are compared in Table 4. They are based on material and energy balances.

COST COMPARISON

Table 5 summarizes the comparative costs. The capital costs, estimated to battery limits in 1980 Canadian dollars, are for grass roots plants located in Western Canada. No allowances are made for financing charges or escalation. The equipment cost for the S-C Copper Process was based on sizing developed from pilot plant criteria and on recent quotations and purchases. The installed costs were derived from equipment costs by a combination of modular estimating and factoring techniques.

The estimated working capital difference is due mainly to differences in in-process inventories.

Operating costs are estimated from the requirements summarized in Table 4. The unit cost for labour, fuel, electric power and ammonia are taken at $12/man hour, $2.11/GJ, 0.50c/kW.h and $250/tonne respectively.

The economic comparison is in favour of the S-C plant, in as much as its fixed and working capital advantage more than compensates for the flash smelter's lower operating cost. If the latter's acid production had to be neutralized for disposal, capital costs would increase by about 5% and operating costs would increase by about 25% (12). These overall economics, coupled with other advantages mentioned earlier make the S-C Copper Process a viable alternative to the smelting/electrorefining route traditional in the copper industry.

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REFERENCES

(1) Swinkels G.M., Berezowsky R.M.G.S.; "The Sherritt-Cominco Copper Process - Part 1: "The Process", CIM Bulletin, Vol. 71 February, 1978, pp 105-121.

(2) Kawulka P., Kirby C.R., Bolton, G.L.; "The Sherritt-Cominco Copper Process - Part II: Pilot Plant Operation", CIM Bulletin, Vol. 71, February, 1978, pp 122-130.

(3) Maschmeyer, D.E.G., Milner, E.F.G., Parekh, B.M.; "The Sherritt-Cominco Copper Process - Part III: Commercial Implications", CIM Bulletin, Vol. 71, February, 1978, pp 131-139.

(4) Harvey W.W., Randlett M.R., Bangerskis K.I.; "Cell Design Components of the Ledgemont Air-Agitation/Close Spacing Methods of High Current Density Copper Electrowinning and Electrorefining", pp 379-80, Chemistry and Industry, May 3, 1975.

(5) Ettel V.A., Gendron A.S., Tilak B.V.; "Electrowinning Copper at High Current Densities:, pp 31-36, Metallurgical Transactions (B), Vol.6B, 1975.

(6) Palmer, K.; "Recovery of Acid and Elemental Sulphur From Copper Smelter Gases to Conform With New Environmental Standards"; Paper presented at June 1974 IMM Symposium, "Minerals and Environment".

(7) Fujii, T., Ando M., Fujiwara Y.; "Copper Smelting by Flash Furnace at Saganoseki Smelter and Refinery"; Paper presented to MMIJ-AIME, May, 1972.

(8) Kaasila K., Loytymaki E.; "Heat Recovery in Metallurgical Applications Within the Outokumpu Company", Advances in Extractive Metallurgy and Refining, Jones M.J. (Ed.), Institute of Mining and Metallurgy, 1972.

(9) Morisaki T., Ikeda H.; "The Design, Construction and Operation of the Tank House at Onahama Smelting and Refining Co. Ltd."; Paper presented to 1968 AIME Annual Meeting.

(10) Brunson W.W., Stone D.R.; "Electrorefining at the Copper Division of Southwire Company, Carrollton, Georgia, USA"; Paper presented to 1975 Annual Meeting of TMS-AIME.

(11) Schwartz W.H.; "Economics of Pyrometallurgical Copper Processes", TMS Paper Selection No. A75-57.

(12) Maschmeyer, D.E.G., Kawulka P., Milner E.F.G., Swinkels G.M.; "Application of the Sherritt-Cominco Copper Process to Arizona Copper Concentrates"; Journal of Metals, Vol. 30, No. 7, July 1978, pp. 27-31.

TABLE 1

ANALYSIS OF COPPER CONCENTRATES

	MANITOBA	BRITISH COLUMBIA
Cu	23.4%	30.0%
Fe	32.0%	20.9%
S	35.0%	26.3%
Zn	3.0%	0.3%
Mg	1.84%	1.2%
OTHER	4.76%	21.3%

TABLE 2

S-C COPPER PROCESS SUMMARY
TYPICAL CHEMICAL ANALYSES OF MAIN PROCESS STREAMS (% or g/L)

	Cu	Fe	Zn	$S^=$	S°	H_2SO_4
Concentrate (%)	30.0	20.9	0.3	26.3	-	-
Activated Calcine (%)	31.7	22.1	0.3	22.0	-	-
Acid Leach Residue (%)	47.4	4.4	0.5	14.2	-	-
Acid Leach Solution (g/L)	-	62.6	-	-	-	29
Jarosite Precipitate (%)	0.2	38.1	-	12.3*	-	-
Jarosite Solution (g/L)	1.0	5.5	-	-	-	53
Oxidation Residue (%)	0.7	3.5	0.02	0.7	28.3	-
Oxidation Solution (g/L)	83.2	11.6	10.8	-	-	13
Purified Solution (g/L)	94.7	2.3	12.3	-	-	34
Return Electrolyte (g/L)	19.7	2.3	12.0	-	-	149

* S as (SO_4)

	S	Pb	Se	SEV*	Conductivity
Cathode Copper	10 ppm	3 ppm	0.2 ppm	400 mm	102% IACS**

* SEV - Spring Elongation Value (a measure of ductility)

** IACS - International Annealed Copper Standard

TABLE 3

COMPARISON OF ENERGY REQUIREMENTS

	kJ/kg Cu		
	Total Fossil Fuel	Total Electric Power	Net Total*
S-C Copper	17 410	13 570	30 980
Flash Smelting/Electrorefining With Acid Production	17 820	3 780	21 600

* Fuel Calorific Value - 37 300 kJ/m^3 (Natural Gas)
Electric Power - 3 600 kJ/kW.h

TABLE 4

SUMMARY OF OPERATING REQUIREMENTS

		REQUIREMENTS PER TONNE OF COPPER	
ITEM	UNITS	S-C COPPER WITH ELECTROWINNING	FLASH SMELTER/ ELECTROREFINERY WITH ACID PLANT
Fuel* (Natural Gas)	m^3 (std)	467	478
Hydrogen	m^3 (std)	240	-
Steam	kg	3000	-
Electric power	kW.h	3780	1050
Ammonia	kg	36	-
Operating Supplies (Chemicals, fluxes, etc.)	$	24	22
Packaging Supplies	$	6.1	6.1
Operating Labour Maintenance Labour and Supervision	Man Hours	7.3	10
Maintenance Materials	$	38	45
Overheads, Taxes,etc.	$	83	101

* Total fuel including that required for steam generation and hydrogen production.

TABLE 5

SUMMARY OF COST COMPARISON

ITEM	S-COPPER WITH EW	FLASH SMELTER ELECTROREFINERY WITH ACID PRODUCTION
Capital Costs (M$)	200	246
Working Capital Difference (M$)		+12.0
Operating Cost (¢/kg/Cu)	40.6	36.5

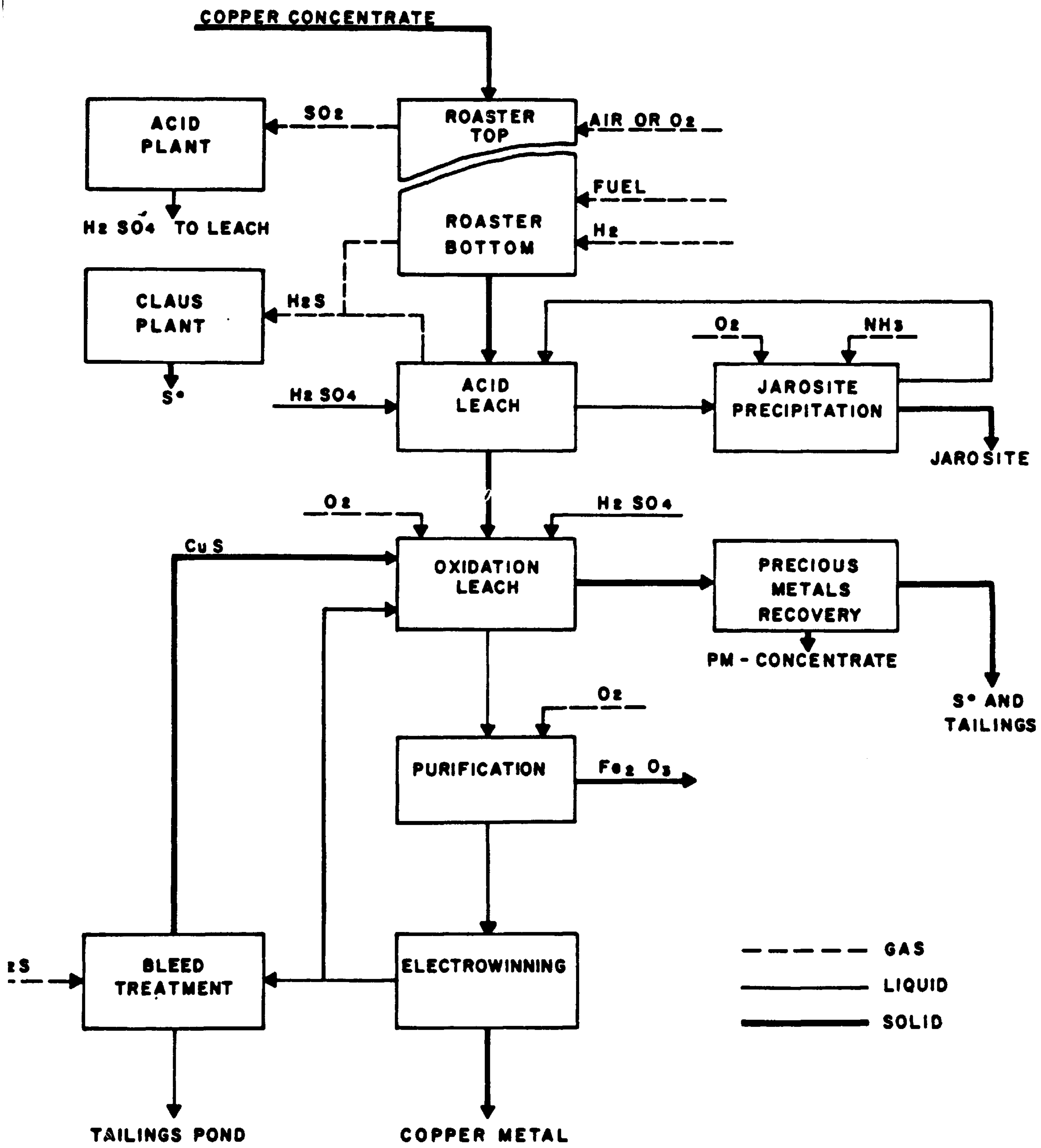

FIGURE 1 THE SHERRITT-COMINCO COPPER PROCESS SIMPLIFIED BLOCK FLOW DIAGRAM

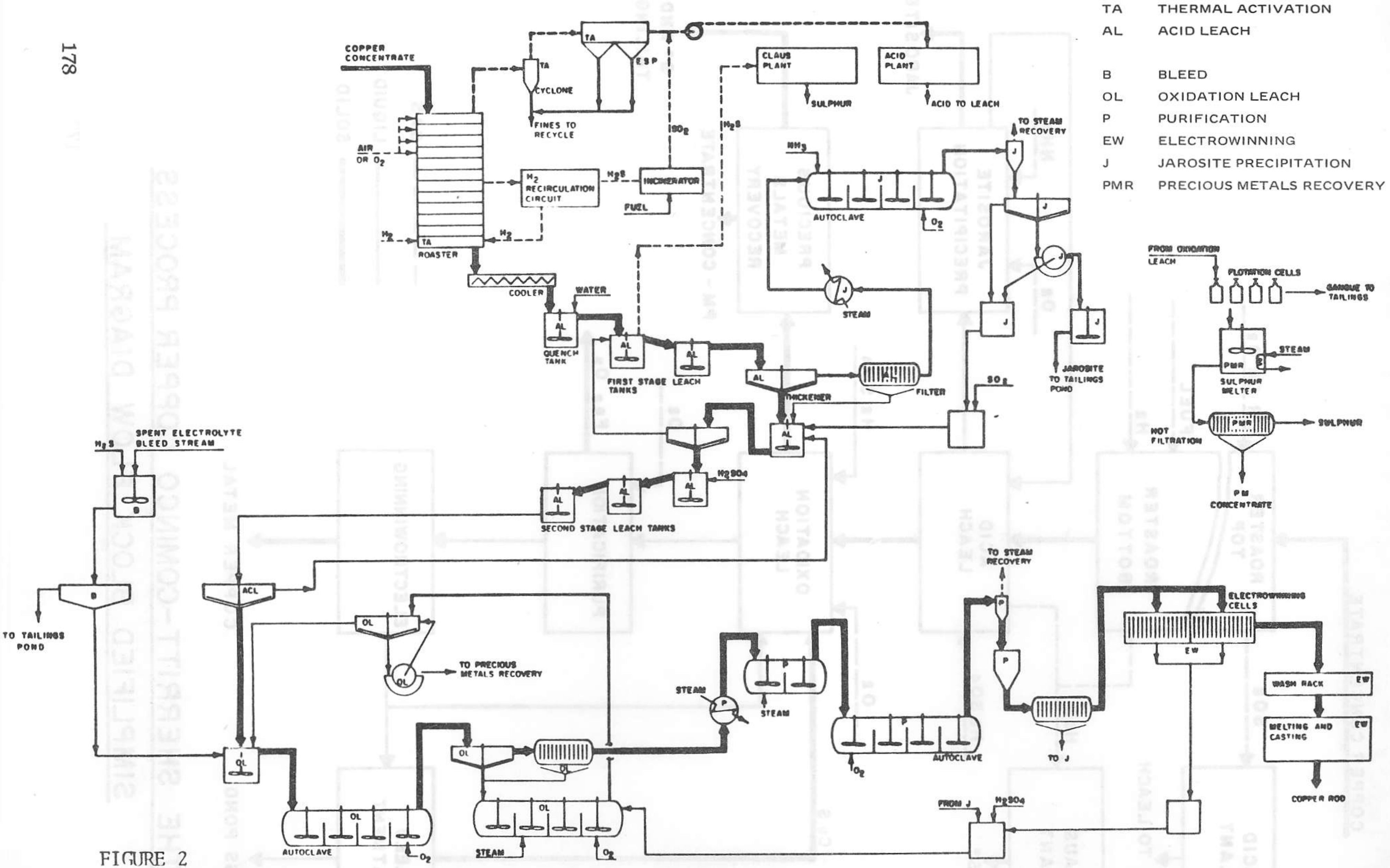

FIGURE 2

S-C COPPER PROCESS SCHEMATIC FLOW SHEET

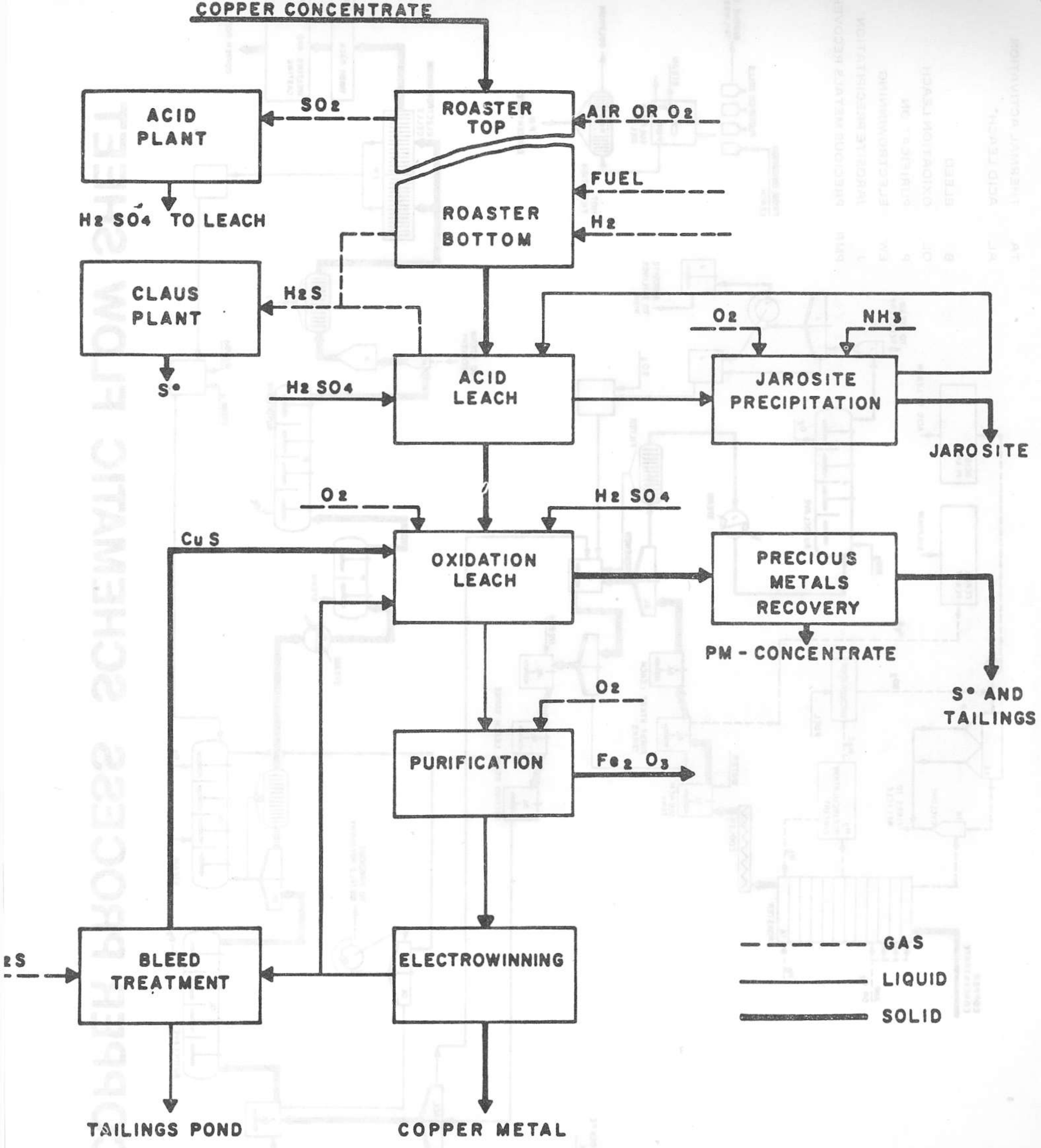

FIGURE 1 THE SHERRITT-COMINCO COPPER PROCESS SIMPLIFIED BLOCK FLOW DIAGRAM

SECTION VIII

HIGH EFFICIENCY SYSTEM FOR PYROMETALLURGICAL RECOVERY OF HIGH-GRADE COPPER MATTE FROM SULFIDE CONCENTRATE

BY D. A. MACKIE
P. PASCHEN
G. MELCHER

INTRODUCTION

KHD, Kloeckner-Humboldt-Deutz, is an engineering and manufacturing enterprise having its head office in Cologne, West Germany. Employee population is in the range of 25,000 world-wide.

The Humboldt Wedag Division of KHD, with a world-wide employee population of about 5,000, is the arm of the company involved with minerals processing. The company is not a metals-producer, however, but is engaged in direct selling of process technology, engineering, and equipment.

Principal factories are in Cologne and Bochum, West Germany. Fabrication of KHD designs is often done local to the user, however, as dictated by economic and other conditions of convenience.

North America is served by KHD Canada, Inc., Montreal; and Deutz Corp. in the U.S.A. with offices in New York, Atlanta, and San Francisco. The Single Machinery Division in Atlanta fabricates and assembles Humboldt Wedag designs.

HISTORICAL

Experiments with top-blowing of copper slag began in the laboratories in Cologne, treating 4 kg. batches, more than seven years ago. The results of these bench-scale tests led to construction of a pilot unit in Cologne that was capable of performing tests on a discontinuous basis on 1,000 kg. batches, proving the elementary principles of the KHD Contop (Continuous Top-blowing) process.

Two years ago, a continuous pilot unit was built in Bochum that was capable of smelting concentrates in a cyclone and treating slag by the Contop process. The unit handled 1,000 kg per hour on a 24-hour-per-day basis.

ABSTRACT

This paper presents a discussion of the results of continuous autogenous smelting of sulfide concentrates and top-blowing in the pilot plant in Bochum, by using technical oxygen in a water-cooled

cyclone followed by treatment in a top-blown reacting vessel. The products are high-grade matte containing up to 80% copper and a slag with a final copper content of less than 0.5%.

A description of a proposed commercial-scale plant planned for a customer in South America is given. The smelter is designed to produce 15 000 tonnes per year of blister copper economically, which illustrates the feasibility of constructing small smelters directly at the mine site. The paper further describes the internal workings of a commercial sized cyclone smelting and Contop unit.

CYCLONE SMELTING

In pyrometallurgy, cyclones meet the requirements for intensifying significantly mass and heat transfer processes by the specific aerodynamics of a turbulent two-phase flow. The generation of high temperatures accelerates the rate of reaction. The specific thermal load is by more than one decimal power higher than in conventional firing compartments:

About 30 GJ/m^3/h against 1 to 3 GJ/m^3/h. The specific smelting capacity too is about 10 times higher than in classical metallurgical furnaces: 100 to 250 kg/m^2/min against 2 to 50 kg/m^2/min.

KHD has acquired experience in construction and processing of cyclone furnaces for volatilizing antimony (1), treating residues from neutral zinc leaching (2), processing low-grade tin concentrate (3) and with the KIVCET process (4-6).

In the test cyclone of our pilot plant, oxygen is blown in tangentially. The solids are admitted either vertically or sucked in by oxygen from an injector, reaching the smelting compartment tangentially with the oxygen. Solids should be minus 3 mm in size and have a moisture content of 1% or less. The cyclone is built on the so-called water-vox principle or as a waste heat boiler.

The first molten charge solidifies on the cold wall, thus creating a protective layer. The molten product is discharged with the waste gas through the outlet opening in the cyclone bottom into a separating compartment.

This cyclone is a smelting unit characterized by easy operation. If necessary, it can be switched on and off within a few seconds. It does not require a specific mixture composition and handles most varying concentrates, either of pure or complex type. Changes in mixture composition can be performed without interruption of the process run. The same is true as regards changes in the matte quality.

The unusual flexibility of the cyclone is shown in Figure 1. For instance, a cyclone designed for smelting 500 tonnes per day of concentrate to a matte of 60% copper can be operated autogenously between 70% and 110% of its rated throughput at uniform matte quality. An even wider range of the rated throughput is possible by changing the matte quality. Similarly, the copper content in the matte can be varied between approximately 46% and 68% at an unchanged throughput of 500 tonnes per day.

A cyclone having a reacting vessel volume of no more than 1.6 m^3 can be operated with a specific throughput of 350 $t/m^3/d$. Copper mattes of 70% to 80% copper content have been smelted in the KHD test cyclone. Temperatures between 1800°C and 2000°C have been calculated for the cyclone on the basis of the thermal balance. In view of the fact that no solid magnetite is stable at such high temperatures, the slag leaving the cyclone has a copper content of no more than 6% to 8%.

SLAG TREATMENT

Although the necessary slag treatment can be carried out by conventional methods, such as flotation or in an electric furnace, KHD consider these too costly and ineffective for a high-efficiency system and have developed the so-called CONTOP process (continuous top blowing) for treating the slag after leaving the smelting cyclone. However, a CONTOP furnace could likewise be set up behind any other smelting furnace too.

For the CONTOP process a gas jet of high velocity is blown on the melt.

In Figure 2 the jet pulse is seen to change the shape of the bath surface deflecting the gas stream in such a way that it causes rotation of the melt, whereby the reacting surface is continually renewed.

For applications where not only a defined mass transfer but also a defined heat transfer is of importance, as in the case of volatilization, for example, the high velocity gas jet can be enveloped by a slowly flowing heating gas jacket. Figure 3 shows this type of blowing lance where the reducing high-velocity gas jet is surrounded by several slowly flowing heating-gas streams. Figure 4 is the continuation of the process towards the bottom. The strongly narrowed high-velocity gas jet impinges on the melt surface, changes the surface shape, and has the melt move.

It was investigated how the degree of oxidation of the slag, i.e. the $Fe^{2}+/Fe^{3}+$ ratio, which is a measure of the slag fluidity, can be controlled by this top-blown process. Figure 7 shows that a conventional slag was first treated in an oxidizing atmosphere and the reaction gas jet then changed to a reducing atmosphere. This reducing treatment was continued until a $Fe^{2}+/Fe^{3}+$ ratio had been set that corresponded to the oxygen potential of the gas stream. Figure 8 shows the extraction of copper from the slag by top-blown reducing gases.

The Fe^{3}/Fe_{tot} ratio ascertained and the corresponding copper content of the slag as a function of the lambda figure of the reaction gas have been plotted in Figure 9. It turns out that the copper content of the slag falls below 0.5% for lambda figures of less than 0.45.

PILOT PLANT

The KHD pilot plant in the Bochum works combined both methods, the cyclone smelting and the contop-blowing, to a high-efficiency system: Figure 10.

Concentrate and additives are fed into the cyclone (1) where they react with pure oxygen. The molten products are discharged with the waste gases downward into a settling chamber (2) where the three components are separated from each other. The slag is transferred continuously into the CONTOP reacting vessel (3) for reduction. The two furnace sections are separated by a partition wall (8).

The lances had to be located parallel to the wall due to space limitations. The partition wall had to be incorporated as a guide element to make sure that the complete slag quantity passed successively underneath all three lances.

Table 1 specifies the results of the continuously operating pilot plant. At a throughput of 1 tph of concentrate containing 25% Cu, 30% Fe, and 33% S, 325 kg/h or matte of 72% Cu have been smelted at an oxygen consumption of 240 m^{2}ph. The slag contained 1% of copper, that value however could have been lower with a larger settling area that had not been available due to lack of space. The copper content of the slag to be dumped can be reduced to minus 0.5% by enlarging the settling zone.

PROPOSED 15 000 TONNES COMMERCIAL-SIZED PLANT

Recently, KHD prepared a quotation for a commercial-sized plant for a customer in South America. This plant was designed to produce approximately 15 000 tonnes of blister copper: Figure 9. Moist flotation concentrate and the additives ground to minus 1 mm are fed continuously into a rotary dryer and dried to a residual moisture of 1%. Hot air discharged from the waste-gas cooler of the smelting furnace is used for drying the concentrate. After the addition of recycled precipitated dust, the dried charge is sent to the cyclone where it is oxidized with technically pure oxygen. The cyclone is 1.0 m high and has a diameter of 0.8 m.

Although the structural design of the CONTOP furnace having a surface of 30 m^2 in this plant differs in several details from the pilot installation described, it is basically the same. An adequate stabilizing and settling zone has been provided between top-blowing lances and matte tapping.

The waste gas, which is discharged from the furnace jointly with the non-burned components of the reducing gas is subjected to secondary combustion by admixing cold air. An indirect air cooler has been provided for cooling the waste gas. A waste heat boiler can be provided as well. The slag to be dumped is allowed to leave the furnace at the front side and is then granulated. The matte is removed from the furnace through a siphon taphole and directed to the converting vessel through a launder; the converting vessel is also operated by the top-blown process. It is intended to use technically pure oxygen as an oxidation agent. The absence of nitrogen ballast will enable the conversion of converting high-grade matte where oxidation and slagging of the iron has only little influence on the thermal balance, without difficulties with respect to temperature control.

The waste gases discharged from the two furnaces are combined and contain approximately 43% SO_2. They are subjected to dust collection in an electrostatic precipitator and are then sent to the sulfuric-acid plant.

CALCULATED DATA FOR A 50 000 TONNES COMMERCIAL PLANT

Tables 2 and 3 quote the consumption and production figures per tonne of concentrate for the production of 50 000 tonnes anode copper.

The cost of investment including all steps from concentrate drying to cast anodes with buildings, civil works, oxygen and sulfuric acid plant, erection cost and start-up will be in the order of $1,000.00 US per tonne yearly capacity.

Hence the combination of cyclone and top-blown technique permits a pronounced shift of the break-even point towards small annual capacities. Consequently, the prospect of a smelter next to the mine should become feasible for the first time.

Figure 10 shows the new furnace combination.

SMELTING IN THE COMMERCIAL UNIT

After drying of the moist concentrate in a rotary dryer, it is stored in a bin, and transported via airlift to the feed hoppers, where the concentrate, together with quartz and technically pure oxygen is delivered to the cyclone. Oxygen and concentrate are so well mixed together, that a concentrate in the cyclone containing as high as 79 - 81% Cu could be smelted. In this manner, almost all of the iron and the sulfur is oxidized in the cyclone. The heat released in the process leads to a cyclone furnace chamber load of more than 21 million $KJ/m^3/h$. Gas and fused mass reach a temperature of over 2100°C. This temperature, however, is too high for the brick lining. Cooling of the gas/fused mass mixture in the cyclone is necessary.

In the pilot plant, it proved to be practical to keep the fused mass/gas mixture at a temperature of approximately 1600°-1700°C at the discharge from the smelting cyclone. This was done by water-cooling the cyclone. A practical method would be to use the cyclone as a boiler. Metallurgically, the high temperature inside the cyclone produces greatly improved thermodynamic conditions for the smelting process, so that a high volatilization of the lead, zinc, and arsenic contained in the concentrate takes place.

SEPARATION OF GAS AND FUSED MASS

The fused mixture and gas are separated in the furnace chamber of the primary smelting chamber under the smelting cyclone. The hot smelted mixure is collected on the floor of the collecting shell and flows into the slag treatment chamber.

SLAG TREATMENT AND REMOVAL OF COPPER MATTE

The slag treatment is conveyed through water-cooled oxygen lances to the slag tank surface by the introduction of reducing gases (2). This reduction operation works in 3 ways:

(1) Reduction of the magnetite still existing in the slag from approximately 10% to 2% with a velocity of $k - 8 \times 10^{-4}$/sec.

(2) The remaining reducing volatilization of zinc, and

(3) The remaining volatilization of lead sulfide and lead oxide.

After the reduction, the disposed slag shows a composition of:

CaO	5.0%
FeO	60.0%
Fe_3O_4	2.0%
SiO_2	28.4%

Al_2O_3	2.5%
Zn^0	1.5%
Cu	0.6%

For control of this slag, no lime admixture is necessary unless some silica is added. This slag shows a viscosity of below 0.5 P at a cut-off temperature of approximately 1350°C and under those conditions is best suited for disposal of granulated slag. The matte has a composition of approximately 81% Cu, as well as 0.4% Fe and sulfur as the balance. The waste gases from the cyclone and the slag treatment are conducted out of the hearth oven. At a temperature of approximately 1400°C, the waste gas shows a composition of:

10%	CO
2%	H_2
6%	H_2O
72%	SO_2
5%	N_2
5%	Zn

For smelting capacity of 500 tonnes concentrate per day, the cyclone has an inside height of approximately 2.2 m and a diameter of 1.6 m. The primary smelting chamber dimensions underneath the cyclone are 13 m by 6 m, inside. The inside height of the primary smelting chamber is 3.2 m at the vortex. The number of lances for the slag reduction is 10.

CONVERSION AND REFINING

The copper matte flows continuously through a launder to a closed trough in the converting and refining oven (3). At approximately 1350°C, the temperature of the matte is so high that an additional heating of the closed trough is not necessary. Referring to Figure 13, furnace chamber 3a has a hearth area of 12 m^2, where the white metal is blown into blister copper by the blowing of an air-oxygen mixture. Because of the small iron content, no formation of slag takes place in the white metal. For the most part, the small iron mass as well as the existing zinc and lead are volatilized due to the high temperature in the dimple; a small amount remains in the refined copper. 12 lances are needed for the conversion and oxidation work. The conversion process is conducted in such a way that the stationary copper mass in the conversion chamber shows an oxygen content of over 1%, and the white metal flowing in sometimes reacts directly with oxygen gas, but sometimes reacts with the oxygen released from the copper. In connection with the conversion work, a further oxidation of the copper to 1.5% O_2 takes place in the oxidation chamber (3b). Through blowing of the reduction gases in the reduction chamber (3c), the oxygen content is lowered to 0.2% O_2 in copper. Total lances are 12. The reduction constant k for the O_2-reduction is $k = 2 \times 10^{-3}$/sec. The hearth area needed for the reduction is 14 m^2.

The expected copper yield in the anode furnace is 98.8%.

ACKNOWLEDGEMENT

The authors would like to thank the German Federal Ministry for Research and Technology for the financial promotion of the tests carried out for slag treating.

REFERENCES

(1) E. Mueller: Die Verfluechtigung von Antimon aus sulfidischen Vorstoffen im Zyklonofen; Erzmetall 31 (1978) 267-275.

(2) R. Nissen, I. Lema-Patino: Die Verarbeitung von Zink-Neutrallaugungsrueckstaenden im Schmelzzyklon; Vortrag auf der GDMB-Hauptversammlung in Innsbruck, Mai 1978.

(3) E. Mueller: Verfluechtigung von Zinn aus armen schwefel-haltigen Konzentraten in Zyklonofen; Erzmetall 30 (1977) 54-60.

(4) N.N.: Kivcet Process for complex ores offered to western smelters; World Min. (1974) June 26-27.

(5) G. Melcher, E. Mueller, H. Weigel: Entwicklung, Grundlagen und Anwendung des Kivcet-Zyklonschmelzverfahrens mit reinem Sauerstoff fuer verunreinigte Kupfererze; Erzmetall 28 (1975) 313-322.

(6) -, -: The Kivcet cyclone smelting process for impure copper concentrates; JOM (1976) July, 4-8.

(7) F. Sauert et al.: High-efficiency system for pyrometallurgical recovery of high-grade copper matte from sulfide concentrate; Erzmetall 33 (1980) 371-377.

TABLE 1

PILOT PLANT TEST DATA

Input		Output	
Concentrate	1000 kg/h	Matte	325 kg/h
- Cu	25 %	- Cu	72 %
- Fe	30%	- Fe	5 %
- S	33%	Slag	660 kg/h
Silica	110 kg/h	- Cu	1 %
Limestone	60 kg/h	- Fe	43 %
Oxygen-cyclone	240 m^3/h*	- SiO_2	28 %
Propane	26 m^3/h*	- CaO	9 %
Oxygen-Contop	64 m^3/h*	Cyclone dust	48 kg/h
		- Cu	13 %
		Cyclone waste gas	270 m^3/h*
		-SO_2	67 %
		Contop dust	7 kg/h
		Contop waste gas	200 m^3/h

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* standard state

TABLE 2

CONTOP FURNACE OPERATING DATA

Concentrate Analysis

Cu	23.5%
Fe	30.7%
S	32.0%
SiO_2	5.2%
CaO	0.9%

Feed

Concentrate	182	TPD	
Silica Sand	29	TPD	
Limestone	18	TPD	
Oxygen, 95% O_2	215	Nm^3/T	1)
Propane	7	Nm^3/T	1)

Output

Matte	58	TPD	
Slag	129	TPD	
Offgas	215	Nm^3/T	1,2)

1) As mixed
2) Before afterburning

TABLE 3

CONSUMPTION AND PRODUCTION FIGURES PER TON CONCENTRATE FOR THE PRODUCTION OF ANODE COPPER

Silica	127 kg
Limestone	-
Oxygen	317 Nm^3
Natural gas	37 Nm^3
Matte (81% Cu; 0.4% Fe; Rest S)	345 kg
Cyclone-slag (61% FeO; 2% Fe_3O_4, 28% SiO_2; 6% CaO; 0.5% Cu)	532 kg
Converter slag	-
Slag anode refining (20% Cu)	12 kg
Waste gas cyclone-converting reactor (43% SO_2)	550 Nm^3
Steam Production (43 Bar, 300°C)	4500 kg

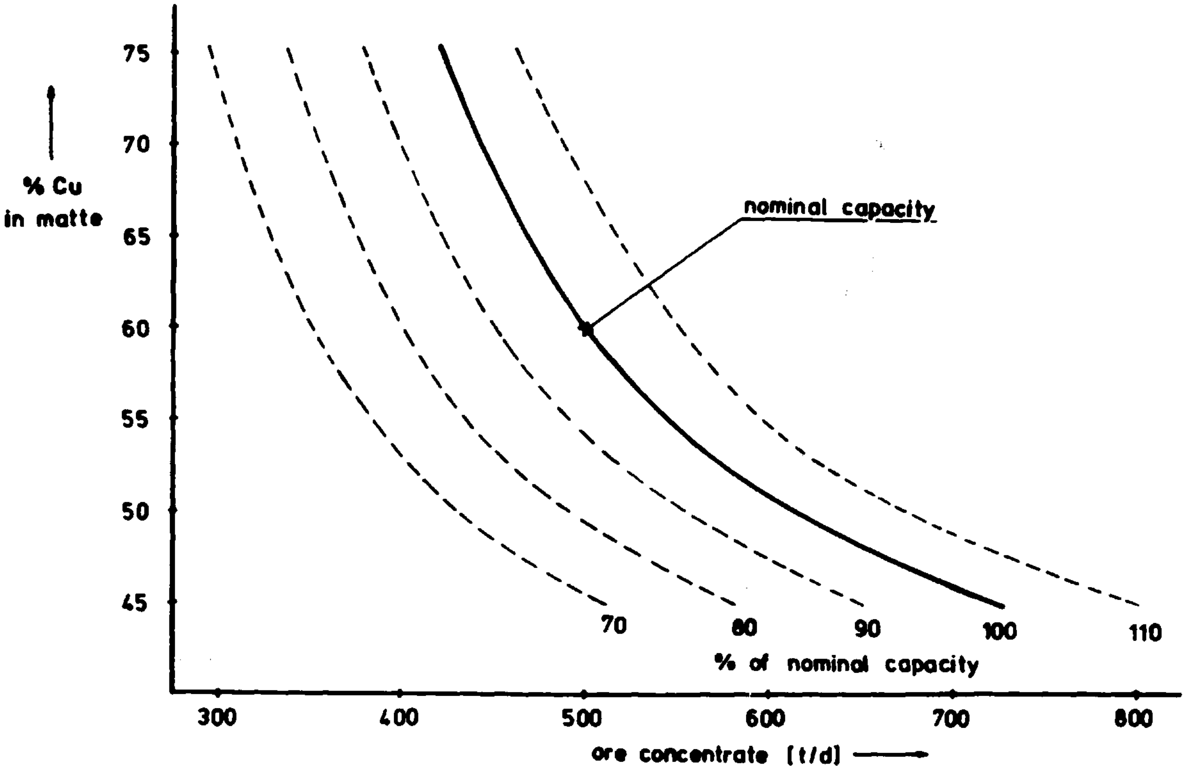

FIG. I Operating range of a cyclone.

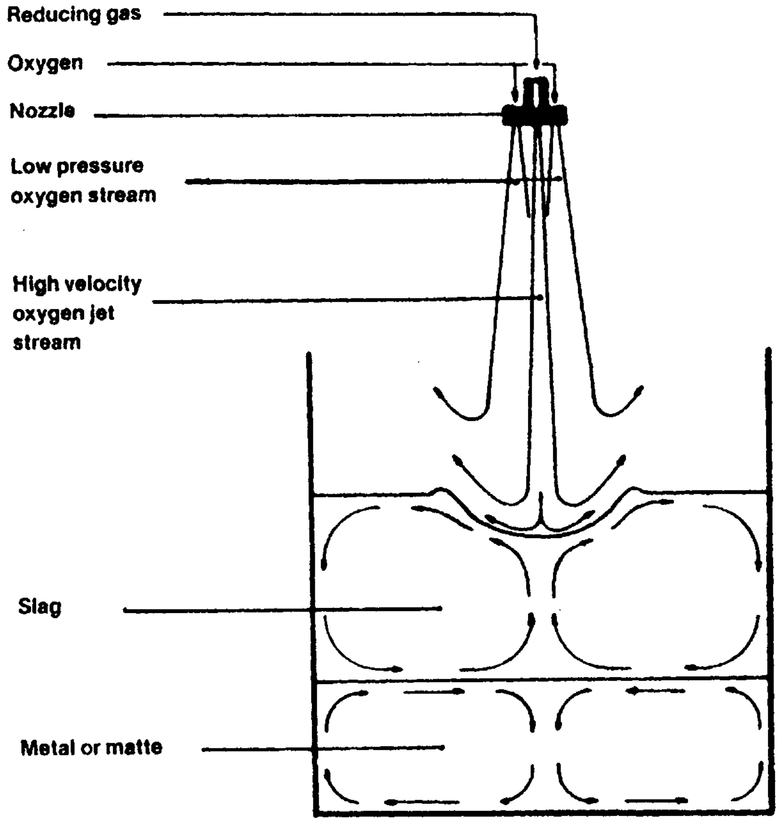

FIG. 2 Top blowing system for recovery of metal values from slag.

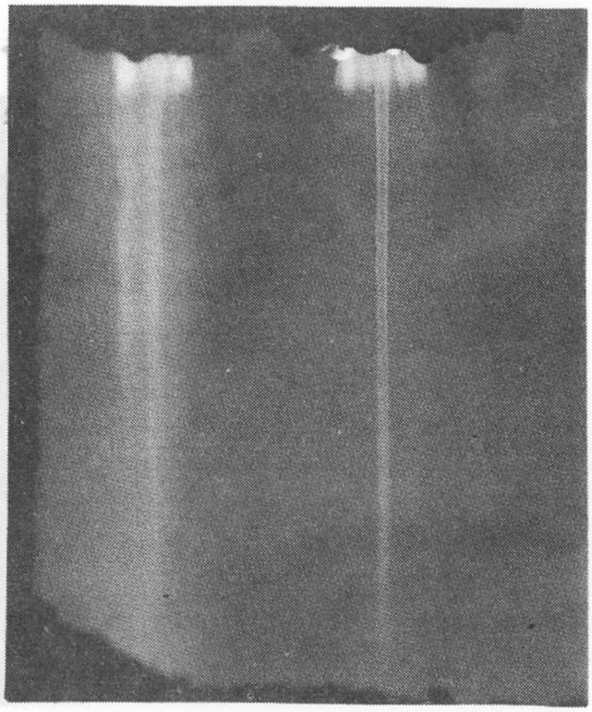

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FIG. 3 View of a top-blowing lance with a high-velocity reaction jet and low-velocity heating-gas streams.

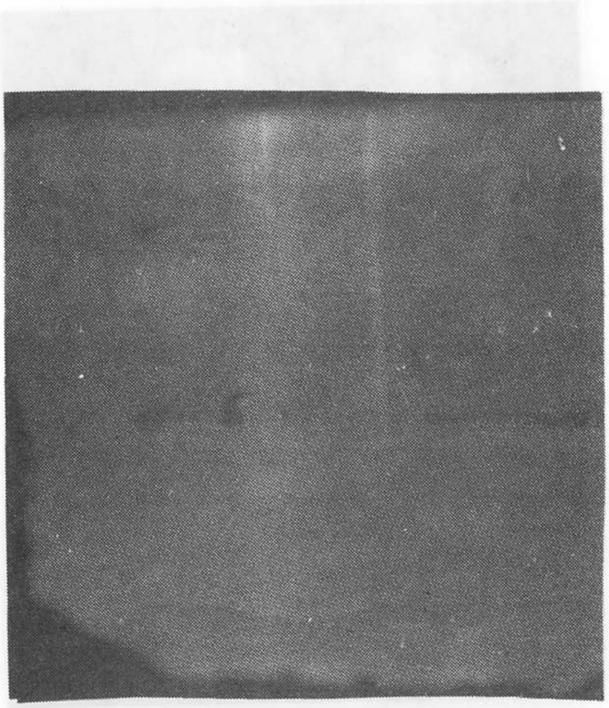

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FIG. 4 View of a dimple in the slag in a top-blown reacting vessel.

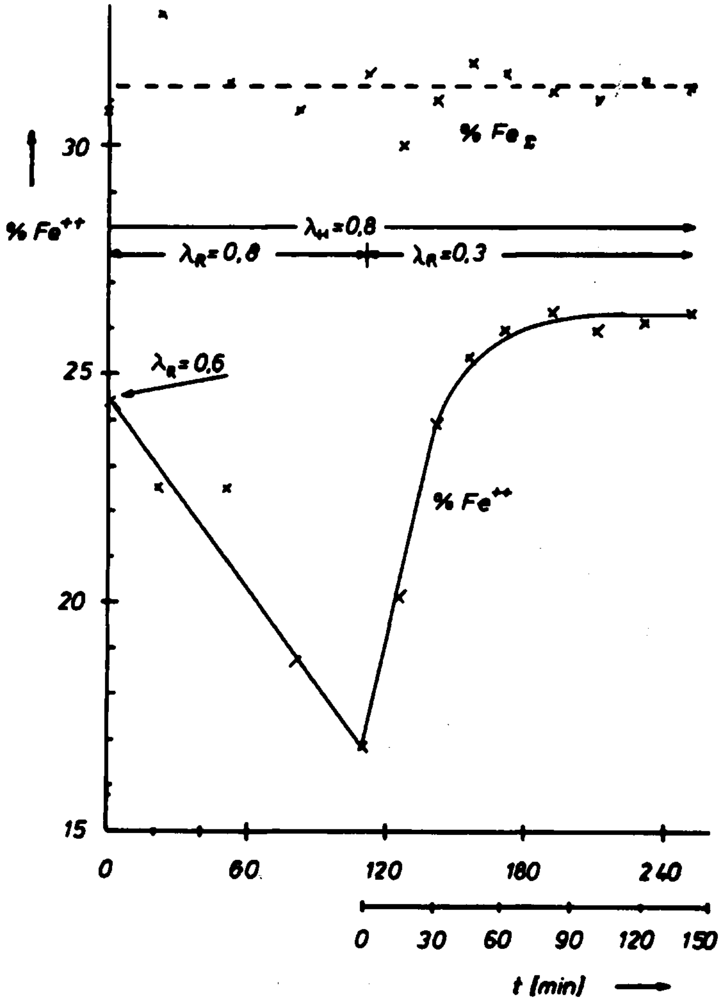


FIG. 5 Oxidizing and reducing $Fe^{+++} \longleftrightarrow Fe^{++}$ in copper slag by the top-blown process.

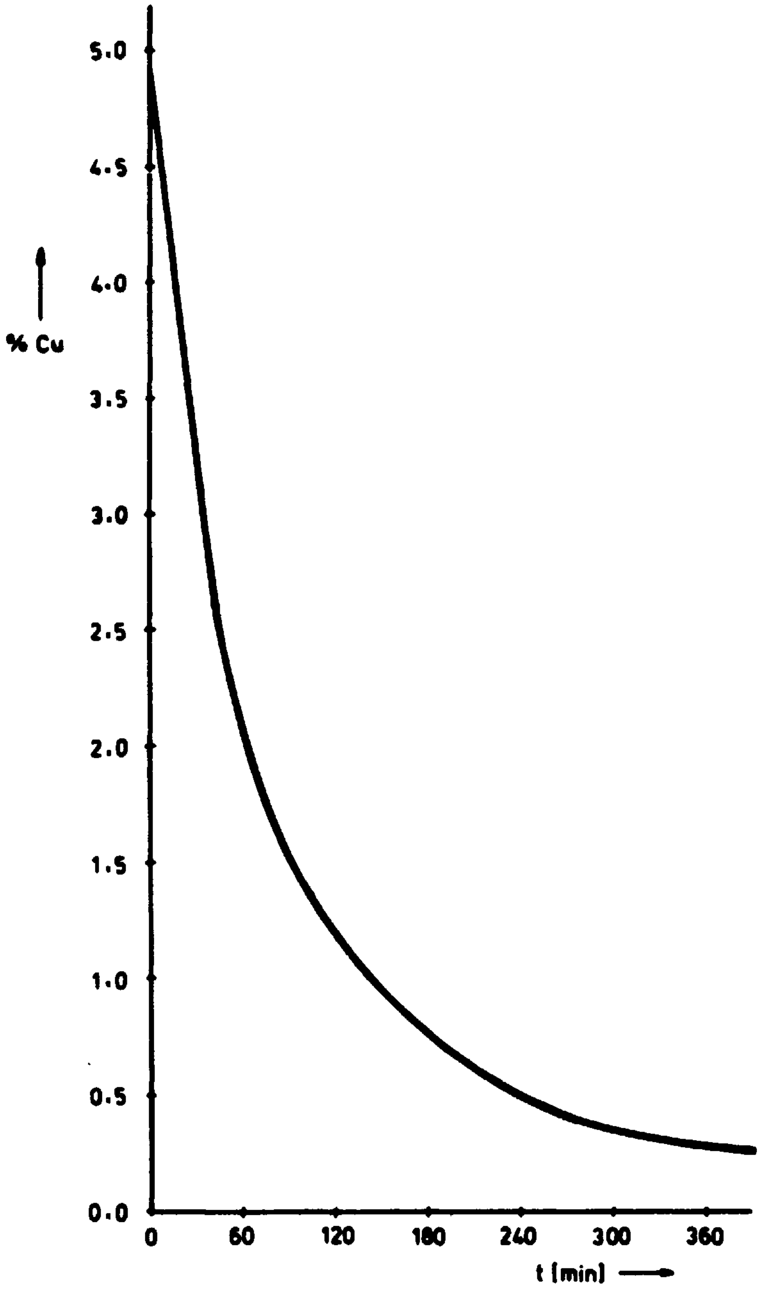


FIG. 6 Copper content in Slag when reducing by the top blowing technique.

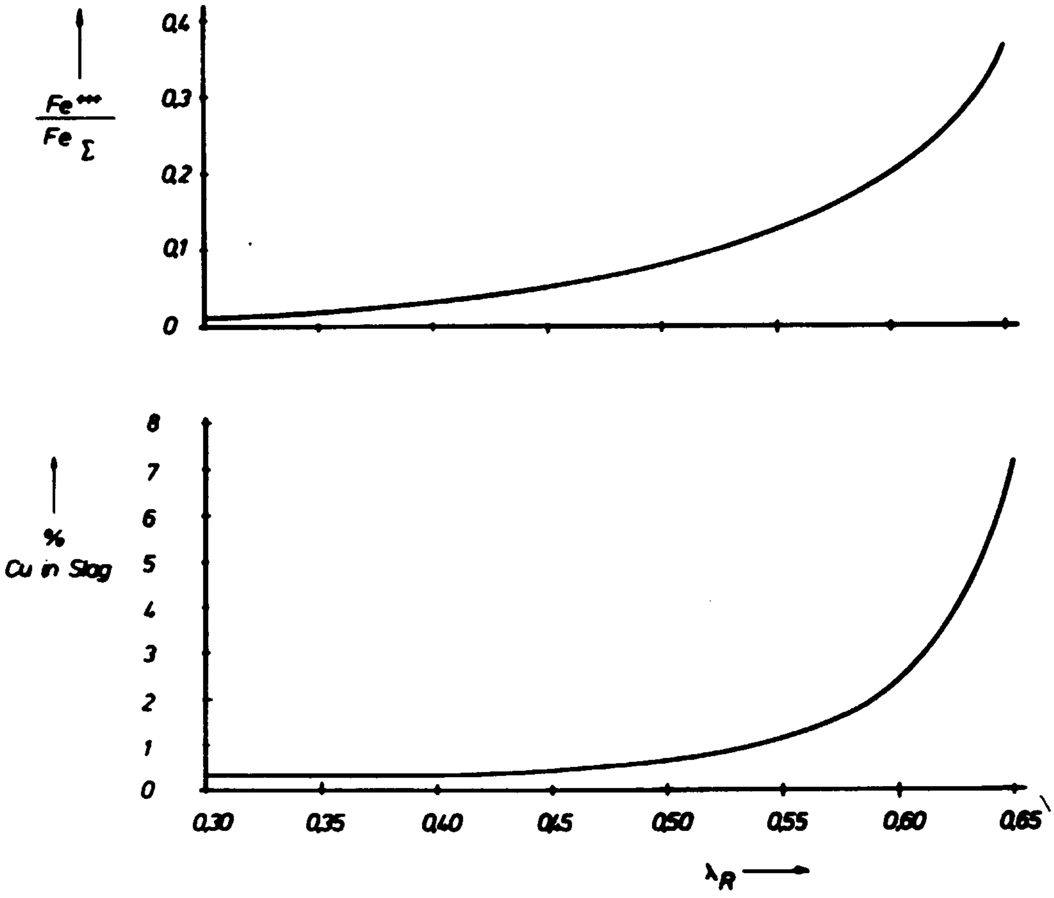
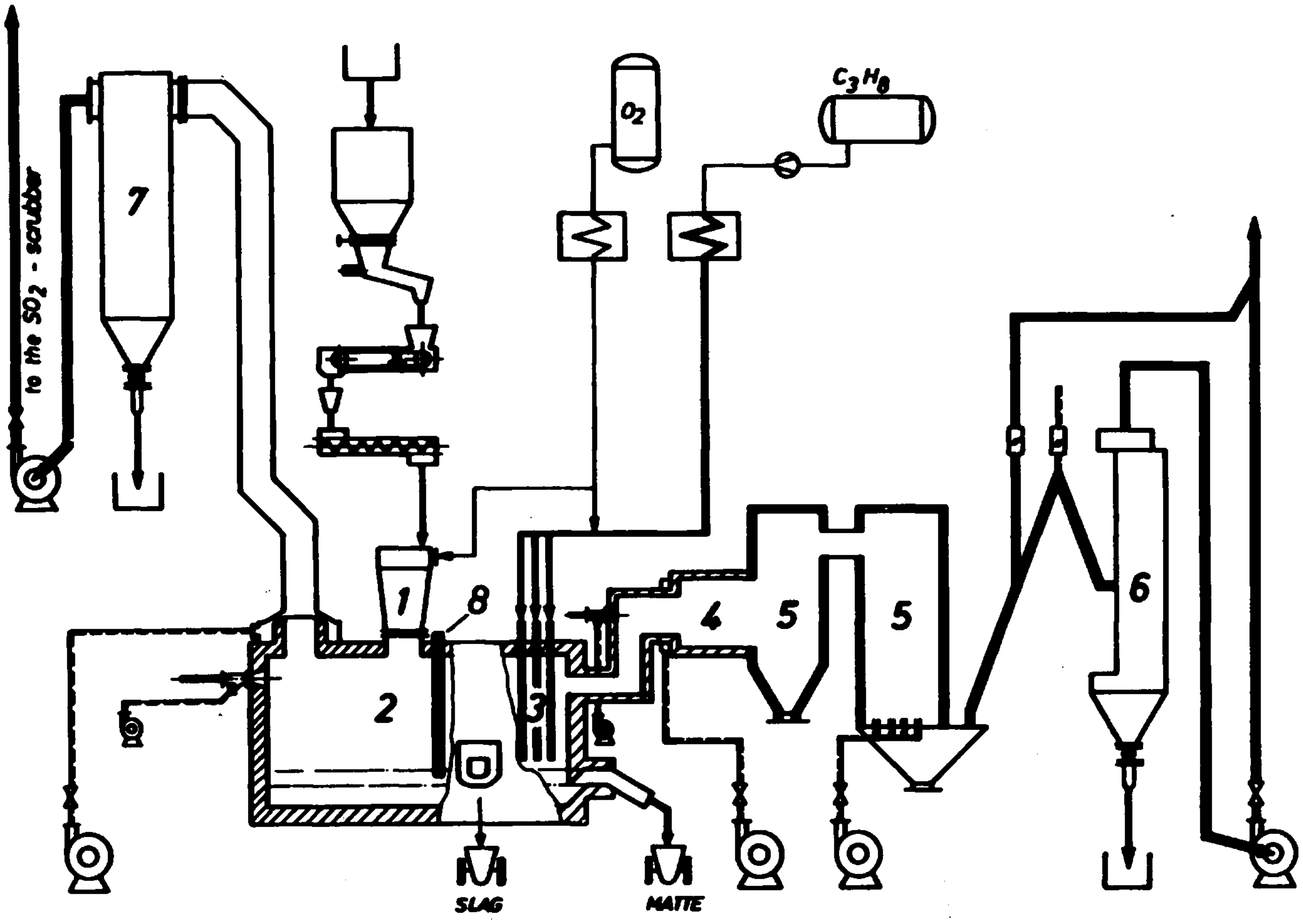


FIG. 7 Ratio Fe^{+++}/Fe_{tot} and wt-% Cu in slag as a function of the jet composition λ_R



1. Cyclone
2. Settling chamber
3. Top - Blowing - Reactor
4. After burning chamber
5. Cooler
6. Bag house
7. Multi - Cyclone
8. Partition Wall

FIG. 8 Pilot Plant
Cyclone furnace with top-blown reacting vessel.

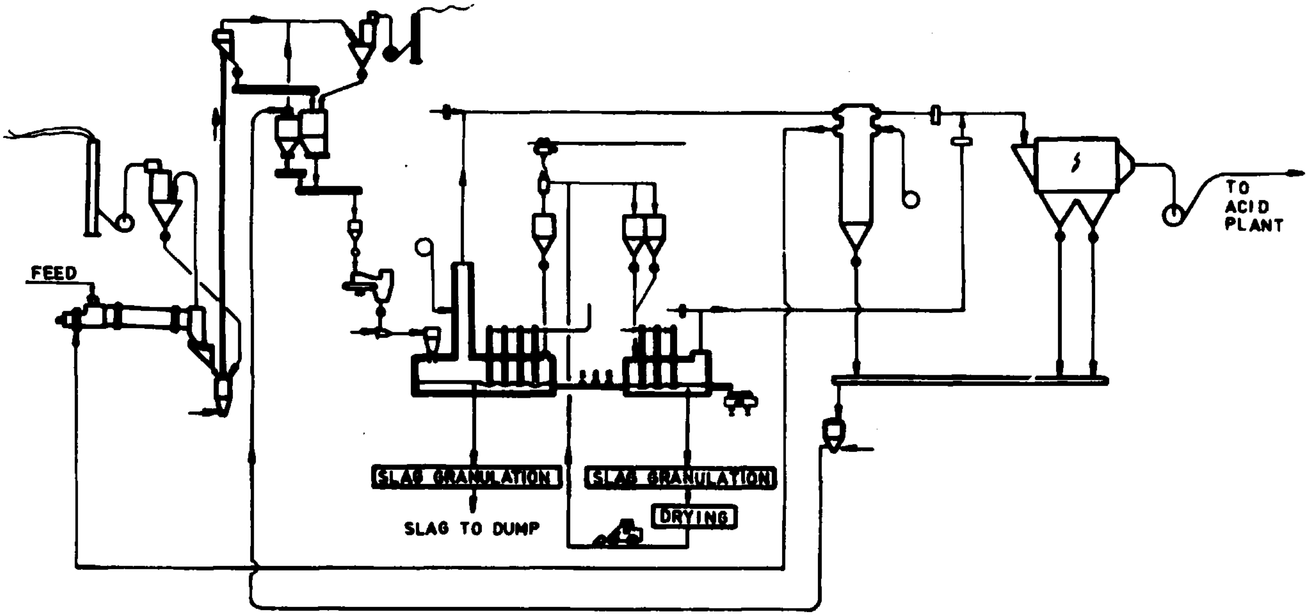


FIG. 9 CONTOP - Plant for the production of blister copper.

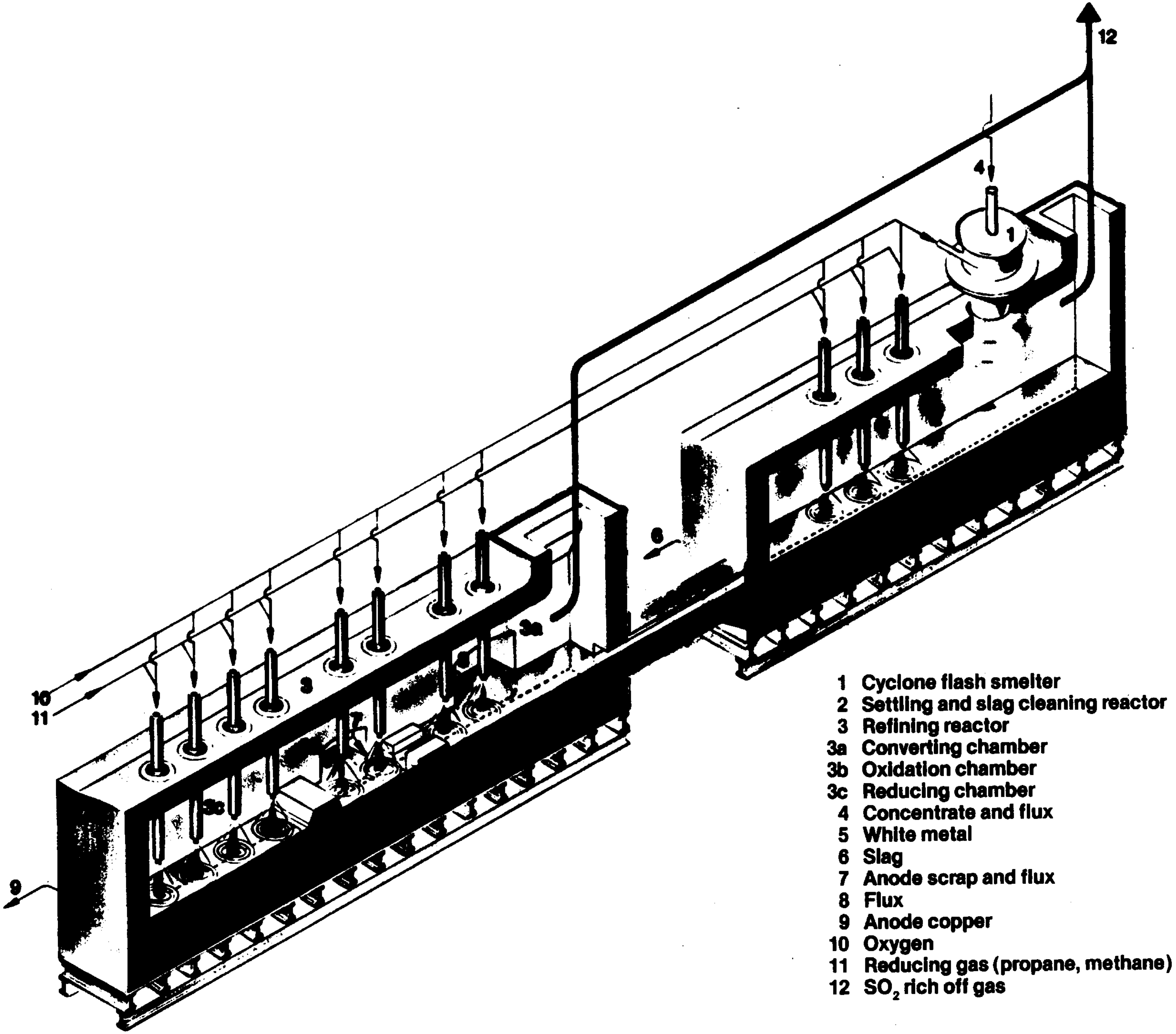


FIG. 10 Section through a Commercial Cyclone-ConTop Unit.

SECTION IX

TOP BLOWN ROTARY CONVERTER (TBRC™) PROCESS FOR NON-FERROUS SMELTING, Q-S OXYGEN PROCESS, OXYGEN SPRINKLE PROCESS

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INTRODUCTION

Three copper smelting processes are discussed in this paper:

(1) TBRC^{TM} Process*
(2) Q-S Process
(3) Oxygen Sprinkle Smelting Process

The TBRC process is currently available technology. It is utilized in the first copper smelter in modern times in British Columbia. The Afton Mines Ltd. smelter is located 16 KM west of Kamloops, B.C.

The Q-S process is one to be considered in the long term. It is a single vessel, continuous process, for the late 1980's.

The Oxygen Sprinkle smelting process has been devised to transform existing reverberatory furnaces to oxygen flash smelting units. Although it does not have application in B.C., it is presented in the context of a step along the way in the development of the Q-S process.

TBRC^{TM} PROCESS

The TBRC^{TM} Process for non-ferrous smelting is an amalgamation of mechanical equipment and process tehnology. Both are evolving and improving since its first use for nickel in 1972. The result is a stirred reactor providing rapid mixing for high reaction rates and high heat transfer rates in a controlled atmosphere.

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* Dravo Corp. is the worldwide licencee from Inco Metals Co. for the TBRC for treating copper materials.

MECHANICAL EQUIPMENT

Figure 1, TBRC Arrangement, is a cross-sectional view of the TBRC. It is basically a refractory lined vessel encased in a steel shell. Attached to the steel shell are cast steel track rings. The track rings and shell are supported in a vessel support frame. Within the frame, motor-driven support rollers are held. These driven rollers, two pair on each side of the vessel, then rotate the cast steel track rings. The TBRC vessel rotates at infinitely variable speeds from 0 to 40 RPM. The support also has a thrust roller to take the downward load from the furnace.

A tight-fitting hood, which is designed to minimize inleakage of ambient air, closes over the vessel mouth against a sealing ring. It has a water-cooled charging chute, and water-cooled process lance.

The equipment is a key part of this process. The following are a few of the steps in the continual development of vessel improvements as used for non-ferrous processing. The first vessels at INCO's Copper Cliff facility had the cast steel running rings welded to the steel shell. These vessels also had flat bottoms. The flat bottom, which is removable, has never been removed. The original idea of the removable bottom was to make it easier to replace the refractory. As it was never used, this particular feature has been eliminated in more recent models. The second generation units were installed in Indonesia and Afton Mines Ltd. represents the third generation.

One of the changes to note in the third generation vessel is that the flat bottom is no longer used. It has been replaced with a dished bottom, which gives higher strength at lower cost. The vessel support system is somewhat different than the previous ones in that it is less bulky. Another item of note is the attachment of the running rings to the steel sheel. No longer are they welded to the shell, but are now attached through 12 pins and a set of springs. These advances, like all technology, are oriented toward solving the problems that have occurred in the past. Each new unit has less problems than the previous ones.

The entire vessel is supported on concrete pedestals through trunnions and pedestal supports. A spur gear rotates the vessel through 360° in either direction.

PROCESS TECHNOLOGY

The second part of the amalgamation in the TBRC^{TM} Process is the process technology. This technology is attributed to three key features.

(1) Atmosphere Control

(2) Temperature Control

(3) Turbulence Control

Atmosphere control is obtained in several manners. First, the process lance is a surface blown lance. It is not submerged below the bath; therefore, it allows the use of high levels of oxygen without the problems of deterioration due to temperatures such as one would experience in the Pierce-Smith converter. The TBRC hood with the process lance and the charging chute, both of which are water-cooled closes over the vessel mouth. The atmosphere inside the vessel can be oxidizing, reducing, or neutral. This flexibility eliminates sulfur and iron when oxidizing, decomposes copper oxide when reducing and volatilizes impurities when neutral atmospheres are used. The atmosphere features provide a degree of flexibility in processing not readily available in other equipment.

Table 1 reflects an example of impurity elimination. This table reflects the full scale testwork on Craigmont, B.C. concentrate performed at the INCO TBRC facility in Copper Cliff, Ontario. Although the quantity of these metals: arsenic, lead, bismuth and selenium are very low, the TBRC still did an excellent job removing them. Over 90% of the arsenic, and the lead, and over 38% of the bismuth and the selenium were removed between concentrate and blister copper. The best eliminations are obtained at temperatures controlled above 2500°F and at high RPM.

Another feature of the TBRC^{TM} Process is selective converting. That is the ability to have a liquid/liquid extraction of precious metals from a molten copper sulfide phase into a phase. Basically, when the finish blow begins in the converting cycle with copper sulfide, 10-15% of the copper sulfide is oxidized to copper metal. At that point, using a neutral or slightly oxidizing atmosphere, the vessel is rotated at high speed.

The copper in this liquid-liquid mixing of sulfide and metal phases extracts the precious metals from the copper sulfide phase. This high grade precious metal copper could be removed and sent to an electrolytic refinery for further processing. The remaining copper sulfide could be converted into a fire-refined copper and marketed directly.

In summary, atmosphere control, is the result of being able to surface blow and adjust the oxidizing, reducing and neutral conditions inside the TBRC vessel.

The second key feature is temperature control. All of these features overlap to a great degree. The temperature can be controlled by blowing oxidizing gases from air to oxygen; by oxy-fuel combustion; and by keeping refractory bath and atmosphere in equilibrium. The correlation between temperature in the molten bath and refractory is very close. Use of optical and immersion pyrometers to check the bath and refractory wall temperatures has shown that the temperature of both correlate within 10°F. This permits the use of optical pyrometers to check the process temperatures.

For oxy-fuel combustion the TBRC has the ability to burn natural gas or liquid fuel. This is primarily for smelting but raising the temperature of the bath is advantageous, for example, in impurity elimination.

Temperature is also controlled through exothermic heat generation control, in particular the burning of sulfur and iron in the material charged. The ability to blow a full range of oxygen levels as opposed to air or slightly enriched oxygen in conventional equipment allows the TBRC to operate at predetermined temperatures. One of the most recent advances in the TBRC process development has been the application of the flash smelting technique to the TBRC. This was piloted at the Port Colborne test facility of INCO. This work, "Flash Smelting in a Converter," by Dr. Malcolm Bell was published in the Journal of Metals in 1978. This allows the TBRC the ability to produce controllable matte grades and clean slag similar to the INCO flash furnace.

A third item on temperature is the refractory-bath-atmosphere equilibrium. Along with being controlled, the rotation of the vessel brings both the bath and the refractory into temperature equilibrium. Rotation also allows the phases to be mixed in the vessel. During the copper blow there is a thin layer of residual slag, white metal (Cu_2S),and the copper. As the vessel rotates, droplets are carried up the side of the vessel and fall through the gaseous atmosphere. This then permits control of oxidation rates to a much higher level than if gases were bubbled through the bath. This gives good contact between the phases. The phases, due to different densities, move at different rates therefore the reaction point is concentrated at the phase interface.

In summary, temperature is controlled through either oxy-fuel combustion, exothermic heat generation, or transmittal through refractory into the bath by the atmosphere equilibrium.

The third key feature is turbulence. Turbulence is controlled by both the variable speed of the vessel and process lance penetration. Rotational speeds are selected depending on the chemical process reaction desired. During charging, relatively low speeds are used. During de-oxidation, when copper reduction is required, high speeds are used. In addition when the constituents to be reacted are present in low concentrations, the high speed allows these to react at faster than conventional process rates, e.g. up to 40 RPM. There are virtually no emissions into the work environment when the process is being carried on even at high RPM.

The other item affecting turbulence is the lance jetting. The process lance releases its gaseous products at speeds greater than Mach 1. It is these high velocities which result in bath penetration. This penetration creates waves which add to the rotational turbulence.

In summary, turbulence is effected by both lance jetting and variable speed rotation.

When analyzing the TBRC processing capability and its ability to control these key features: atmosphere, temperature, and turbulence, the result is a capability not obtained in conventional equipment. It's reflected in Figure 2 which is a graph of "copper in slag" versus the "percent copper in the matte". Due to the high reaction rate ability of the TBRC, it is possible to shift the normal curve where in the slag an increase in copper matte grade results in a related in copper slag. The shaded area represents the matte slag area during the initial full-scale testing of the TBRC^{TM} at INCO.

In order to develop the process parameters of the Afton Mines Ltd. facility, pilot plant work was undertaken at Port Colborne. One of the items in the pilot plant effort was the determination of the slag cleaning ability in the TBRC^{TM}. Four tests demonstrated the ability of the TBRC to clean slags. Typical values were: 67% copper in the matte, 0.65% copper in the slag; 78% copper in the matte and 1.2% copper in the slag.

At Afton one of the key elements of the TBRC process is that it made a mini-smelter possible. Historically, it has been a general concept that smelters had to be large tonnage facilities to be economical. With the advent of the TBRC^{TM}, single vessel mini-smelters were technically, environmentally, and economically possible. A smelter can be built to treat as little as 100 tonnes per day concentrate, or 8-10 000 tonnes per year copper economically, using a single vessel. At Afton Mines Ltd. the TBRC process provides an excellent method by which relatively small quantities of concentrates are treated; however, the design of a mini-smelter must incorporate innovative equipment not common to industry in order to maximize vessel availability and minimize cost.

Work area emissions are a very important item. These are controlled by a hood which covers the vessel mouth with a flexible seal. Inleakage into the process off-gas system is controlled through pressure measuring devices on the hood. This plays a key part in the off-gas cleaning system. Off-gases, and cost of off-gas cleaning, is a function of the volume; by minimizing leakage, the volume is kept to a minimum. The more concentrated SO_2 is in the off-gas, the easier it is to fix. This type of hood design permits this capability. The entire TBRC^{TM} area is enclosed in a hood which is ventilated. The TBRC^{TM}, when the process hood is open, produces a small initial puff of emission as the vessel comes to equilibrium with the ambient pressure. Beyond that, there is very little emission. This is an important environmental consideration when using the TBRC^{TM}.

Normally not included in smelters are large slag recovery hoods, which collect the fugitive emissions. At Afton, the fugitive emissions are not scrubbed as the SO_2 concentrations are relatively low.

Another item is the use of a concrete converter aisle. Old time copper operators will shudder at the thought, but after 2 1/2 years of operation, although it doesn't look brand new, there are very few spill marks, and a power sweeper is able to keep the convertor aisle clean. This keeps down in-plant particulate emission and it also encourages operators to be far more careful than they might normally be with the typical dirt floor in a copper converter aisle.

In a modern smelter, material balance control is a very important item. Most plants weigh the charge to the smelter and weigh the copper out the door, but few weigh the intermediate products. At Afton the ladles are weighed on a platform scale and these weights are used to keep both copper and slag material balances. Copper weighed out the door is the final product, but there are always recycle products; weighing permits the operators to monitor the difference between copper out of the TBRC and copper which goes to market.

Another key item in the mini-smelter is the type of crane which is chosen. The typical smelter crane is a two-girder crane. It consists of the main hoist and two auxiliary hoists. The weakness of this type of system is that the access of the main hoist to the edge of the converter aisle is limited by the width of the auxiliary hoists. In order to minimize this, two main hoists have been used. This in fact does cut down the distance or improve the approach. However, main hoist speeds are 30-70 feet per minute about 1/3 or 1/4 of auxiliary hoists. This slows down the operation. The crane at Afton which is the first application in North America, is a four-girder crane. A four-girder crane offers both operating and economic advantages. Its difference lies in that the auxiliary hoist travels on its own pair of girders; the main hoist on a separate pair, hence four-girder crane. The auxiliary hoist travelling on separate girders; has the ability to make very close approaches to either side of the main hook, no matter how close the main hook comes to the converter aisle edge. The economic advantages lie in that, because closer approaches are made, capital costs of cranes are less, because cranes are priced more on span than on capacity. The other item affecting operating costs is that four-girder cranes, historically a normal item in a steel mill, have much higher reliability than typical copper smelter cranes.

Another innovation which also affects capital costs is the use of the slide gate valve. The valve is hydraulically operated and can be opened at any degree from full closed to full open. The ladles used at Afton Mines are heated refractory lined ladles. In order to minimize plugging of the orifice for the slide gate valve, silica sand is poured down into the valve nozzle area; when the valve is opened there is a flow of the silica sand followed by the molten copper.

The ladle is used instead of a holding furnace after being heated and supported on a casting stand with a cover. In-plant fugitive emissions are cut in half, because molten copper transfers are now reduced to one, rather than two.

In a TBRC™ copper smelter, smelting, converting, and refining, are performed in a single vessel. To man the TBRC requires one operator and one helper.

Process gases go from the cooler to an electrostatic precipitator and then to the dual alkali scrubbing system, which Afton Mines utilizes to control all SO_2 emission. The cleaned process gas stream passes into a venturi, a tray tower SO_2 and then off to reheating and to a stack. The scrubbing is accomplished through a dual alkali system. Regenerated solution, a mixture of sodium sulfite and caustic, is pumped to the venturi and down the tray tower, the solution after stripping the SO_2 from the process off-gas stream, is collected in a spent solution tank. The spent liquor is regenerated by reacting with mild of lime solution, removing the sulfur dioxide as calcium sulfide crystals, which are settled in a thickener. The pulp is filtered for additional water removal and recovery of the scrubbing agent. The solution is then returned to the regenerated liquor storage for use on the next cycle. The sludges are then sent to the tailings pond. At Afton less than 100 TPD of sludge, is mixed with 7,000 TPD of tailings. The use of a fiberglass stack reduced the cost to one quarter the cost of a concrete stack.

In large tonnage facilities, instead of using one vessel for smelting, converting and refining, it is preferable to have a primary smelting furnace and then use the TBRC as a converter.

In light of the development of small high grade or remote mineral deposits, local highly efficient mini-smelters become increasingly attractive. Local populations may be more willing to accept a small integrated smelter rather than a large central smelter. The versatility of TBRC for both a process and an environmental control offers a viable alternative to the Pierce-Smith converter and anode furnace when used in conjunction with large primary smelting furnaces.

The QS oxygen process is an attempt to reach that 'will of the wisp' panacea of a smelting process, with the entire process being conducted in a single vessel. The process advantages of the QS as proposed, offer single vessel steady-state conditions, countercurrent flow of matte and slag, staging of oxygen activities, temperature control, and shielded submerged tuyeres. The accomplishment of this objective is done in the QS oxygen reactor, which is approximately 20' dia. x 150' long, at a 1° slope. Concentrate is charged to the QS reactor via vortex burners which smelt concentrate with oxygen, and produces a matte. Subsequently, air is blown into the matte through shielded tuyeres. The matte flows counter-current to the slag and, prior to discharge, coal, SO_2, nitrogen and/or oxygen are injected through a shielded tuyere to lower the oxygen activity of the QS process.

One of the problems with large innovative processes is the large sum of money they take to develop. In order to minimize these expenditures, an innovation which is directly applicable today to existing reverberatory furnaces is the Oxygen Sprinkle Smelting furnace. The Oxygen Sprinkle Smelting furnace is an outgrowth of a need to develop the vortex burner for QS and what has transpired is the development of an oxygen sprinkler burner which is designed to mix oxygen and concentrate, and ignite them inside the reverberatory furnace. The key to this is the ability of the sprinkler burner to provide a very broad horizontal distribution within a very short vertical height. This transforms the reverberatory furnace into an oxygen flash smelting reactor. In addition to making it an oxygen smelting reactor, the addition of lower grade concentrate coal and oxygen at the fourth sprinkler burner towards the slag skimming end, reduces the oxygen activity of the slag in that area by sprinkling low grade matte on the surface of the slag, washing the slag and producing potentially much lower copper in slag. The advantages of this process are that it can salvage the reverberatory furnace. To replace an existing reverberatory operation with one of the new efficient flash smelting type furnaces is a pretty costly item, whereas the conversion of an existing reverberatory to an Oxygen Sprinkle Smelting furnace costs 5-7 million dollars. The largest portion is for the fluid bed dryer, because as in all flash smelting processes, the material must be dried to less than 1%.

A second advantage is that it increases the SO_2 concentration. The reverberatory furnace, by use of oxygen has reduced gas volume and increased SO_2 concentration which permits the gases to go te an acid plant.

The third advantage is the greater throughput; the sprinkler burner can smelt at rates up to 450 TPD which, using four burners, amounts to 1350 STPD.

Fourth, by adjusting the oxygen concentrate ratio to the various burners, the matte grade can be controlled. The matte grade at the first burner could be controlled at 65% copper, the second burner 55% copper; the third burner would then produce a 45% matte, and the fourth burner would be a 30% matte. The net effect would be a 50-55% copper matte as a product of the furnace.

The fifth advantage is a potential for less copper in the slag due to the cleaning of the slag by the low-grade goal concentrate smelting burner, and reduction in the oxygen activity of the slag.

A list of papers referenced follows.

REFERENCE PAPERS

(1) FULL SCALE TBRC SMELTING TESTS ON COPPER CONCENTRATE by R.A. Daniele and L.H. Jaquay, Dravo Corporation, Pittsburgh, Pa. Presented at 1974 AIME Annual Meeting.

(2) TBRC - A NEW SMELTING TECHNIQUE by R.A. Daniele and L.H. Jaquay, Dravo Corporation, Pittsburgh, Pa.
Presented at 1972 AIME Annual Meeting.

(3) THE Q-S OXYGEN PROCESS by P.E. Queneau, Thayer School of Engineering, Dartmouth College, Hanover, New Hampshire and R. Schumann, Jr., School of Materials Engineering, Purdue University, West Lafayette, Indiana Reprinted from the Journal of Metals, August 1974.

(4) METAMORPHOSIS OF THE COPPER REVERBERATORY FURNACE: OXYGEN SPRINKLE SMELTING by Paul Etienne Queneau, Thayer School of Engineering, Dartmouth College, Hanover, New Hampshire and R. Schuhmann, Jr., School of Materials Engineering, Purdue University, West Lafayette, Indiana Reprinted from Journal of Metals, Vol. 31, No.12, December 1979, pp. 12-15.

TABLE 1

ELIMINATION OF IMPURITIES

	ARSENIC		LEAD		BISMUTH		SELENIUM	
	Assay %	% of Total	Assay %	% of Total	Assay %	% of Total*	Assay %	% of Total
Concentrate	.0057	-	.0027	-	.0005	-	.013	-
Slag	.0013	14.8	.0015	35.8	.0001	13.2	.001	5
Blister Copper	.0020	8.7	.0010	9.0	.0002	10.3	.006	11.3
Dust	.0300	65.9	.0110	50.3	.0016	39.7	ND	-

* Due to the difficulty of measuring these low concentrations, it was not possible to account for all the bismuth.

FIGURE I

TBRC ARRANGEMENT

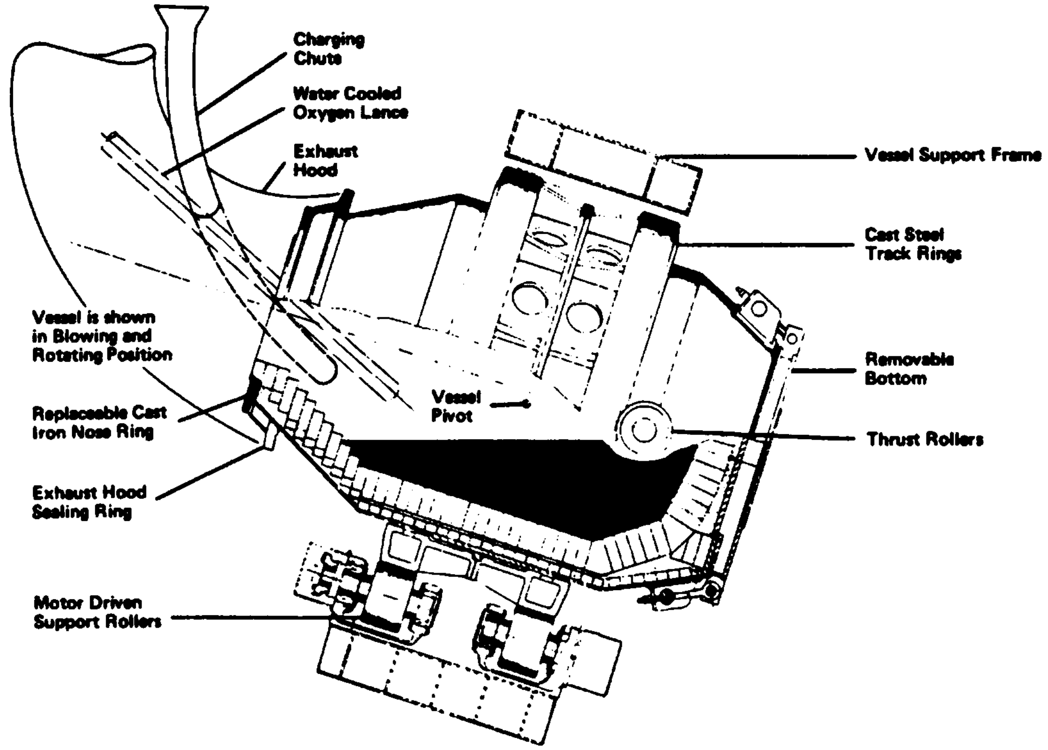
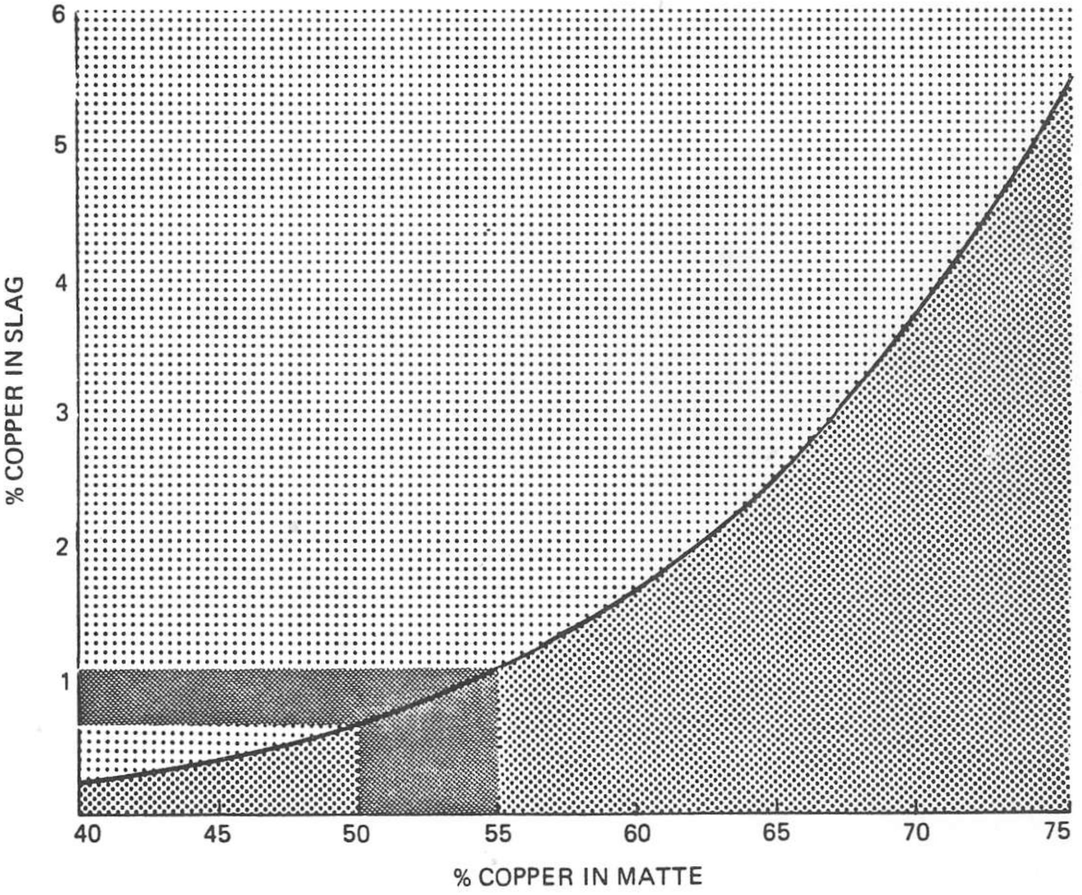


FIGURE 2

COPPER IN SLAG AS A FUNCTION OF COPPER IN MATTE

A curve (Figure 2) was prepared from analyses of various slag and matte samples taken during these tests. Based on operating at an average matte concentration of 50-55% during the period of concentrate addition, the recirculating load of copper through the slag cleaning equipment would be less than 3% of the total copper in the concentrate.



Analyses of the matte and slag for arsenic, lead, bismuth and selenium is presented in Table II. No specific steps were taken during these tests to drive off these impurities. Elimination of these elements was excellent, even at the low levels present in the concentrate: 93% of the arsenic, 91.0% of the lead, 89.7% of the bismuth and 88.7% of the selenium.

SECTION X

OVERVIEW ON COPPER EXTRACTION TECHNOLOGY FOR BRITISH COLUMBIA

BY W.J.S. CRAIGEN

OVERVIEW ON COPPER EXTRACTION TECHNOLOGY FOR BRITISH COLUMBIA

by
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CANMET
Dept. of Energy, Mines and Resources Canada

For Presentation at
British Columbia Copper Smelting and Refining
Technologies Seminar

Organized by the
B.C. Ministry of Energy, Mines and Petroleum Resources

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Vancouver, B.C.

SUMMARY

This overview is an attempt to document on a comparable basis the relative merits of various copper smelting/refining processes, with special emphasis on factors relevant to a British Columbia location.

From an industry point of view, return on investment, which usually can be related directly to capital and operating costs, is nearly always the most important criterion for evaluating processes and making investment decisions, with higher risk technology usually requiring a higher return on investment. However, from a government viewpoint, be it federal or provincial, factors related to environmental impact, efficiency of resource recovery, energy type and efficiency, plant location, employment and potential long term benefits are often of equal or greater concern than those related to the economics of the process. These factors are given due consideration in this assessment of the alternatives for the further processing of copper in B.C., so that hopefully conclusions and recommendations can be made that will facilitate the efficient recovery and marketing of a valuable resource, while reflecting the needs and desires of both government and industry.

INTRODUCTION

This overview was prepared at the request of the B.C. Ministry of Energy, Mines and Petroleum Resources. It is intended primarily to document the major factors which affect the selection of appropriate copper extraction technology for B.C., and to offer some suggestions on how government and industry can inter-act to promote the development of the most desirable technology.

First, I would like to congratulate Frank Basham and his associates in the Ministry for their excellent efforts in organizing this very useful and well attended Seminar. I would also like to thank Frank for this opportunity to say a few words here today. Ottawa and Vancouver, as you know, are far apart geographically and sometimes even further apart politically, so I'm sure you will all be gratified to know that cooperation at the working level between our respective departments, at least with respect to minerals has never been better, and I certainly intend to do everything I can to maintain this close co-operation, which I believe is essential to facilitate the expansion of the mineral industry in B.C.

We have heard some excellent presentations over the past two days on copper extraction processes which could have special application for a B.C. location. We are are perhaps fortunate, in this day and age, in having so many options to choose from. However, having so many choices obviously makes the evaluation and selection task more difficult and time-consuming. This overview therefore is certainly not intended to be an in-depth evaluation of the various process options, but rather an attempt to document on a comparable basis the relative merits of copper extraction processes, with special emphasis on factors relevant to a B.C. location.

As a basis for comparison of the various process alternatives, five criteria have been chosen which it was felt would reflect the major concerns of both government and industry in the selection of appropriate copper extraction technology. These criteria are listed, not in order of priority, but rather in order of increasing complexity. The criteria are:

- Efficiency of Resource Recovery
- Plant Size Considerations
- Environmental Considerations
- Energy Type and Efficiency
- Capital and Operating Costs

In this overview I will attempt to deal individually with each criterion, although obviously there will be some overlap and the importance of each criterion to the overall process evaluation will be very dependent upon other factors such as the characteristics of ore and concentrate, the location of the processing plant, by-product marketing problems, etc.

EFFICIENCY OF RESOURCE RECOVERY

B.C.'s present production of copper concentrates amounts to about 340 000 tons per year of contained copper. Most of this is recovered from porphyry type ores containing less than 0.5% copper. The copper concentrates produced average about 29.5% copper, with recoveries from the ore averaging about 86%. For most producing mines the recovery of copper from these low-grade ores is reasonably good, and it is unlikely that significant improvements in copper recovery can be realized economically. However, there are significant quantities of gold, silver and often molybdenum associated with nearly all B.C. deposits, and in most cases the recoveries of these valuable by-products leave something to be desired. Table 1 shows the range and weighted average analyses of presently known B.C. ores and concentrates and existing recoveries of the metal values. The range of values shown for ore and concentrate analyses include Afton and Western Mines. However, because the grade and mineralogy of these two deposits are certainly not typical, they were omitted from the average analyses calculations.

Table 2 shows the range and average value of the precious metals in the concentrates relative to the copper value at today's prices. The precious metals already amount to about 20% of the total concentrate value, even though recoveries are only about 50%. What does all this have to do with the evaluation of copper extraction processes? Well, for one thing, it shows that the extraction process chosen will have to be capable of achieving high recoveries of precious metals, as well as copper, from most B.C. concentrates. This is going to become increasingly important in the future if gold and silver values keep increasing at rates well above the inflation rate, which is very probable. For the pyrometallurgical processes described at this seminar, precious metal recovery should be no great problem. Pyrometallurgical copper processes in general have always been able to achieve very high (97%) recovery of precious metals in the blister copper produced. These valuable by-products can then be readily recovered in the conventional electrorefining process. For hydrometallurgical processes, however, although copper recoveries are comparable to pyro-processes, precious metal recovery can be much more difficult. Although only one hydrometallurgical process (S-C process) (1) was described at this seminar there are at least two others that I think are worth mentioning, namely the Duval "Clear" (2) and Cyprus Mines "Cymet" (3) processes. There is also a new hydro-process (4) presently in the development stage right here in B.C. However, the efficiency of recovery of precious metals from all of these processes, and the cost of so doing, has not as yet been adequately demonstrated.

You might say therefore that from a resource recovery point of view pyrometallurgical processes have the edge over hydrometallurgical processes for applieation to B.C. copper concentrates. I would temper that a bit, however, by stating that the major loss of precious metals, as indicated from the recovery figures in Table 1, is in the mill tailings. Reducing the losses here could nearly double the value of precious metals recovered. It would seem therefore that it is even more important, from a resource recovery view-point, to improve the recovery of precious metals in the concentration process, as it is to ensure high recoveries in the extraction process. To achieve this additional recovery it may be necessary to reduce the present grades of concentrates. Since costs and recoveries in hydrometallurgical processes are less dependent on grade than in pyro-processes, long-term application of the hydro approach to B.C. copper concentrates containing precious metals could still prove to be a desirable option. The recovery of molybdenum could also conceivably be improved using a hydrometallurgical process to treat a bulk concentrate. However, the problem here is much more complex and will require studies that are certainly beyond the scope of this overview.

PLANT SIZE CONSIDERATIONS

From a strictly governmental viewpoint it would be desirable to process all of B.C.'s copper concentrates to metal, and to take full advantage of economies of scale by doing so in one extraction plant. Studies carried out by EMR's Economic and Policy Analysis Sector on zinc smelters (5) indicate that internal rate of return can be increased by about 4.5% in going from 100 000 to 300 000 tonnes per year plant capacity. Technically, plants in this size range for copper are feasible, using either pyrometallurgical or hydrometallurgical technology. To be realistic, however, production throughputs have to be tailored to market requirements, and it is unlikely that copper markets will be adequate, at least initially, to justify plant capacities in excess of 100 000 tonnes per year. The question then arises: "Can we be competitive at throughputs considerably less than 100 000 tonnes per year, and if so, would it perhaps be beneficial, particularly for regional employment, to have a number of smaller processing plants, rather than one large complex?". In the past it has been generally conceded that for throughputs below about 50 000 tonnes per year copper, hydrometallurgical processes would be preferable to pyrometallurgical processes, particularly with respect to capital costs and associated environmental costs. However, the development of the TBRC process (6,7) for copper, and the installation of a commercial smelter at Afton Mines, (28 000 tonnes per year) has more or less shot down the theory that pyro-processes cannot compete at low throughputs. Admittedly the Afton concentrate is unique and very high-grade, and the low sulfur and high precious metals content make it an ideal concentrate for the TBRC. The KHD Cyclone Smelting process (8,9) can be efficiently applied to conventional copper concentrates at throughputs in the range of 15 000 to 50 000 tonnes per year copper. This will allow relatively small producing mines a choice of either hydrometallurgical or pyrometallurgical process options for the production of a metallic copper product at or near the mine site. The technology ultimately selected will be very dependent on the environmental costs associated with plant location. At these low throughputs the economics of an acid plant become questionable even if a market for acid exists, particularly for inland locations where transportation costs will severely limit acid marketability. It is likely therefore that the costs of SO_2 capture-fixation will have to be considered very carefully in the evaluation of the pyrometallurgical options. The KHD process is continuous, and in that respect it has a potential advantage over the TBRC by producing a reasonably constant gas flow and uniform SO_2 concentration which should facilitate SO_2 fixation. The hydro-processes of course produce elemental sulfur, which can be recovered and sold if markets exist, or stored indefinitely with no detrimental environmental effects. Hydro-processes therefore have no production constraints related to environmental considerations. The environmental advantages of the hydro-process will, however, have to be weighed very carefully against their higher energy requirements, which I will discuss later, and the potential precious metal recovery problems mentioned previously.

As a matter of interest CANMET recently initiated a project to evaluate SO_2 capture-fixation options for smelters, which do not entail the production of sulfuric acid. The results of this study should be available by late spring of 1981.

Technically, all of the pyrometallurgical and hydrometallurgical processes available for treating copper concentrates could be utilized at throughputs greater than 50 000 tonnes per year copper.

However, at this point in time it would appear that the TBRC and KHD process economics are most favourable at throughputs below about 800 tonnes per day of concentrate (80 000 tonnes per year Cu), while the other pyrometallurgical processes look more economically attractive at throughputs above this figure. I would think however, that the KHD being a continuous process with a very high smelting rate, could eventually compete with the other processes at higher throughputs.

The hydrometallurgical processes should be capable of competing economically at any throughput. However, economies of scale at very high throughputs are unlikely to be as significant for hydro-processes as those projected for the pyrometallurigcal alternative.

ENVIRONMENTAL CONDITIONS

There is little doubt that environmental problems related to SO_2 will be the largest single constraint imposed on the pyrometallurgical process options for copper, particularly at throughputs greater than 50 000 tonnes per year copper. A recent study by CIL (10) indicated that for a B.C. coastal location only acid or elemental sulfur would be marketable at production rates of about 100 000 tonnes per year contained copper or over. Sulfur dioxide, ammonium sulfate and gypsum were not considered marketable at these throughputs. Unfortunately a phosphate fertilizer product was not included in the study. The CIL figures indicated a positive netback to the smelter was possible for both sulfur and acid, based on off-shore markets, but was unlikely for domestic or U.S. markets. Under no circumstances however could the netbacks come close to paying for the production costs of either the acid or the sulfur. Based on projected future prices this scenario is unlikely to change over the next ten years, although elemental sulfur for off-shore markets offers the best potential for increasing netbacks in real terms and minimizing sulfur fixation costs. Table 3 shows the actual costs to the smelter for sulfur fixation based on capital and operating costs estimated by Reimers and Associates (11). The average sulfur to copper ratio in most B.C. concentrates is about 1.1 to 1, so the costs shown, if based on $/ton of copper produced, will only be about 10% higher. The Reimers' costs for acid production are based on a continuous gas flow containing 12% SO_2, and a single contact acid plant. Sulfur plant costs are based on a continuous flow of 12% SO_2 gas with very low oxygen. Capital charges for both plants include amortization at 10% interest over 15 years. The two plant sizes are comparable at 120 000 tonnes per year S (390 000 tonnes per year H_2SO_4) for the acid plant, and 100 000 tonnes per year S for the sulfur plant.

The Reimers' costs for sulfur are based on the ORKLA process used by Outokumpu. Also shown in Table 3 for comparison is a cost estimate for the Allied Chemical Sulfur Process (12) updated to mid 1980 dollars. These costs are based on an 80 000 tonnes per year sulfur plant, with electricity costs at 2c/kWh and natural gas costs of $2.00/1000 ft^3, the same as that used in the Reimers' study. The interesting feature here is that using the Allied Chemical process costs, it would appear to be possible to break even at the netbacks estimated for off-shore sulfur sales. In fact, a profit of about $17.00/ton of sulfur could actually be realized at costs based on inlet gas concentrations of 85% SO_2.

How does all of this impact on our relative evaluation of copper processes for B.C.? If the CIL and Reimers' figures (10,11) are correct then the production of elemental sulfur is the most cost effective option for a B.C. copper smelter to control SO_2 and emissions. If the cost figures on the Allied Chemical sulfur process are correct the pyrometallurigical process capable of generating the highest concentrations of SO_2 and the lowest concentrations of oxygen in a constant flow gas stream is the most coet effective. There are of course other considerations, such as the large quantities of natural gas required to produce elemental sulfur, which according to the figures documented (12) remains constant at the equivalent of about 17.4 million Btu/ton sulfur (about 55% above theoretical minimum), at SO_2 inlet gas concentrations of 25% or over. The effect that this has on the total energy consumption of the process is of course very significant, and I will touch on this again a little later.

In the Allied Chemical process the reduced costs of making sulfur from high SO_2 concentration are due entirely to lower capital costs, which can be attributed to lower gas volumes. However, the process still requires a constant gas flow. Table 4 lists the pyrometallurgical process options for copper which have been discussed at this seminar, and shows the percent SO_2 and O_2 in the off-gas streams and the percentage of total sulfur in the concentrate that can be recovered at a relatively constant flow of high strength SO_2. Obviously if the ultimate answer to the SO_2 problem at copper throughputs of 100 000 tonnes per year or over is to convert all of it to elemental sulfur, the pyro-process best capable of doing so will be one that utilizes oxygen most efficiently for converting all of the sulphides in the concentrate to a constant flow of very high strength SO_2 gas. None of the processes developed to date have achieved this capability. However, as can be seen from Table 4, at least two or three are coming close, and I'm sure that if the incentives are there, further improvements can and will be incorporated to minimize the costs related to sulfur fixation.

The hydrometallurgical pnocess options for copper have of coarse virtually no environmental constraints related to SO_2. The Duval (2) and Cymet (3) processes produce elemental sulfur directly in the extraction step, while in the S-C process (1) half of the sulfur is produced directly in the process and half is produced in a Claus plant by converting SO_2 and H_2S off-gas streams to elemental sulfur. In the economic evaluation of the hydrometallurgical options no credit is ever given for elemental sulfur. There is little doubt therefore that the hydro-processes will give pyro-processes very sarious competition at either low or high throughputs, but particularly in locations where sulfuric acid cannot be sold at a positive netback to the smelter, and where the most cost-effective alternative requires the production and sale of sulfur.

There are of course other environmental concerns that both pyro and hydro-processes have to contend with, such as the levels of arsenic, antimony, mercury, heavy metals, etc. in gaseous and aqueous effluents. However, with one or two exceptions the levels of contaminants in B.C. copper concentrates are relatively low, and the costs involved in controlling them should be minor compared to that required for SO_2 control. For all processes, however, the stability of arsenic, mercury and heavy metal-bearing residues will have to be examined carefully to ensure that the process can meet federal and provincial water quality standards.

ENERGY TYPE AND EFFICIENCY

To the best of my knowledge there has only been one serious effort made to document the relative energy efficiencies of pyrometallurgical copper processes on a comparable basis. This is the study by Kellogg and Henderson (20) in which the process thermal and electrical energy requirements are broken down into smelting, converting, anode production and acid manufacture, for a concentrate feed analyzing 29.5% Cu, 26% Fe 31% S and 8% SiO_2. The electrical energy requirements documented in this report are converted to process fuel equivalents (PFE) using a factor of 10,500 Btu/kWh which is the average efficiency for U.S. thermal plants. Although this 33% efficiency factor is justifiable for thermal plants, it can hardly be justified for a B.C. location where more than 95% of the power is hydro generated. Table 5, therefore, is an attempt to document the comparable thermal and electrical energy requirements for pyrometallurgical processes, using a PFE factor of 3410 Btu/kWh. Most of the figures are based on the Kellogg data with ranges shown where comparable figures were available for different concentrate feeds. The conclusions drawn are essentially the same as those reached by Kellogg. For pyrometallurgical processes, the maximum use of oxygen enrichment with very high oxygen utilization efficiency is desirable to minimize total energy requirements. This tends to maximize the electrical energy input to the process, while also maximizing the utilization of the sulfur fuel in the concentrate.

With B.C.'s low electrical power costs (16 mils) this approach should be even more desirable to minimize total energy costs. In fact, because of the low power costs the economics of waste heat boilers for power generation may be somewhat questionable, in which case those processes which have the lowest energy requirements without credits should have an advantage.

These same low power costs can of course make the hydrometallurgical process options which utilize electrowinning for copper recovery more competitive. Table 6 shows a comparison of the energy requirements of the pyro and hydro-systems. It should be noted that although the energy requirements of the hydro-processes are higher, in most cases the additional requirements are electrical, which may not pose much of a burden on the process economics because of B.C.'s low power rates.

It is also evident from Table 6 that the difference in energy consumption between hydro and pyro-processes will completely disappear if acid cannot be marketed and SO_2 has to be converted to elemental sulfur, which as discussed earlier, may be the most cost effective option for copper pyrometallurgical processes in B.C. However, justification of this approach, from an energy point of view, could be very difficult, since nearly all of the additional requirement is thermal, and since most of the sulfur would be exported, the energy content of the sulfur (11.5 x 10^6 Btu/tonne) would also be lost. It is evident, therefore, that we have a dilemma between energy and environmental considerations for pyrometallurgical copper processes, which, if energy is given priority, could limit copper production to the equivalent marketable acid, or could impose much higher costs to the smelter for alternate methods of controlling SO_2 emissions.

CAPITAL AND OPERATING COSTS

Because no process evaluation would be complete without some mention of costs, I have attempted to add up together in Table 7 an estimate of capital and operating costs of the various copper extraction processes in mid 1980 dollars. These estimates are for the most part based on published or unpublished figures obtained from various sources over the past eight years. The ranges shown indicate differences in the source data due to plant size or location and/or probable inaccuracies in extrapolating to 1980 dollars. The figures are not definitive with respect to a specific concentrate feed, and are meant only to reflect the probable range of costs likely to be encountered, with the lowest costs being applicable only to very high throughput plants located in urban industrialized areas. The pyro-process costs include oxygen and acid plants, but not electrorefining or any costs attributable to negative netbacks from acid sales. Operating costs do not include depreciation or amortization. The pyro-process costs are based on throughputs of 100 000 tonnes per year copper or over, while the hydro-processes are based on throughputs in the 40 000 to 80 000 tonnes per year range. Partly for this reason, the pyro-process operating costs are considerably lower than the hydro-process costs. Also, to legitimately compare with the S-C and Cymet processes the pyro-process costs would have to include electrorefining to produce wire bar copper. The figures, therefore, are not directly comparable, nor are they definitive as the range implies. It has been shown numerous times in the past, by the proponents of hydrometallurgical process (1, 3), that pyro and hydro-process operating costs for the production of wire bar copper are about equal at throughputs of 75 000 tonnes per year copper or less. The hydro-process capital costs, however, as indicated from Table 7, are very competitive with the pyroprocess costs, even at high throughputs, and will be much more competitive at low throughputs where the pyro-process capital costs per tonne of product will increase substantially. The KHD and TBRC processes, however, appear to be exceptions to this rule, with the capital costs shewn being valid at 15 000 to 20 000 tonnes per year copper. Coupled with the low operating cost estimates, these two pyro-processes should be able to compete exceptionally well with hydro-metallurgical alternatives at low throughputs.

In the long term, because inflation factors, plant size and location, and by-product marketing have such a great effect, it will become more and more difficult to select an extraction process solely on the basis of comparable capital and operating costs. Obviously costs have to be considered, and investigated very thoroughly, but only in conjunction with other relevant criteria, some of which I have attempted to outline in this overview.

CONCLUDING REMARKS

Now that we have documented what I believe are the major criteria for evaluating copper extraction processes, what are the next steps to make forward progress?

According to a recent EMR report (22), Central B.C. and the Yukon offer the greatest potential in all of Canada for major mineral development, with copper being the primary mineral component in Central B.C. and lead and zinc in the Yukon. However, the report goes on to say that since constraints to development are numerous, it is a region where government policy initiatives could have the greatest impact on future mineral development.

This is clearly the case when we consider the opportunities and constraints related to the further processing of B.C.'s considerable copper resources. Some of the aspects that government initiatives may assist in resolving are, for example:

- The most cost-effective alternative for SO_2 control (i.e., acid or elemental sulfur) for a large B.C. smelter. The production of phosphate fertilizer for instance, has not as yet been evaluated, and long range forecasts for sulfur and H_2SO_4 markets require further verification.
- The evaluation of the alternate technologies, by helping to finance the in-depth studies required to recommend a specific technology for a specific application.
- The promotion and coordination of industry participation in the commercialization of relatively new technology, if the in-depth studies indicate that this is a superior option.

Governments can also ensure that up-to-date information is available on concentrate supply, analyses, long-term market commitments, and future copper supply/demand relationships, so that the copper extraction process can take full advantage of whatever economies of scale exist.

Broader policy decisions are equally important, and can have a very significant effect on the selection of process technology. For example:

- Should natural gas be used to produce sulfur from SO_2, and, if so, what is the relationship required between long term sulfur prices and natural gas costs?

- How important are energy considerations? Should processes which use a high percentage of electrical energy be given priority or preference over processes which may use less total energy, but whose thermal energy requirements are higher?

- How important is resource recovery? Should minimum recovery requirements be set for copper and associated by-products, to force improvements in extraction technology?

- Will low throughput plants be encouraged to promote regional employment, and if so will infrastructure costs be subsidized to improve the economic viability of these smaller plants?

- How important is plant location? A large smelter complex would have to be located on or near tidewater to minimize transportation costs for sulfur or sulfuric acid marketing. Hydrometallurgical plants do not have this constraint, since sulfur marketing is not essential to the process economics.

These are all important considerations requiring government action and input. However, industry too should be doing something to improve the weak points in the copper extraction processes. For hydro-processes much more work is required to document their effectiveness for precious metal and molybdenum recovery. For pyro-processes, although the SO_2 constraint can never be eliminated, the costs associated with it can be minimized by developing processes that convert all of the sulphides into a continuous flow of high concentration SO_2 gas.

Recently Anaconda announced the permanent closing of its west coast smelter refinery totalling 200 000 tonnes per year Cu, primarily because of the estimated costs required to meet environmental regulations. There has been some talk about a "super-smelter" to replace the lost production. It would appear, therefore, that this is an ideal time to promote B.C., with its low-cost power, for the establishment of a "world scale" copper extraction plant. But we must do everything possible to ensure that the technology chosen is superior, environmentally, socially, and economically, and can compete favourably on world markets for a long time to come.

REFERENCES

(1) The Sherritt-Cominco Copper Process, CIM Bulletin Feb. 1978, p. 105.

(2) Atwood, G.E. and Livingston, R.W., "The Clear Process - A Duval Corp. Development"; Annual Conference of Gesellschaft Deutscher Metallhutten und Bergleute; Berlin; Sept. 26-29, 1979.

(3) McNamara, J.H., et al., "Hydrometallurgical Process for the Extraction of Copper"; CIM Bulletin; p. 201; March 1980.

(4) Craigen, W.J.S., 'Great Central Mines Hydrometallurgical Copper Process - Evaluation and Recommendations for Commercial Application"; CANMET Minerals Research Program Report No. MRP 80-2(CF); June 1980.

(5) Unpublished Report EMR, Economic and Policy Analysis Sector, Ottawa, Ont.

(6) Daniele, R.A. and Jaquay, L.H. "TBRC - A New Smelting Technique"; AIME Annual Meeting, San Francisco, Calif.; Feb. 1972.

(7) Daniele, R.A. and Jaquay, L.H. "Full Scale TBRC Smelting Tests on Copper Concentrate"; AIME Annual Meeting, Dallas, Texas; Feb. 1974.

(8) Sauert, F. et al. "High Efficiency System for Pyrometallurgical Recovery of High-Grade Copper Matte from Sulphide Concentrate"; Erzmetall 33:7/8: 371; 1980.

(9) Chaudhurz, K.B. "Top-Blowing Technology in Copper Pyrometallurgy"; CIM Conference of Metallurgists, Sudbury, Ont.; Aug. 1979.

(10) "Sulfur Values from a British Columbia Zinc Smelter"; Prepared for B.C. Lead/Zinc Smelter Steering Committee by CIL, Montreal, Que.; Sept. 1980.

(11) "A Study of Sulfur Containment Technology in the Non-Ferrous Metallurgical Industry"; Environment Canada Report EPS 3-AP-79-8; Prepared by J.H. Reimers & Assoc.; April 1980.

(12) Hissong, D.W. et al. Battelle Columbus Laboratories, "Reduction Systems for Flue Gas Desulfurization"; Chemical Engineering Progress; p. 73; June 1977.

(13) Antonioni, T.N. et al. "Operation of the INCO Flash Smelting Furnace with Recycle of Converter Slag"; CIM Conference of Metallurgists, Sudbury, Ont.; Aug. 1979.

(14) Diaz, C.M. et al. "INCO's Oxygen Flash Smelting Process for Copper and Nickel Concentrates"; INCO Metals Co. Process Research; Nov. 1978.

(15) Bell, M.C. et al. "Oxygen Flash Smelting in a Converter"; Journal of Metals; p. 9; Oct. 1978.

(16) Harkki, S. et al. "High-Grade Matte Production with Oxygen Enrichment by the Outokumpu Flash Smelting Method"; Extractive Metallurgy of Copper; Vol I, p. 488.

(17) Nagano, T. and Suzuki, T. "Commercial Operation of Mitsubishi Continuous Copper Smelting and Converting Process", Extractive Metallurgy of Copper; Vol I, p. 439.

(18) Mills, L.A. et al. "Design and Operation of the Noranda Process Continuous Smelter"; Extractive Metallurgy of Copper; Vol I, p. 458.

(19) Bailey, J.B.W. and Storey, A.G. "The Noranda Process After Six Years' Operation"; CIM Conference of Metallurgists; Sudbury, Ont.; Aug. 1979.

(20) Kellogg, H.H. and Henderson, J.M. "Energy Use in Sulphide Smelting of Copper"; Extractive Metallurgy of Copper; Vol I, p. 373.

(21) Mangeng, C. and Mead, R. "Sulfur Dioxide Emissions from Primary Non-Ferrous Smelters in the Western United States"; Los Alamos Scientific Laboratories LA-8268-MS; Aug. 1980.

(22) "Some Common Regional Constraints to Mine Development"; Mineral Policy Sector, Internal Report MRI 80/81; Energy, Mines and Resources Canada.

TABLE 1

BRITISH COLUMBIA COPPER CONCENTRATES

	ANALYSIS				RECOVERY (%)	
	ORE		CONCENTRATE			
	AVE.*	RANGE	AVE.*	RANGE	AVE.*	RANGE
Cu (%)	0.43	0.14 - 1.32	29.4	23 - 60	86	79 - 91
Au (oz/t)	-	-	0.15	0.01 - 1.2	-50	<50 - 79
Ag (oz/t)	-	-	2.1	1.0 - 50	-50	<50 - 67
Mo (%)	0.015	0.006-0.036	-	-	66	45 - 77

* Averages are weighted, based on concentrate production.

TABLE 2

VALUE OF METALS IN B.C. COPPER CONCENTRATES

	$/TON OF CONCENTRATE		
	AVE.	RANGE	
COPPER	588	460 - 1200	($1.00/lb)
GOLD	105	7 - 840	($700/oz)
SILVER	42	20 - 1000	($20/oz)
TOTAL	735		
Au + Ag % of total	20		

TABLE 3

SMELTER COSTS FOR SO_2 CAPTURE-FIXATION
\$/TON SULFUR FIXED (1980)

SMELTER PRODUCT	OPERATING COST	CAPITAL CHARGES	TOTAL SULFUR FIXATION COST	MAXIMUM NETBACK(2)	MINIMUM ACTUAL COST TO SMELTER
Sulfur (ORKLA Process - 13% SO_2)	67(1)	51(1)	118	91	27
Sulfuric Acid	19(1)	30(1)	49	14	35
Gypsum(3)	84(1)	45(1)	129	0	129
Sulfur (Allied Process -25% SO_2)	39(4)	52(4)	91	91	0
Sulfur (Allied Process -85% SO_2)	39(4)	35(4)	74	91	-17

(1) Based on mid 1979 estimates of costs for 390 000 tpy H_2SO_4 & 100 000 tpy S.
(2) Based on Aug. 1980 off-shore prices F.O.B. Vancouver.
(3) Based on limestone neutralization of sulfuric acid.
(4) Based on 1976 costs up-dated to mid 1980.

TABLE 4

SO_2 CONCENTRATION AND DISTRIBUTION IN PYROMETALLURGICAL COPPER PROCESSES

	CONSTANT GAS FLOWS			INTERMITTENT GAS FLOWS	
	% SO_2	% O_2	% of total S in conc.	% SO_2 (ave.)	% of total S in conc.
INCO Flash (13,14,15)	75-85	0.1	48-65	5-8	35-52
KHD Cyclone (8,9)	60-70	0.1	60-77	6-9	23-40
KHD Cyclone (8,9) with TB converter	65-75	1	100	0	0
OUTOKUMPU (16) O_2 enriched	16-20	0.5-1	70-78	6-9	22-30
MITSUBISHI (17)	12-15	1-3	100	0	0
NORANDA (18,19) O_2 enriched	18-22	0.5-1	78-100	6-9	0-22
TBRC(6,7) O_2 enriched	-	-	-	40-50	100

TABLE 5

ENERGY REQUIREMENT IN PYROMETALLURGICAL COPPER PROCESSES

PROCESS	SMELTING/CONVERTING 10^6 Btu/ton Cu	% ELEC.	TOTAL FOR PROCESS(1) 10^6 Btu/ton CU	% ELEC.	PROCESS DESCRIPTION
TBRC (6,7) without credits	4.1 - 4.5	46-51	7.0 - 7.4	50-53	O_2 enriched, smelting/ converting & slag cleaning in TBRC 20 000 tpy Cu
INCO (13,14,15,20)	3.58-4.43	30-50	6.4 - 7.3	40-53	INCO Flash Furnace 45-54% Cu Matte P.S. converters, slag recycle 150 000 tpy Cu
without credits	3.08-3.93	20-42	5.9 - 6.8	35-49	
with credits (2)					
NORANDA (18,19,20)	4.76-6.41	30-41	7.7 - 9.3	38-49	O_2 enriched 75% Cu Matte Slag milled for cleaning converter slag recycled 200 000 tpy Cu
without credits	3.46-5.56	19-31	6.4 - 8.5	32-43	
with credits (2)					
OUTOKUMPU (16,20)	7.48	18	10.4	29	O_2 enriched, 200°C air 61.5%, Cu matte; P.S. Converters. Slag milled for cleaning 100 000 tpy Cu
without credits	6.60	7	9.6	23	
with credits (2)					
MITSUBISHI (17,20)	6.67-8.77	21-26	9.6-11.8	30-36	65% matte converter slag recycled 70 000 tpy Cu
without credits	5.56-7.66	10-12	8.5-10.6	22-28	
with credits (2)					
KHD (8,9) without credits	9.87	46	11.8	48	70% Cu matte converter slag recycled 15 000 tpy Cu

(1) Includes electrorefining.

(2) Steam credits converted to Kwh at 14,500 Btu/Kwh and reconverted at 3410 Btu/Kwh.

TABLE 6

ENERGY REQUIREMENTS FOR COPPER EXTRACTION PROCESSES

PROCESS	TOTAL ENERGY (10^6 Btu/ton Cu)	% ELEC.
S-C(1)	22.2	54
CYMET(3)	16.2	42
GCM (modified)(4) (estimated)	10-12	40-50
DUVAL(21) (estimated)	17-18	40-50
Most Efficient Pyro Processes with steam credits	6-10	30-50
As above with sulfur production from SO_2	25-29	7-17

TABLE 7

CAPITAL AND OPERATING COSTS OF COPPER EXTRACTION PROCESSES
(MID 1980 DOLLARS)

PROCESS	CAPITAL $/A ton Cu	OPERATING $/ton Cu
INCO	1300 - 1750	160 - 210
OUTOKUMPU	1600 - 2300	180 - 240
NORANDA	1550 - 2200	170 - 210
MITSUBISHI	1500 - 2150	160 - 215
TBRC	1700 - 1800	85 - 115
KHD	1000 - 1400	100 - 175
S-C	1800 - 2500	300 - 340
CYMET (estimated)	1100 - 1500	200 - 300
DUVAL (estimated)	1400 - 2000	250 - 350
GCM (modified) (estimated)	1500	320 - 330

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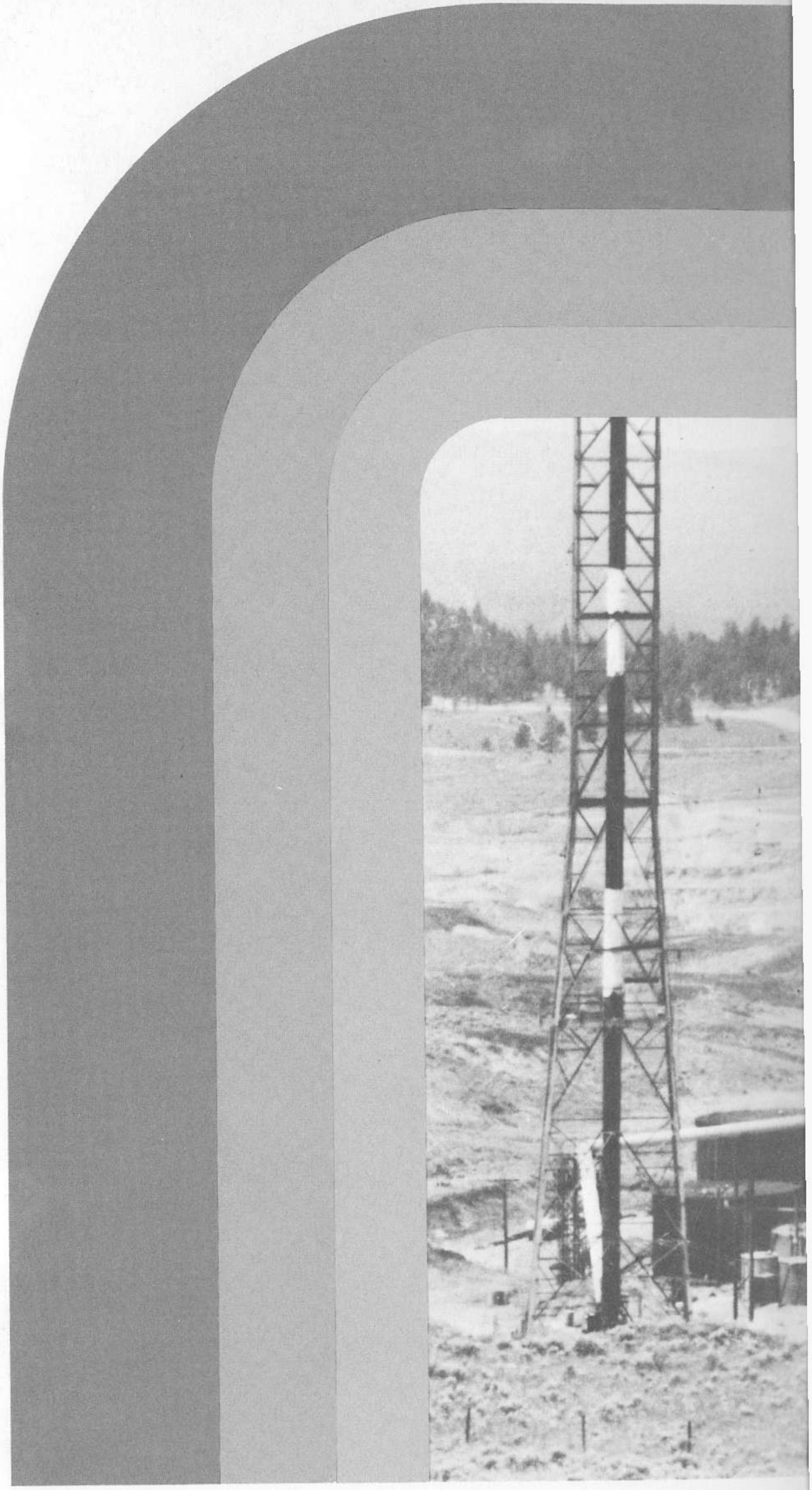

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