OCCASION

This publication has been made available to the public on the occasion of the 50th anniversary of the United Nations Industrial Development Organisation.

DISCLAIMER

FAIR USE POLICY

CONTACT

Please contact publications@unido.org for further information concerning UNIDO publications.

For more information about UNIDO, please visit us at www.unido.org

UNITED NATIONS INDUSTRIAL DEVELOPMENT ORGANIZATION
Vienna International Centre, P.O. Box 300, 1400 Vienna, Austria
Tel: (+43-1) 26026-0 · www.unido.org · unido@unido.org

16522-E

United Nations Industrial Development Organization

Distr.
LIMITED

ID/WG.470/5
24 July 1987

ENGLISH

First Consultation on the
Non-ferrous Metals Industry

Budapest, Hungary,
30 November - 4 December 1987

TECHNOLOGICAL ALTERNATIVES
FOR
COPPER, LEAD, ZINC AND TIN IN DEVELOPING COUNTRIES *

Prepared by

the UNIDO Secretariat

* This document has been reproduced without formal editing.

V.87-88012

Preface

This study presents a hard based survey of existing technologies which are being used in the processing and recovery as well as new innovative patterns of development that are taking place in copper, lead, zinc and tin.

This first analysis of the technologies of these base metals can be a starting point of reflexion for the future selection, changes and creation of technologies in developing countries.

This study is based on the document "Technological Alternatives for Non-Ferrous Base Metals in Developing Countries", prepared by the UNIDO consultant Alexander Sutulov. Annex I is based on the document "Technological alternatives for the Fabrication of Semi-finished and Finished Products of Copper, prepared by the UNIDO Consultant Tamàs Gròf.

Table of Contents

1.0 INTRODUCTION

The world situation of the base metals under study has affected in a great measure the economy of many developing countries where the non-ferrous industries played a key role in the process of development of the economies.

Since the beginning of the 1980's the non-ferrous metals industry was confronted with serious problems such as low prices, which fall almost to the level of prices during the depression, when calculated in constant dollars; high undebtedness of the sector, where in many cases the capital/undebtedness ratio came close to 50 per cent; installed production overcapacity.

These negative events for the non-ferrous industries took place in a process of basic restructuring in the world economy. The principal characteristics of these changes are mainly related with lesser consumption of materials and energy per unit of GNP, which is in many cases related with a more rational use of materials through numaturization, substitution and also due to a greater growth of the services sector as compared with the industrial production.

Under these circumstances, the demand of base metals stagnated and almost without notable exception, the base metals industries started to work with relevant losses since 1981. Developing countries experienced an impotant reduction in the foreign income from the export of these industries, and in many cases even required new loans for servicing the debt generated by the non-ferrous industries. Capital formation in these industries was drastically reduced since the early 1980's.

However, the world economy experienced some changes since the end of 1985. Oil prices started to fall dramatically, which was reflected in lower production costs and increases in the demand for products. Other event is the decrease in interest rates that could contribute to a better balance of payments situation in the developing world. But this will not be sufficient to reactivate their economies. With such huge international undebtedness, may developing countries lack of sufficient savings and income to service debts, less to embark on economic growth of any significance.

The combination of all the factors mentioned requires new strategies of development of the non-ferrous metals in developing countries that can make a better use of their national resources and contribute in a decisive way to the creation of more coherent productive systems, through increasing the linkages of these industries with the other sectors of the national economy. Also promoting complementaris at the subregional and regional levels through co-operation. In this context the selection and improvement of technologies are an important aspect for the effective implementation of new strategies as well as in the efficient operation of the non-ferrous plants.

In general terms, the developing countries have already built different scientific and technological capacity for their own research in all pertinent areas of base metal production. Some of these research centres and laboratories are completely comparable with advanced institutions in the industrial world, although sometimes they need a more specialized staff. It is important, however, to understand that all research in these problems, essentially, should be carried out in local laboratories so that the accumulated experience does not leave these regions with foreign experts when they finish their missions. In this respect, national research centres should enjoy all the support and understanding of their national governments and of international institutions.

We suggest that an initiative be taken in development of autoctonous research effort by interested governments from developing countries, with the assistance of the international organizations, to conduct the development of new technologies and adapt emerging technologies to non-ferrous base metals production with an ultimate goal to adequate them to the size and resources of these countries, improve efficiency and decrease costs. This initiative should also help developing countries to master those technologies and integrate mining and metallurgical activities in non-ferrous base metals horizontally and vertically with other industrial activities, thus contributing to the overall industrial development of their respective countries.

2.0 TECHNOLOGICAL ALTERNATIVES - GENERAL IMPROVEMENTS IN PROCESSING

The progressive depletion of high-grade base metals orebodies and the necessity to mine lower grade ores on a considerably higher scale for the same metal production in the last 15 - 20 years led to a considerable escalation of costs. This happened along with escalation of energy costs, which between 1972 and 1981 increased by 10 times, and with an unprecedented escalation in labor costs, particularly, in the developed nations. In the United States and Canada, for example, the hourly earnings in the copper industry increased from \$ 3.35 - \$ 3.85 per hr in 1970 to \$ 9.20 - \$ 9.50 in 1979 and to \$ 13.00 - \$ 14.00 per hr in 1984. With fringe benefits included these wages were at between \$ 24 and \$ 26 per hour in 1984. This compared with average salaries in Japan of \$ 13 per hr and \$ 2 to \$ 3 per hr in Latin America.

This process of growing costs unfortunately went along with a build-up of a huge oversupply in base metals and a dramatic fall in metals prices. In the beginning this situation was considered as of a temporary character and most producing companies absorbed losses by heavy borrowing. However, such strategies proved to be the end of several of them. Between 1981 and 1985, US copper producers alone lost a staggering \$ 3 billion, while most of the base sector saw itself gradually indebted to the limits with mostly negative profits

It is then not surprising that the sector reacted with drastic action to reduce labor and production costs by streamlining operations and improving technology to reduce materials, energy and labor costs.

In this context, the general trend since the mid-1970's, after the beginning of the energy crisis, was the escalation of size in mining and milling equipment to bring about reductions in operating and maintenance costs (the so called economies of scale); intensification of chemical, pyrometallurgical processes by injection of oxygen, by which reactions were sped up and capacity of furnaces increased; replacement of expensive pyrometallurgical processes with high energy consumption by less expensive hydrometallurgical processes which promote chemical reactions at lower temperatures and with a more thorough treatment of materials. Along with all these changes came automation of controls, computarization of operations and deployment of high technology in planning, development and execution of productive activities with a minimum labor force. The industry started to change its face rapidly through a quiet technological revolution to insure its eventual survival.

2.1 MINING

In mining, technological improvements range from improved and more efficient explosives to more accurate and greater mobility drills. In mine ore handling systems, improvements have ranged from in-pit movable crushers to development of fleets of giant trucks and change to large tonnage transportation (away from trains and trucks) and huge conveyors.

For more effective drilling and blasting, drilling equipment and explosives have been redesigned and improved. Highly efficient all-hydraulic rock drills have replaced pneumatic air drills with a resulting 50% cut in energy consumption and 25% improved performance in terms of efficiency. Large diameter rotary drills have replaced all older types of drills, and rotary drill bits have also been redesigned by using new alloys and special inserts which increase penetration rates, give more uniform drill speeds, less gauge wear and longer life for drill steel and bits.

In blasting, chemical explosives were challenged by nuclear explosives tests, although with understandably great sensitivity towards public opinion and safety risks. In the area of chemical explosives, new metallized ANFO (Ammonium Nitrate Fuel Oil) formulations with varying amounts of metal particles were designed. Such formulations provide a tremendous deployment of chemical energy and have a high degree of water and extreme low temperature resistance. Moreover, metallized blasting agents do not require high explosive sensitizers.

Of particular importance was the increased use of aluminium additives in traditional ammonium-nitrate-based slurries. Such additives impart energy and sensitivity to many explosives. Aluminium powder and granules increase the strength of explosives and are used as a fuel in the production of blast. Aluminium flake powder is being added now to increase the sensitivity of the mix to the initiation of detonation.

Substantial improvements were also achieved in shovels and draglines. Modern shovels are massive, strong and efficient equipment, providing round-the-clock production with 90% to 95% availability over long years of service. They are almost exclusively of the electrical type and have no competition in bank excavation of hard rock minerals where tough digging conditions are an important factor. In the last years a predilection for large 12 to 25 yd electric shovels has been noticed, and they are definitely the prime choice for large, open pit hard rock mines.

Open pit transport of ore and waste is handled today largely by truck haulage, and the importance of this operation is in the fact that it often represents one half of all mining costs. The rapid growth in equipment and parts costs, combined with the unprecedented escalation of fuel prices, forced significant action to cut down operational and maintenance costs. One example was the replacement of 35 and 85 traditional trucks by 100, 170 and 280-ton new trucks with high speed Diesel engines of up to 1,600 hp. Most engines are turbocharged and either aftercooled or intercooled. Lately, a 350 ton low speed Diesel engine truck, with motorized wheels and giant tires, was developed for hauling. Its cost - $ 1 million each.

The use of such huge off-highway rear dump trucks poses some problems in the development of in-mine roads, because such trucks when fully loaded weigh some 600 tons. The power source for these trucks is a 3,300 hp Diesel locomotive engine driving an electric alternator that powers four electric traction motors mounted within the truck's rear wheels. This truck features automatic rear-axle steering.

These trucks have their advantages and disadvantages, which put in question further increments in their sizes. They certainly reduce the labor operating cost but their tire costs are phenomenal. Also, investment cost for a 200-ton truck is likely to be three times higher than for a 100-ton truck. As already mentioned, large truck dimensions adversely affect pit haul road sizes and the size of prime loaders and service facilities. Downtime on the giant trucks is extraordinarily costly in terms of wasted capital and production.

On the positive side, the giant trucks tend to offer productivity increases that exceed capacity multiples. Queuing time at loading and dumping points is reduced, and so is the need for operating and repair personnel.

The high cost of fuel induced a new tendency to switch from conventional fuel to electric energy, and trolley-type trucks were introduced in mines having a cheap hydroelectric power supply, such as the Palabora mine in South Africa. However, the present fall in oil prices may disincentivate this measure for the time being.

The high cost of truck haulage has stimulated in recent years the development of a new portable crusher and belt conveying system for transportation of ore and waste from the open pit. Large scale conveying is not a new technology since it had been practiced already for some years in very different areas. However, its combination with a semiportable or portable crusher, which helps for ready access to the mining area and provides the adequate size of material for conveyor transportation of ore, is of a recent date. This technology is obviously associated with a convenient tunnel access to the pit bottom if the open pit is deep. In practice, a 60-in conveyor can move about 8,000 tons per hour and uses radial tires on the flanged edges of the conveyor belt to provide the driving force. The latest technological development in this area is construction of a single-flight conveyor system for unlimited distances by use of intermediate drive units. This eliminates the cumbersome and cost-escalating transfers of a multiple stage conveyor.

In underground mining, which can be particularly costly when cut and fill or undercut and fill methods are used and orebodies are small and cannot make efficient use of block caving systems, much new innovation is likewise going on. Inco, which used to mine such small orebodies in Ontario, has now introduced vertical retreat mining (VRM) - a bulk mining method which is less labor intensive, safer and less costly than traditional techniques. VRM has made possible the introduction of crater blasting and the use of in-the-hole (ITH) drills. Also, it has eliminated slot raise boring and slot blasting, with improved fragmentation of the ore. The next generation of mining equipment is designed for continuous mining, which includes advanced mechanical handling devices, electrification, automation and microprocessors. Operations are also improved by improved ground control and blasting techniques. Inco has set up a company by the name Continuous

Mining Systems Ltd. to design, manufacture and market its prototype equipment line, which in addition to its ITH drill, includes a continuous loading machine and battery-powered, remote-controlled underground locomotive. It is also testing a new low-profile, portable, underground jaw crusher.

Remote controls allow for simple operation and control from safe locations at the rear, left or right side of the rig. They also permit one miner to operate more than one drill at a time. These drills now have hydraulic control of the drill string, which provides greater drilling accuracy.

The continuous loading machine is reportedly capable of handling up to 1,000 tph of broken muck, or 10 times that of conventional LHD's, by means of a patented oscillating lip connected to the conveyor. As the machine moves into the muckpile, the lip fluidizes the material by changing the angle of repose of pile. This reduces the penetration force and regulates the material flow onto a short chain conveyor while leaving the roadted clean.

To promote materials handling efficiency, a portable jaw-crushing plant was stationed at a stoping area. The plant, which costs only 10% of a stationary underground crusher, can be disassembled and reassembled for fast and easy access to reach working levels.

These innovations certainly result in substantial reduction of costs. Here is how progressive technological improvement has reduced mining costs at a large copper mine in the last six years, expressed in dollars per ton of ore:

	1980	1981	1982	1983	1984	1985
Drilling	0.25	0.22	0.12	0.10	0.12	0.11
Blasting	0.66	0.63	0.32	0.30	0.32	0.32
Loading	0.55	0.54	0.30	0.26	0.40	0.39
Hauling	1.72	1.60	1.21	1.06	1.15	1.10
Crushing & Stockpiling	0.48	0.49	0.42	0.40	0.39	0.40
Others	0.51	0.48	0.41	0.45	0.46	0.45
TOTAL $/T	4.17	3.96	2.78	2.57	2.84	2.77

Some of these costs were also influenced by exchange rates policies between the national currency and the US dollar, but on the whole the change in costs was mainly influenced by higher efficiency of labor and more effective use of materials and technology.

The mining cost at different Chilean mines in dollars per metric ton of ore experienced the following changes:

	1980	1981	1982	1983	1984	1985
Chuquicamata	4.17	3.96	2.78	2.57	2.84	2.77
El Teniente	2.70	2.55	2.98	2.63	2.40	2.08
El Salvador	4.86	4.05	3.25	2.73	2.50	2.04
Andina	2.88	2.65	2.39	3.35	3.84	2.59
Overall	3.88	3.80	2.89	2.67	2.74	2.43

2.2 COMMINUTION

Comminution operations account for over 50% of overall milling (crushing-grinding-classification-concentration-dewatering) costs. They are energy and materials intensive, with problems of high grinding media consumption and of overgrinding, which results in operationally and metallurgically adverse effects on the mill operation and its recoveries.

Classification operations generally include intermediate comminution operations after removal of material already reduced to a specified size, thus avoiding overgrinding, production of undesirable fines, and excessive consumption of energy, while improving metallurgical recovery in the concentrating processes.

In this respect, in the last two decades a fundamental change in classification technology has been taking place almost universally by replacing rake and bowl classifier technology with hydrocyclones. The Dutch hydrocyclone was originally developed for wet classification of coals but since mid 1950's has found an increasingly attractive role in hard rock grinding and classification, thus replacing almost completely the traditional classifiers. The tremendous advantages of hydrocyclones are their effective classification and separation of fines, low consumption of spare parts, very small size of equipment, permitting to double grinding capacity under the same roof, and their easy adaptation to automatic controls of the grinding circuit.

The principle of operation of a hydrocyclone is given in Fig. 1, and its automatic operational control is presented in Diagram 1. The ground pulp, discharged by a ball mill, is fed tangentially in a cyclone cylinder under a certain pressure and, due to centrifugal forces, segregates into a coarser product, which descends along the walls of the cone as an underflow, while the fine fraction, close to the air core, is propelled as an overflow. The system can be automatically controlled by keeping the apex orifice at a constant underflow density, which determines the overflow size. In large-scale operations, cyclones are used in batteries of several units.

By cutting down on the overgrinding of ores, hydrocyclones significantly reduce energy consumption in these circuits, are easy to control and operate and thus represent a very convenient substitute for classical classifiers. In most plants the change from classical classifiers to hydrocyclones has already taken place. However, in many developing countries, and particularly in old mills, traditional classifiers are still being used. This offers an opportunity for significant improvements in mill metallurgy and costs as well as in creation of extra floor space for grinding capacity expansion.

In comminution itself important advances have also been made. It was always a millman's dream to reduce crushing and grinding costs, which represent close to 50% of the total mineral processing cost. In this context, two principal problems attracted attention: (a) possible reduction in energy consumption; and (b) minimization of steel consumption in crushing and grinding processes.

In respect to the first question, it was established that actually only a very small fraction of the energy consumed in a comminution process really goes into size reduction of particles, while most of it - probably 98-99% - is dissipated in the form of thermal energy, which is caused by the fast comeback and reunification of molecular and ionic bonds broken in the comminution process. Attempts to solve this problem by addition of special chemicals to impede such reunification gave only minor results, thus permitting economies of only between 7 and 10 percent of the energy.

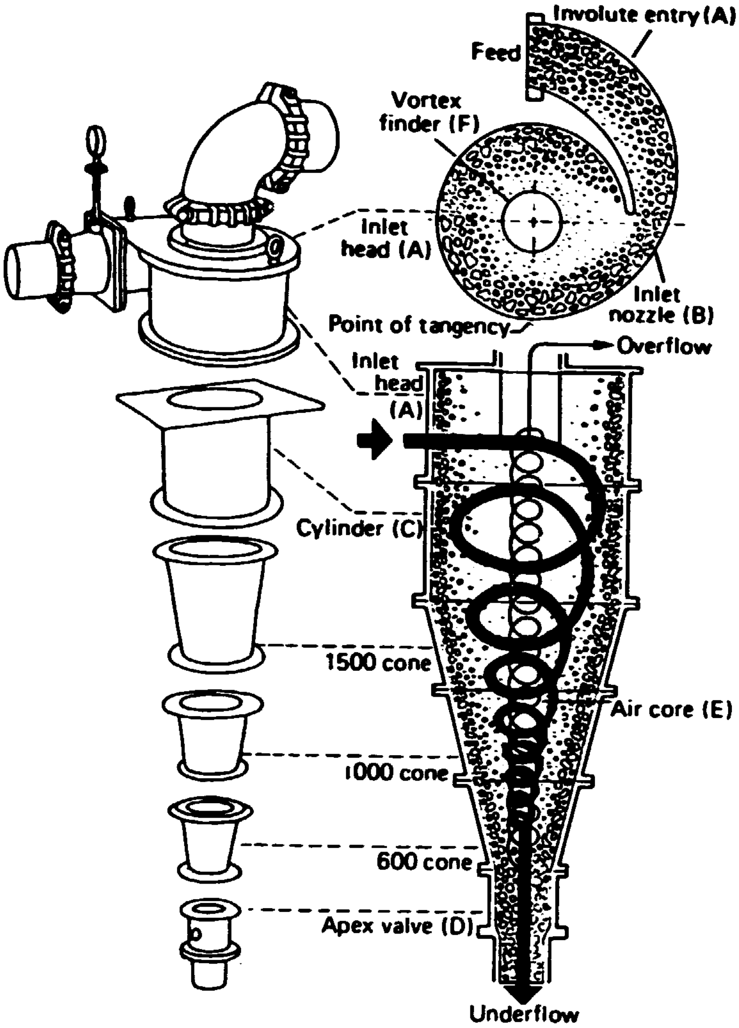


Fig. 1 : Schematic view of Webco Hydrocyclone

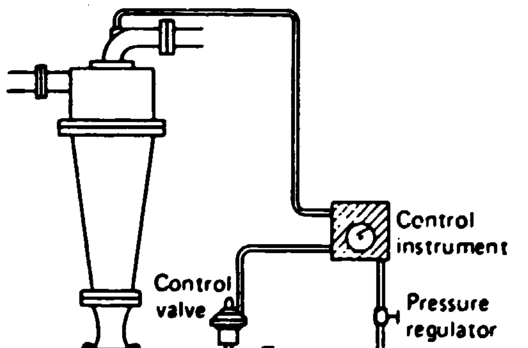


Diagram 1 : Automatic control of apex orifice for constant underflow density control.

Attention was then focused on simplifying the crushing and grinding system itself by modifying traditional tumbling mills and changing from steel grinding ball charges to use of the rock itself. In the past, the so called pebble mills using river pebbles as a grinding media were rather popular, because they saved on steel consumption as rods and balls and, more importantly, because they avoided contamination of pulp with iron. This originated the idea of autogenous grinding, which is the grinding of ore by itself rather than by special metallic or nonmetallic grinding bodies distinct from the ore. The first attempts to introduce these mills industrially were made by the gold industry in South Africa. Here, the so called large diameter AEROFALL autogenous mills were introduced in the late 1950's and early 1960's with a great success. However, autogenous grinding is not always successful for crushing and grinding big chunks of rocks by itself, due to a deficiency in the rock media, or when frequent changes in quality of media occur. In this case in order to assure smooth operation, large steel grinding balls are added in quantities between 2% and 10% of the total volume. Currently, most of such semi-autogenous mills carry less than 5% by volume of such steel balls, while the traditional ball mills use a 45% by volume ball charge.

The introduction of steel balls to autogenous mills is intended to reduce the total energy required to grind a primary crusher product, or possibly run-of-mine ore, to a manageable size for further processing. The product of a semi-autogenous mill may be ground further in a ball or pebble mill, or may be a finished product ready for flotation or other concentrating operations. In most porphyry copper operations today, which treat such large tonnages as between 20,000 and 150,000 tons per day, semi-autogenous mills in fact replace secondary and tertiary crushing and the rod mill grinding stage, as indicated in Diagram 2.

In practical terms, in Chile for example, the introduction of semi-autogenous mills instead of the classical three-stage crushing and two-stage grinding circuit means reductions of the grinding cost, including energy savings, grinding media and other costs, from a level of $ 1.51 per ton to only about $ 0.99 per ton, i.e., by fully one third, as indicated in attached Table 1 .

The concept of semi-autogenous grinding (SAG) holds promise for simpler circuits, elimination of troublesome fine crushing problems, and a possibly low-cost treatment of otherwise uneconomic ores. To mention a few examples, we can evoke past experience with SAG circuits at the Pima and Bagdad mills in Arizona, Henderson mill in Colorado, Gibraltar and Lornex mills in British Columbia and Chuquicamata and the envisaged at the planned Escondida mine in Chile. The Los Bronces project, carried out by Disputada, a subsidiary of Exxon, also provided for use of SAG grinding.

Given the importance of porphyry copper deposits in Latin America and other developing countries, it seems of great significance to pay adequate attention to this technological improvement which will certainly reduce not only production costs but also investment costs. The problem requires careful testing and consideration of rock properties for adequate use of this technology. This can be carried out locally in properly equipped laboratories. In Chile, for example, there is already important experience in this respect.

TABLE 1

COMPARATIVE COSTS : STANDARD VS AUTOGENOUS GRINDING

(all costs in US $/ met. ton)

	STANDARD TWO STAGE GRINDING			SEMINAUTOGENOUS GRINDING		
	Volume	Unit Cost	Total Cost	Volume	Unit Cost	Total Cost
Energy Consumption	8.3 kwh	$ 0.045	$ 0.373	7.4	$ 0.045	$ 0.333
Liners Consumption	0.091 kg	$ 1.92/kg	$ 0.175	0.083	$ 1.92/kg	$ 0.159
Balls Consumption	0.83 kg	$ 0.71/kg	$ 0.589	0.11 kg	$ 0.71/kg	$ 0.078
Labor & Supervision			$ 0.077			$ 0.065
Depreciation & Amort.			$ 0.210			$ 0.320
Maintenance Cost			$ 0.081			$ 0.035
Total cost per ton of ore			$ 1.505			$ 0.990

DIAGRAM 2

CONVENTIONAL VS SEMIAUTOGENOUS GRINDING

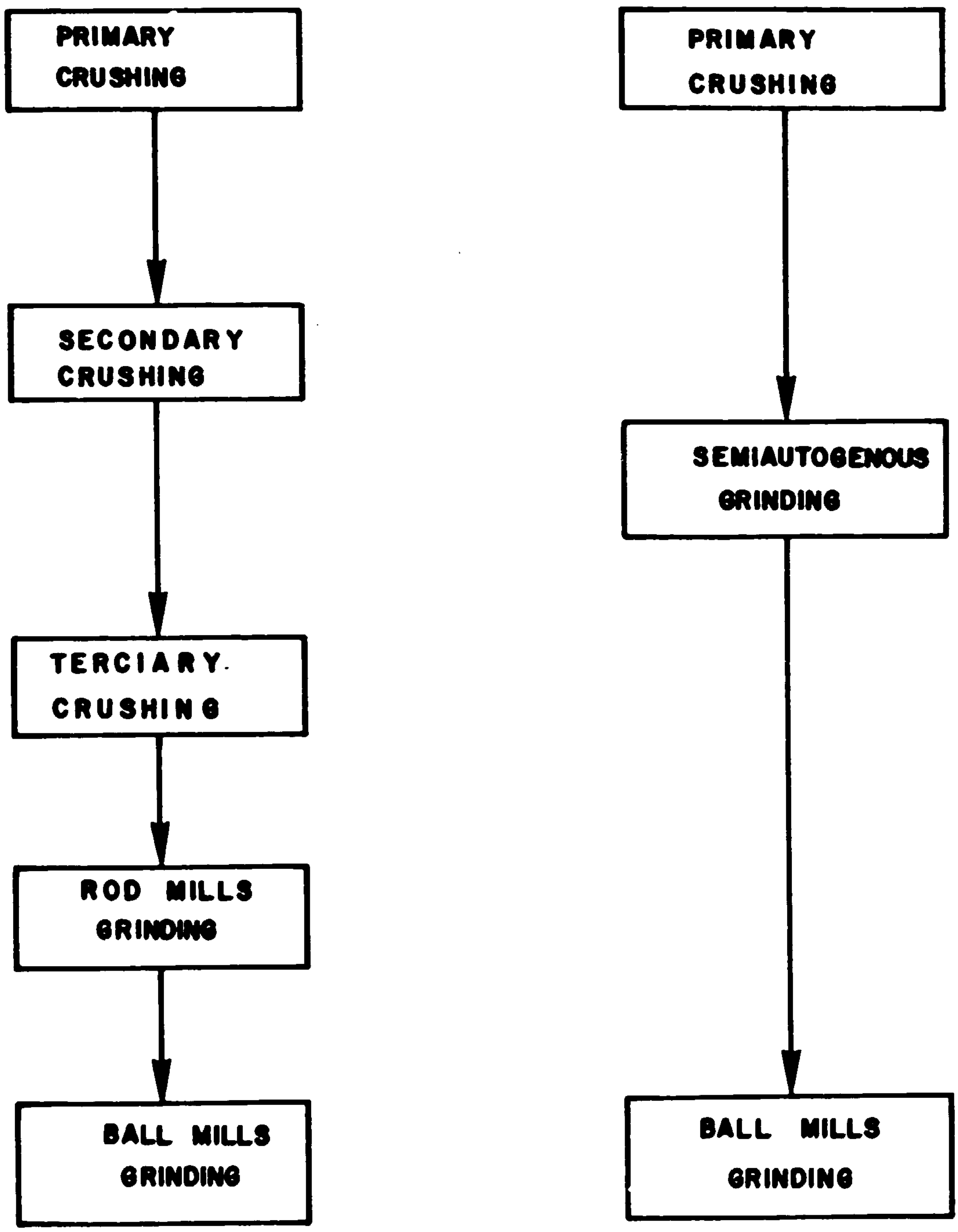


Fig. 2 and Diagram 3 give us, respectively, a cross-section of the Aerofall mill and how this system operates in a dry circuit. The feed is directly fed to the mill, which autogenously grinds by a fast rotation. A draft fan, located above the system, drives the finely ground material through a vertical classifier and cyclone collectors, while excess air and very finely dispersed particles are driven to an exhaust system. Draft air is preheated in order to keep the material dry. The flowsheet of a closed circuit Aerofall system is shown in Diagram 4.

In Diagram 5, a typical setup for a semiautogenous grinding flowsheet is given, as is normally tested in research laboratories on different ores, while in Diagram 6 we see this semiautogenous flowsheet applied in a large US copper plant.

General progress in milling technologies at Chilean copper mines in these last years, which affect both comminution and flotation costs in terms of dollars per ton of ore, can be appreciated from the following figures:

	1980	1981	1982	1983	1984	1985
Chuquicamata	3.65	3.16	2.06	2.07	2.12	1.86
El Teniente	3.33	2.72	2.43	2.09	2.09	1.84
El Salvador	4.15	2.94	2.15	2.10	2.04	1.91
Andina	3.37	2.70	2.14	2.07	2.05	2.16
Overall	3.40	3.20	2.20	2.08	2.09	1.89

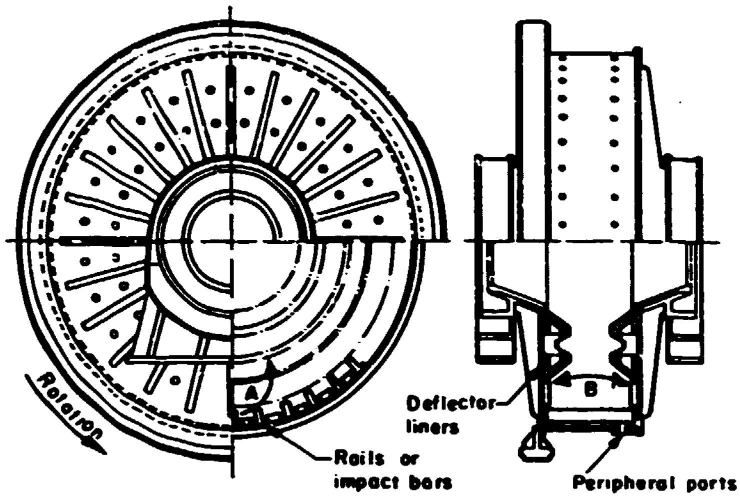


Fig. 2 : Cross-section of an Aerofall mill

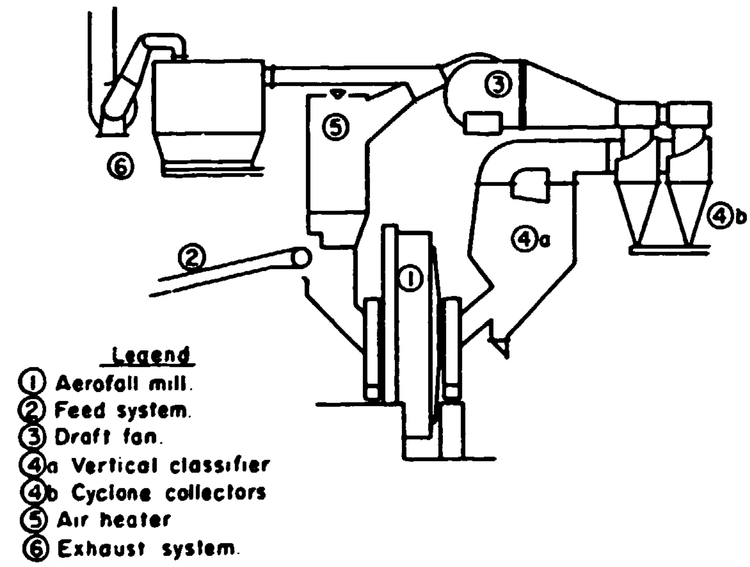


Diagram 3 : Operation of an Aerofall system

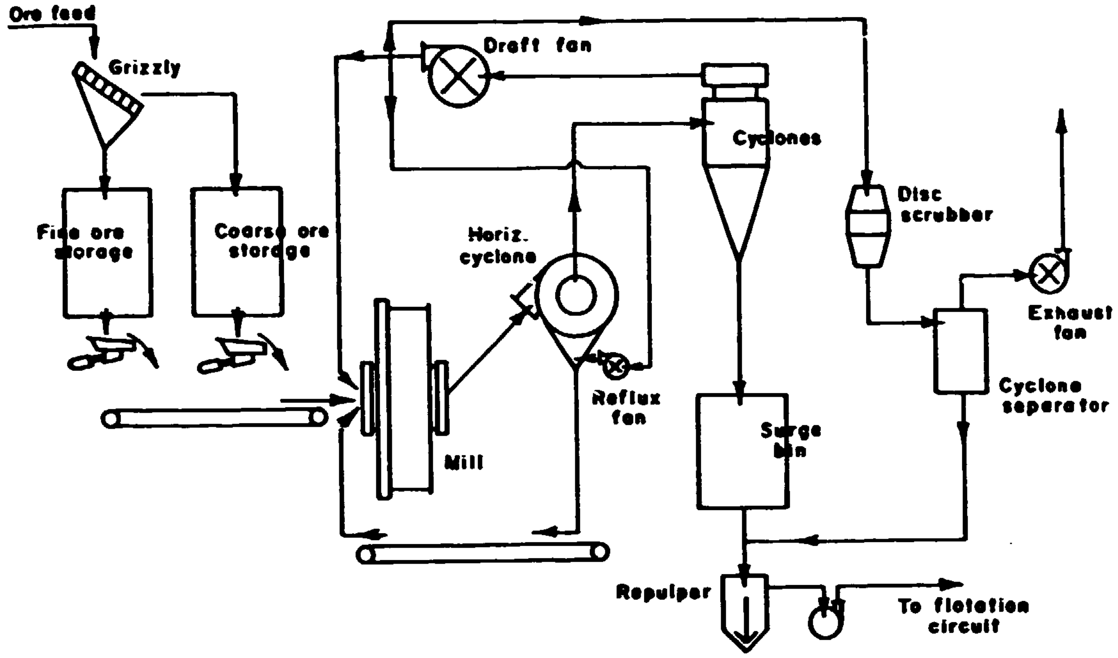


Diagram 4 : Flowsheet of a closed circuit grinding system

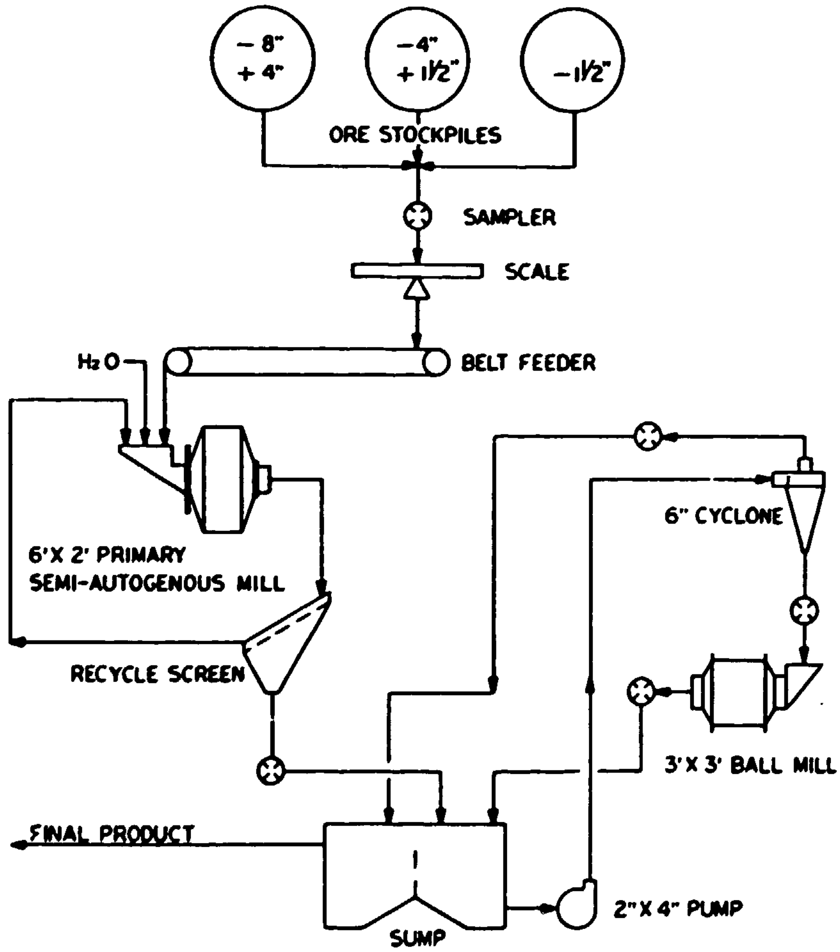


Diagram 5 : Flowsheet of a Semi-Autogenous Mill Operation

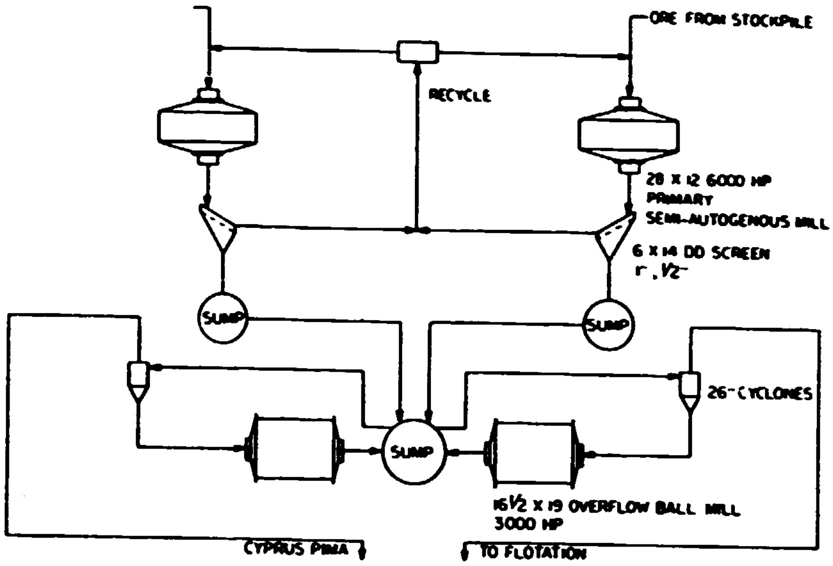


Diagram 6 : Semi-Autogenous Grinding flowsheet at Pima Mine

2.3 CONCENTRATION

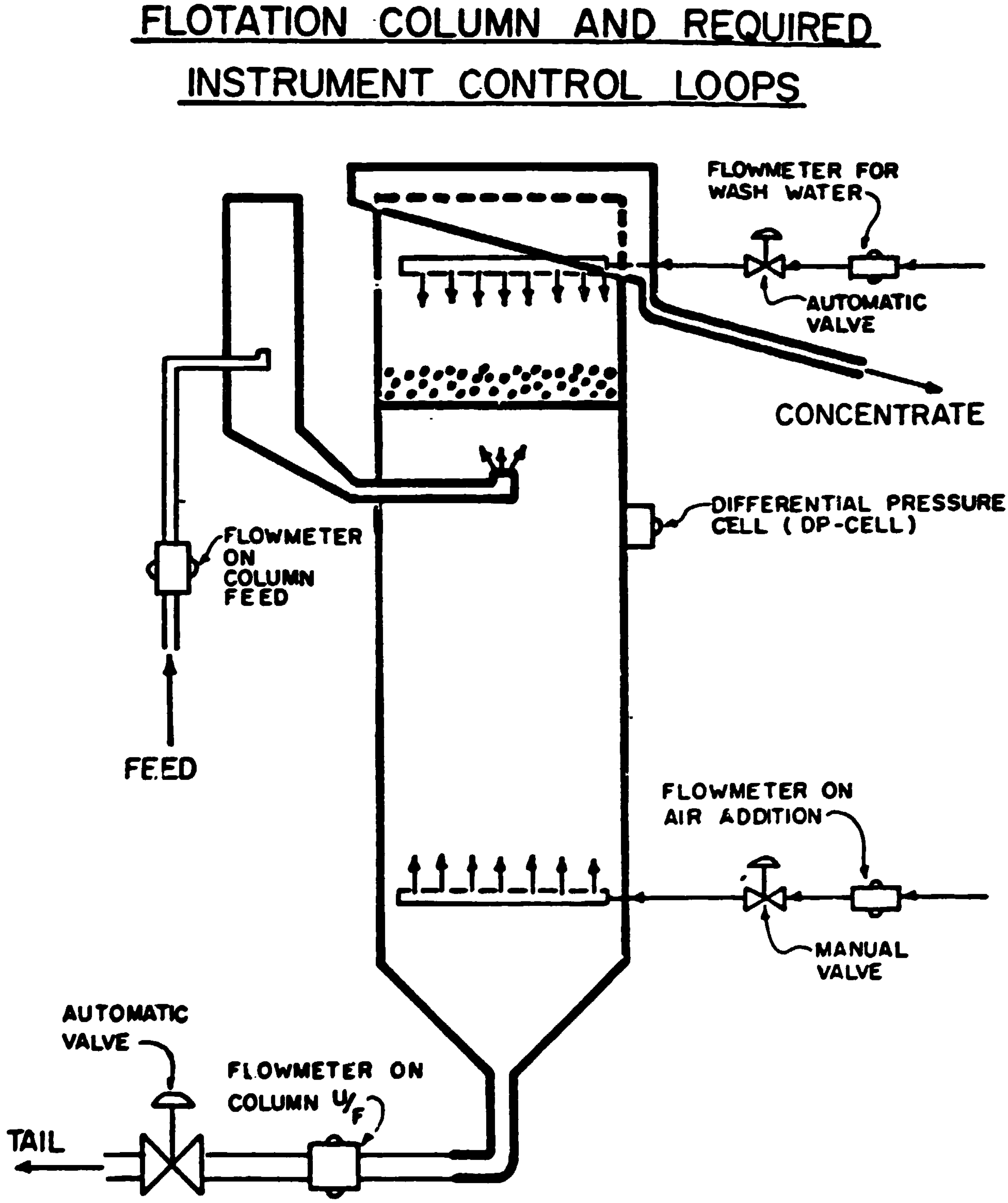
In the area of concentration, general progress has been developing in the escalation of equipment size, particularly of flotation machines which have now reached colossal dimensions of 1,000 cubic feet and more (up to 1,250 cu ft so far). This compares with standard machines of only 100 to 350 cu ft only a few years ago. The technological impact of giant cells is in their reduction of operating and maintenance costs, but basically not much innovation is involved.

However, a technologically new system for replacement of traditional flotation cells was developed in these last years in Canada. This is the so called column flotation. The idea offers a number of advantages in the separation of different minerals, particularly in the copper industry and in the area of byproduct molybdenum recovery. Also, this apparatus has been used in flotation of chromite and fluorite ores and has even been tested on recovery of copper from its ores and concentrates. In this context, it has a great potential which, however, has still to be assessed. This apparatus is apparently most promising for concentration and separation of very fine products.

The principle on which column flotation operates can be appreciated from Fig. 3. The column is approximately 12 m long and 1 m in diameter. The flotation column feed enters approximately 3 meters beneath the overflow lip and comes into contact with a bed of rising bubbles. Consequently, chemically treated minerals (by flotation reagents) float and gangue or depressed minerals settle out of the column. The mineral laden bubbles rise out of the pulp into a froth column, which is maintained by a gently percolating stream of wash water that cleans the froth before it reaches the overflow lip.

The principal characteristic of this column is that it has no

Figure 3


FLOTATION COLUMN AND REQUIRED
INSTRUMENT CONTROL LOOPS
FLOWMETER FOR
WASH WATER
AUTOMATIC
VALVE
CONCENTRATE
DIFFERENTIAL PRESSURE
CELL (DP-CELL)
FLOWMETER
ON
COLUMN
FEED
FEED
FLOWMETER ON
AIR ADDITION
MANUAL
VALVE
AUTOMATIC
VALVE
FLOWMETER ON
COLUMN U/F
TAIL

moving parts, and solids are kept in suspension by rising bubbles alone. Spray pipes are installed a few inches below the froth surface. The purpose of this water is twofold: first, it keeps in balance the flow of material through the column; second, it washes the rising froth and removes unwanted fines attached to the bubbles in the lower flotation section. As a result, the concentrate produced is of higher purity than that from conventional flotation. However, even temporary disruptions of wash water result in a drop in concentrate grade.

There are also a number of instrumental control loops which must be operational for the proper functioning of column. Magnetic flowmeters are installed on the feed and tailings line, while the wash water line is fitted with an orifice plate. The readings are continuously recorded on graph charts on the instrument panel. Automatic valves on the tailings and wash water line operate in a loop with the flowmeters and control the respective flows. Any change in the feed flowrate results in a corresponding change in the tails discharge rate and also affect the volume of wash water entering the column.

The principal characteristics of this column are its simplicity and efficiency, along with easy and complete control of the operation. Similar to a modern day petrochemical processes, this column represents a complete break away from the conventional manner of floating on a batch basis, which is characteristic of existing flotation cells.

When used as a cleaner, one pass through the column is the equivalent of four or more stages of conventional cleaning. In the first commercial operation in Canada, three columns of this type replaced 13 stages of cell cleaning and resulted in better grades and recovery.

When accepting normal flotation feed to produce both final tail and concentrate, one column is sufficient for a simple ore. With a complex ore requiring roughing, scavenging and 3-4 stages of cell cleaning, two columns in closed circuit will produce superior metallurgy. At a massive sulphide chalcopyrite copper operation in Peru, a rougher-scavenger column together with a closed circuit cleaner column were tested against a conventional plant flowsheet consisting of one stage of roughing, two of scavenging and four stages of cell cleaning. The column produced 4.8% higher grade copper concentrates and improved by 0.4% the overall copper recovery.

2.4 HYDROMETALLURGY

In a recent article, Mining Journal of London commented that in spite of the fact that mineral leaching technology has not found universal acceptance for a variety of reasons, there are indications that this position is rapidly changing. The reason for this is that the mining industry continues to suffer from weak metal prices and needs to reduce operating costs as a vital step for survival. In this respect, leaching technology proved to be a safe, efficient and cost effective method for a number of metals, which include copper, gold and uranium. These chemical processes can be carried out in a number of ways, starting with in situ leaching, when fragmented and fractured material is not excavated, or by the heap leaching method when the broken rock is dumped on specially prepared pads and sprinkled with leach liquors which are conveniently recirculated. Leaching can be carried out also at atmospheric pressure or in closed vessels at elevated temperature and pressure. It can be purely chemical, using acid, caustic soda or cyanide, or biological using particular strains of bacteria. The bacteria do not actually leach the materials but rather render them amenable to subsequent chemical leach by speeding up the oxidation of the sulphide minerals.

Leaching is relatively cheap and simple technology, easy to introduce since it requires little in the way of sophisticated equipment. This is particularly important in view of the circumstances under which the industry has to operate today. Furthermore, the low capital cost of the process can act as an incentive for small mine development. Also, the attractive feature of these operations is that they can be constructed in a short period of time - a few months in some instances instead of the years required for other large operations. These operations have low operating costs due to their low energy consumption. They do not require finely ground material, but rather operate with crushed ore. Also, they are continuous operations which require little supervision and maintenance.

Although these operations use toxic chemicals for processing of ores, they are relatively clean and are conducted in closed circuits, by which they produce a minimum disturbance to environment in terms of noise, atmospheric discharge or obvious contamination. In this respect they are often a viable alternative to pyrometallurgical methods for the elimination of sulphur, arsenic and other contaminants from ores.

Since most of the world's copper is being produced at costs which range from 40 to 80 cents per pound, with the present copper prices of 65 c/lb many sulphide flotation plants, particularly in the USA and Canada, have been forced out of business. Also, many other producers with lower costs remain in the critical range of between 62 and 70 c/lb, which make them very marginal under the present conditions. This explains then why so many copper producers are now rapidly switching to hydrometallurgical methods where they can produce copper considerably cheaper at between 30 and 45 c/lb.

Essentially, what is happening is that the old classical sulphide copper flowsheet, consisting of crushing, grinding, flotation, dewatering, smelting and refining is being replaced by crushing, leaching and electrowinning, through which the expensive stages of fine grinding, flotation and smelting are eliminated with the subsequent reduction in processing costs.

Leach liquors obtained either by in situ, heap, vat or agitation leaching are conducted for purification to a solvent extraction unit (to be described later) and then subjected to reduction to the metallic state by electrowinning (to copper cathode).

These alternatives have been extensively tested all around the world and now have been introduced in the United States, Canada, Chile, Peru, Zambia, Zaire and many other countries. In the United States, Kennecott operates such a plant in Utah where it produces 30,000 tpy of copper. Phelps Dodge also operates such a plant at Tyron in New Mexico with a 20,000 tpy Cu output and is constructing a new one at Morenci in Arizona with a potential production of 36,000 tpy of Cu. Newmont is now in the process of constructing a $ 70 million plant at San Manuel in Arizona, which will produce 22,500 tpy Cu with a cost of about 40 c/lb. This plant will treat an oxide ore in situ. However, another project at the same site provides for leaching of a mixed sulphide-oxide ore at a 150 million ton orebody on a larger scale. In situ leaching here is also favored costwise over heap leaching.

In Canada, these techniques have been used at Gibraltar in British Columbia, where bacterial leaching yields daily about 14 tons of copper cathodes. In Chile, such a plant was installed at El Teniente to treat mine waters. It produces about 4,000 tpy of cathodes. A considerably larger plant is being projected at Chuquicamata. Eventually it will produce about 250,000 tpy of copper cathodes by treating both oxide and sulphide ores.

This type of plant, described in detail in the chapter on copper, is particularly convenient for developing countries since it requires minimum capital, is cheap and fast to construct and can be amortized very quickly. Such plants have low operating costs and require a minimum of supervision. They are particularly favorable for treatment of copper oxide ores, although they can be used also for treatment of sulphides. In Zaire, such a plant was constructed at Shitura and produces 125,000 tpy of copper cathodes. In Luilu, also in Zaire, Gecamines operates a similar plant at 90,000 tpy of copper and in Chambishi, Zambia, such a plant produces 18,000 tpy of copper.

2.5 PYROMETALLURGY

More than anywhere else, cost cutting technologies have spread in the pyrometallurgical area, where costs are high because of high energy consumption. On an overall basis, one ton of copper requires an average consumption of an equivalent to 2,400 kwh of energy, of which mining consumes 21.8% (explosives - 11.4% and transportation 10.4%), milling requires another 17.8% (of which comminution is 15% and flotation and other operations only 2.8%), while smelting and refining require fully 60.4%, of which smelting is 41.4%.

This explains then why the conventional comminution-concentration-smelting sequence, which consumes almost 60% of the energy, is often bypassed and replaced by hydrometallurgical means.

The other alternative is to increase the efficiency of pyrometallurgical operations. One way to do this is to decrease the temperature of conversion of minerals into metal, such as is accomplished in the Segregation Process explained in the copper chapter. The other method is to intensify the process with oxygen injection, through which reactions are sped up, capacity of furnaces increases and products are obtained at a lower cost. Here, obviously, the cost of obtaining oxygen is compared with savings in the pyrometallurgical cost. Also, some other methods of energy savings, such as preheating of air with exit gases or by some other means, are in practice.

Another advantage of oxygen smelting techniques is that they produce SO_2 gases of 3 to 4 times higher concentration than do conventional converting operations, and almost 10 times higher than conventional smelting operations. This permits, first, to solve some important ecological problems and, second, to obtain high quality sulphur products, such as liquid SO_2 or sulphuric acid.

Historically, the traditional schemes treated sulphide ores of base metals by first roasting them to oxides and then reducing them with carbon to metal. These schemes were then replaced by conventional concentrate smelting and converting processes, when higher grade materials originating from flotation concentration of ores became available. Now, the evolution is to new alternatives where energy saving, autogenous smelting. using the sulphur of concentrates as a fuel, are becoming more and more popular. In the next chapters we will analyze these alternatives for the different base metals.

3.0 COPPER TECHNOLOGIES

Copper is certainly the most important base metal in developing countries' production today, and it holds a prominent place in their future development. Most of Latin America - Chile, Peru, Argentina, Mexico, Brazil, Panama, Colombia, Ecuador and a number of smaller countries - hold large copper resources. It is calculated that Latin America as a whole holds about 35 to 40 percent of the world copper resources.

In these last years, the copper business has been under a strong pressure from oversupply and low prices, which has made it extremely competitive. Technological improvements in copper production, byproduct recovery and in more effective use of labor work has played a key role in reduction of copper production costs. Since the outlook for copper prices in the next decade is rather pessimistic, technological improvements rather than improvements in copper prices should be emphasized from now on. This makes research and development activities in this area vital for survival of the great mineral complexes in South America, North America and Central Africa.

Fortunately, the bulk of copper deposits in developing countries belongs basically to one of the following two large groups: (a) porphyry coppers, which are characteristic for the American continent; and (b) sedimentary copper deposits, which are characteristic for the African continent. In both cases, findings obtained through research and experimentation in one area can greatly benefit the other. This permits classification of all research effort in big groups which will benefit many clients.

For example, copper porphyries account today for about 45% of the world's total copper reserves. They are spread throughout the Western Coast of America from Antarctica to Alaska, and indeed they form a ring around the Pacific Ocean, which includes the Western Coast of Americas, parts of Siberia, China and the Philippines, Indonesia, Papua New Guinea and other islands, forming the so called Fire Ring. Also, they penetrate through Central Asia (Kazakhstan, Uzbekstan, Iran) through Asia Menor into the Balkans, where they appear in the form of low-grade porphyries in Yugoslavia, Bulgaria and Rumania. All told there are more than 100 of these huge orebodies, whose general characteristics are as follows:

(1) Low-grade disseminated mineralization

(2) Occurrence in a large volume of rock - generally between 100 and 1,000 million tons, although mega deposits go up to 4 - 5 billion tons

(3) Substantial byproduct credits, principally for molybdenum, gold and silver

(4) Stratified positioning of three essential zones: (a) oxidized cap, (b) secondary enrichment, (c) primary sulphide zone

These common characteristics of these deposits have helped to typify mining and metallurgical strategies for their exploitation. For example, recovery of copper from oxidized caps uses common solvent extraction and electrowinning methods, which have become very popular in these last years of low copper prices and which permit some mines to survive. Also, an effective byproduct molybdenum recovery technology, including generalized use of sodium hydrosulphide as the principal depressor of copper in a nitrogen atmosphere, have helped to cut down sharply on production costs. The same is true for the brand new "column flotation" technology, which is rapidly spreading from Canada where it was originally introduced.

In this chapter, we will cover all important innovations carried out in recent years. Obviously, no space is given to refer to more traditional technologies presently in use. But it can be said that the material covered in this chapter, i.e., SX/EW Process, Bacterial Leaching, Column Flotation and Sulph-Hydrate/Nitrogen technology, may have vast repercussions at many copper mines. This is apart from several pyrometallurgical processes, such as INCO and OUTOKUMPU flash smelting, oxygen smelting with NORANDA and EL TENIENTE processes and some others.

3.1 COLUMN FLOTATION

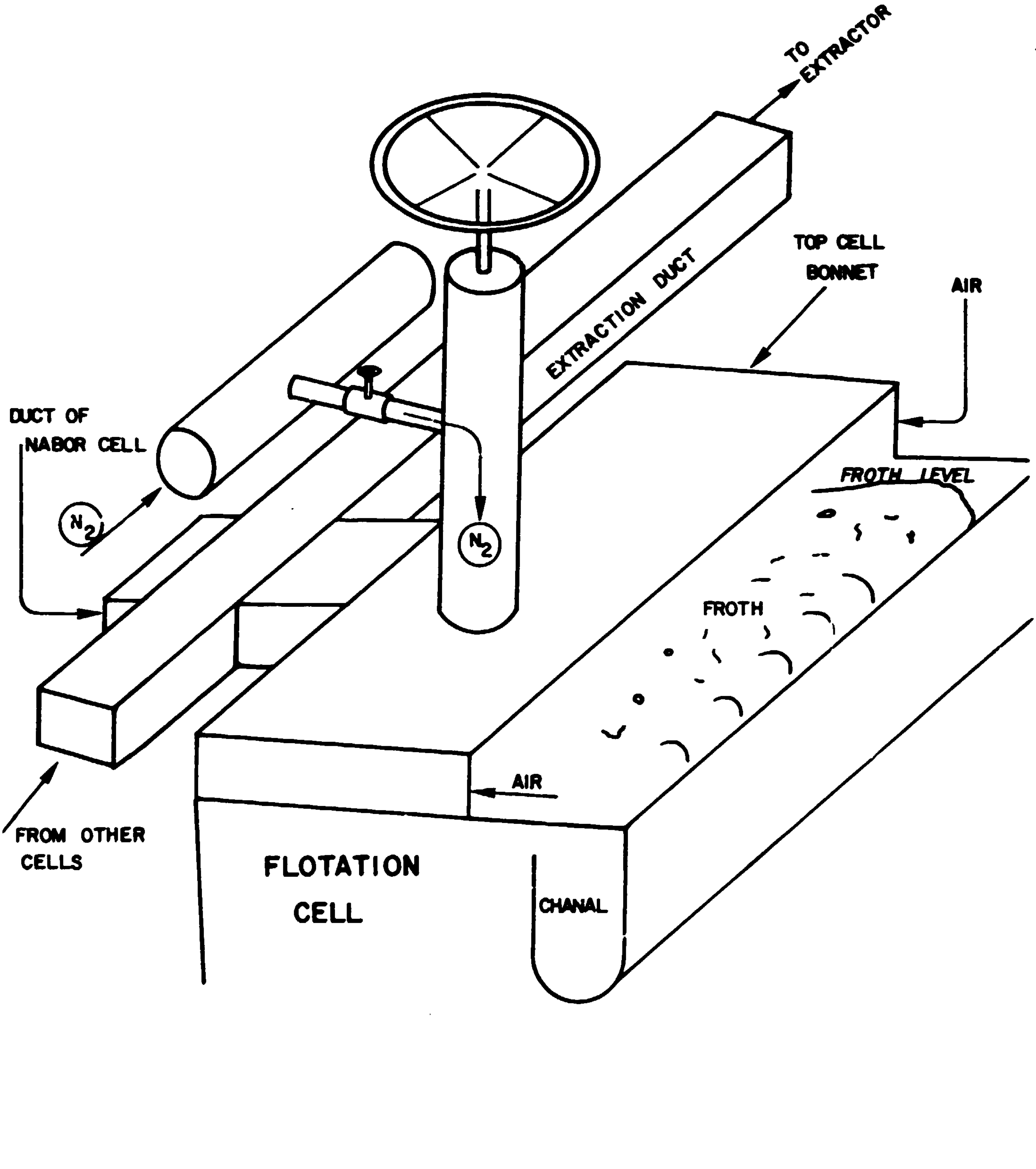
World molybdenum production today amounts to about 200 - 210 million pounds per year, of which roughly 40% is produced by primary molybdenum mines while the other 60% is recovered as a byproduct of copper. These 120 million pounds of molybdenum per year are worth roughly $ 350 million at present molybdenum prices, but in the past their value was from two to eight times higher and represented a formidable byproduct credit towards copper production costs. The essential point in byproduct molybdenum recovery from copper concentrates is to achieve a low cost of production in order to be able to subsidize effectively low copper prices. These costs highly depend on the cost of reagents, which are now about 65 percent of the total operating cost, and on the process cost which represents the balance.

Flotation reagent cost and its principal item which is the cost of copper depressor has been rationalized by introduction of sodium sulph-hydrate depressor and its use in neutral (nitrogen) atmosphere. Sodium sulph-hydrate has an instant depressive action on copper sulphide, which form the bulk of concentrates from which molybdenum byproduct is extracted. However, the oxygen from air, present in the bubbles of froth flotation, also attacks sodium sulph-hydrate and decomposes it. By replacing air with nitrogen, consumption of the depressor can be roughly halved and molybdenum recoveries improved. A rough sketch of this technology as operated in Chile and Peru is given in Fig. 4.

The other important component of this new technology is the use of column flotation equipment, whose action has been already explained. By introducing these columns instead of the standard flotation cells, operations have been greatly simplified and excellent metallurgical results obtained. In the first place, operation of these columns is much smoother than that of regular cells. Second, they occupy much less floor space, which

FIG. 4

OPERATION OF A FLOTATION CELL
ON NITROGEN



is helpful for expansion of operations. Then, they produce an excellent froth and high-grade molybdenite concentrates just in a couple of steps, where previously numerous and lengthy counter-current circuits (up to 12) had to be applied. Finally, molybdenum recoveries are higher, and consumption of reagents is significantly lower.

These columns were first introduced in Canada, where they have been installed at Gaspé, Lornex and Highmont mines. Although experimentation is going on in Chile with the distinct possibility of installing them at Chuquicamata. El Teniente, El Salvador and Andina mines. The significance of this technology lays in the fact that it can be potentially used in all developing countries which are developing byproduct molybdenum recovery, thus resulting in a considerable economic benefit. In attached Diagrams 7, 8 and 9 , we show how the conventional molybdenum plant circuit at Gaspé (Quebec) has been replaced by alternate circuits with 2 and 3 flotation columns.

In all cases, retreatment of copper-molybdenum concentrate starts with thickening of the concentrate to remove excess flotation reagents from bulk flotation and treatment of the concentrate with activated carbon to absorb excess remaining reagents. Sodium sulph-hydrate is added to the pulp to depress copper values and fuel oil to activate molybdenite for flotation. After the discharge of the copper concentrate, which is now flotation tail in the primary circuit, upgrading of the molybdenite concentrate is necessary in a counter-current treatment. As indicated in Diagram 9, in the original flowsheet this required 11 consecutive counter-current steps. But with column flotation, as in Diagram 10, two columns replace 6 counter-current steps and, as in Diagram 11, 3 columns replace all 11 steps. In all cases MIBC was used as a frother.

More importantly, however, column flotation radically improved operation and recovery of molybdenite. With the conventional

DIAGRAM 7

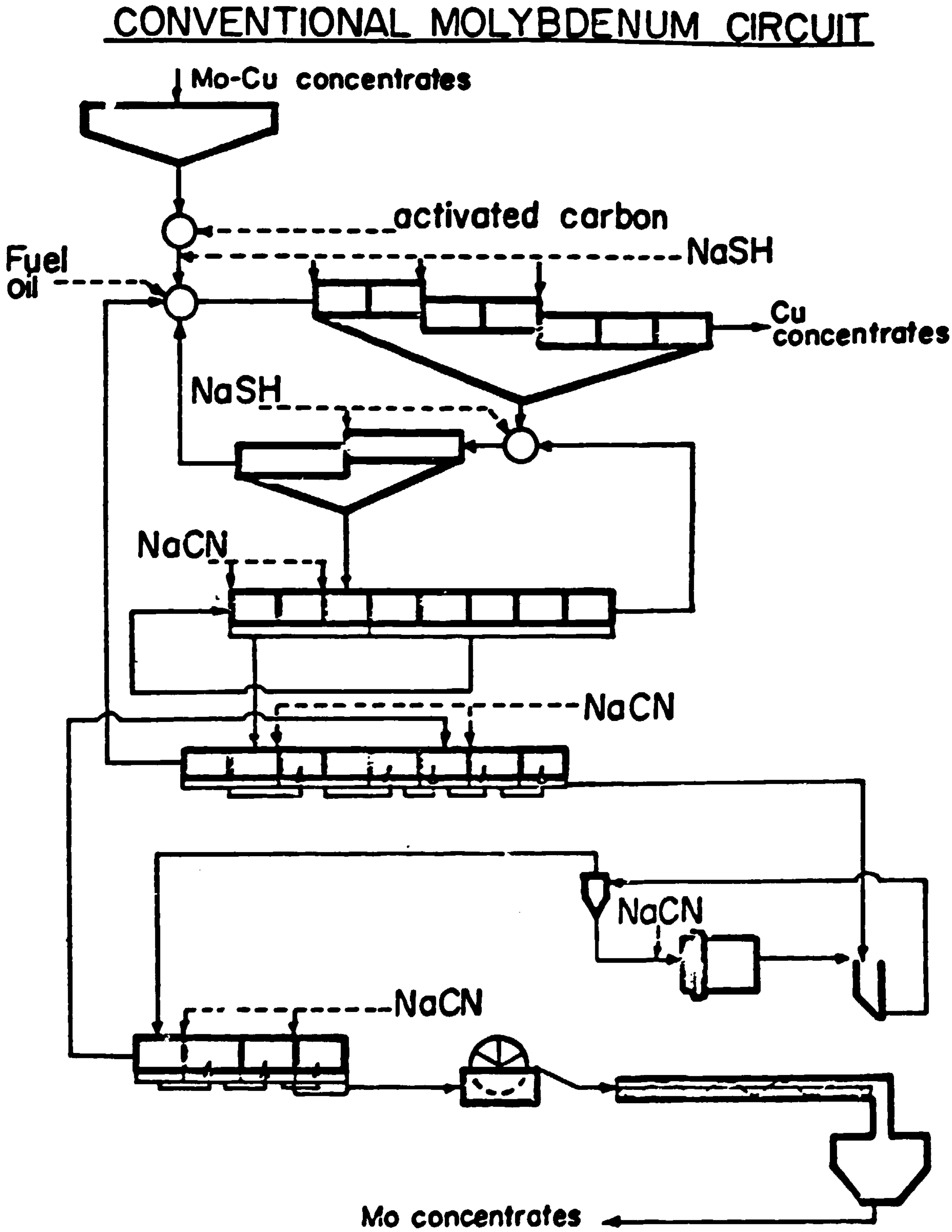

CONVENTIONAL MOLYBDENUM CIRCUIT
Mo-Cu concentrates
activated carbon
NaSH
Fuel oil
Cu concentrates
NaSH
NaCN
NaCN
NaCN
NaCN
Mo concentrates

DIAGRAM 8

MOLYBDENUM CIRCUIT WITH 2 COLUMNS

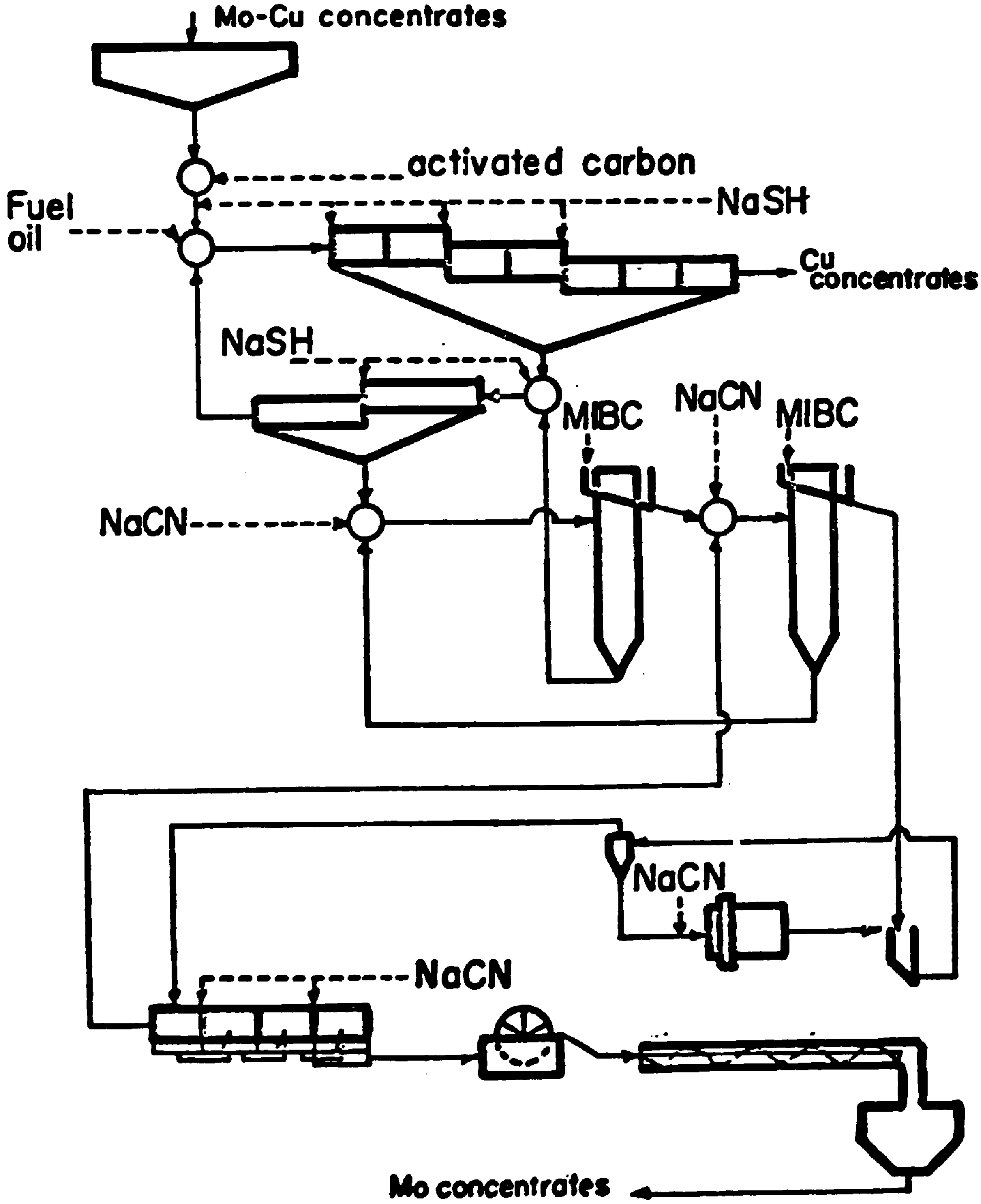
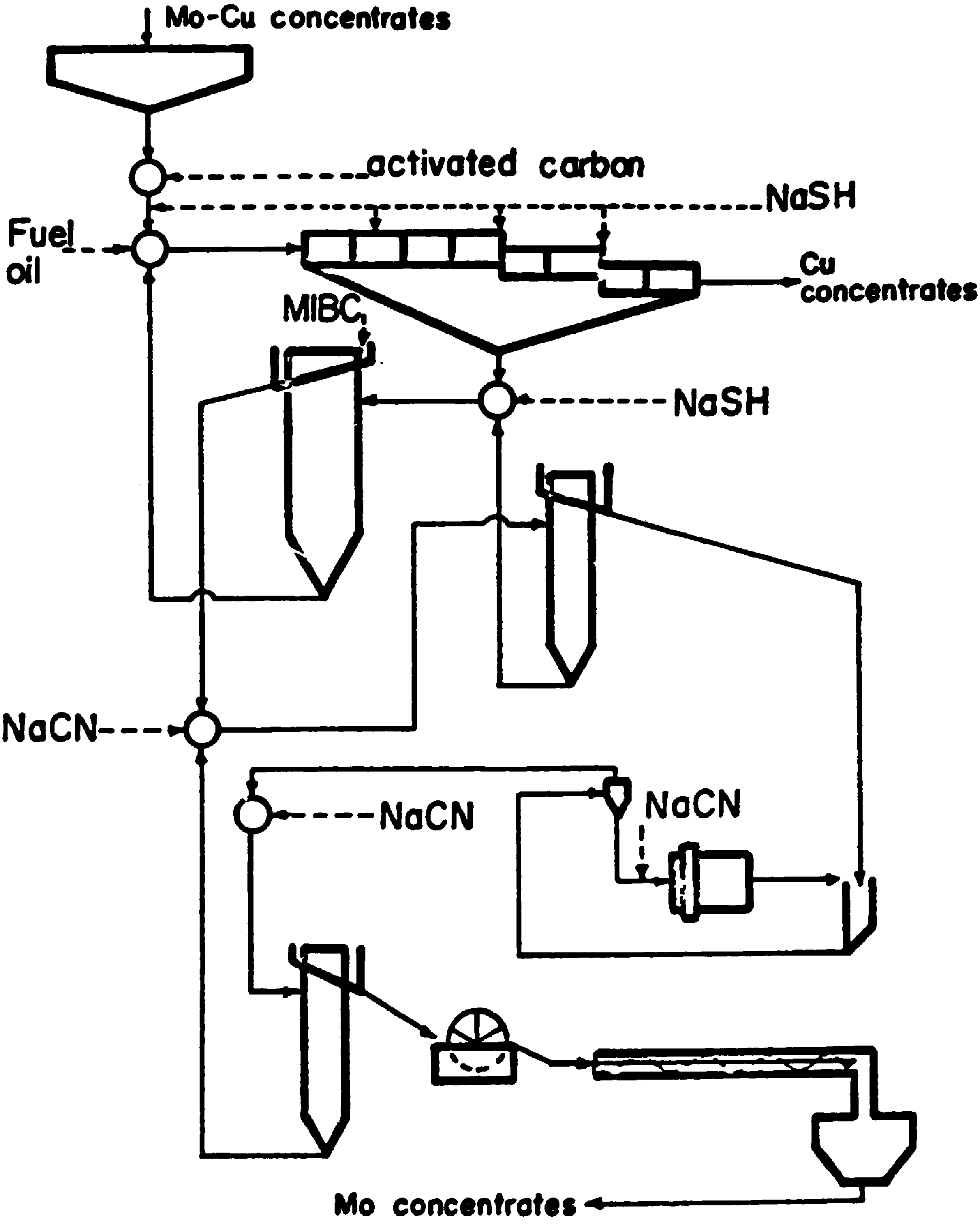


DIAGRAM 9

MOLYBDENUM CIRCUIT WITH 3 COLUMNS



circuit, retention times for flotation were very long and circulating loads very large. The circuits were easily upset and required long periods of time to reach stability. Each subsequent retreatment step increased molybdenum losses, and the overall molybdenum recovery was seldom higher than 55%.

With the two column circuit, of which the first was 0.9 x 12 m and the second 0.45 x 12 m, recoveries improved to 72% for the same grade of concentrate, while for the three column circuit recoveries surpassed 80% and molybdenite grades of the final concentrate improved from 83% to 87.3% MoS_2. In practical terms, this means that column flotation gave recoveries up to 32% better and molybdenite grades up to 4% higher. In a plant like Chuquicamata, where the annual molybdenum recovery is about 9,000 tons, this improvement can mean up to 2,900 tpy of Mo, which at present prices is valued at more than $ 17 million per year.

However, advantages of this technology are still very little perceived, except for Canada and Chile. They require additional research and testing. Also, more importantly, they require additional testing in new circuits. For example, a great possibility is also indicated for flotation of exclusively copper ores. Tests started in Peru were interrupted because of the bankruptcy of sponsors. An international effort in this respect will certainly be welcome, and it can be possibly extended to other base metals and their combinations.

Meanwhile, flotation progress is limited in the copper area to introduction of larger and larger flotation cells, which go up to 1,000 and 1,250 cu ft, as compared with 300 or 400 cu ft as a maximum only a few years ago. Such large equipment permits larger throughput at lower operational and maintenance cost and results in a significant economy of scale. In the recent expansion of Chuquicamata from 52,000 to 104,000 tpd, new technology is present in the form of SAG mills and giant flotation cells, all equipped with hydrocyclones as classifiers, which resulted in an important saving of space and reduction of production costs. The experience was so favorable that now a third expansion step to 156,000 tpd is being planned with all savings in investment costs.

Eventually, Latin America will turn into one of the most important copper suppliers in the world and particularly of European, Asian and even North American markets. This expansion will be carried out on account of its large mineral resources and very efficient standardized technology. Thus, cooperative effort to develop and improve such technology will give very rewarding results for this continent.

3.2 BACTERIAL LEACHING OF COPPER SULPHIDES

The necessity to process low-grade copper ores, which cannot pay with their copper content for the rather expensive standard flotation treatment, has led in these last years to the increased use of the so called "bacterial leaching" extraction method. The simplicity of this process, with its inexpensive investment and production costs, makes this process very attractive particularly because it produces solutions convenient for further processing by the SX/EW Process, discussed in the next chapter.

Bacterial leaching is principally applied to sulphide ores, which are less soluble than oxides under normal conditions with classical solvents. It was discovered that several types of autotrophic bacteria, i.e., those bacteria which live in absence of organic matter, can accelerate the leaching reactions in mine wastes, milling tailings and all sorts of other dumpings. Of course, the oxidizing conditions necessary for leaching of sulphide copper minerals are principally provided by atmospheric air, but bacteria such as Thiobacillus ferrooxidans, can speed up oxidation reaction by the following chain of reactions:

When attacked by oxygen and sulphuric acid, ferrous sulphide minerals, such as chalcopyrite, provide ferrous ion in aqueous solution according to the following reaction:

$$CuFeS_2 + 4\ O_2 = CuSO_4 + FeSO_4$$

Thiobacillus ferrooxidans chemically attacks the ferrous ions in solution to form ferric ions, according to the following reaction:

$$2\ FeSO_4 + H_2SO_4 + \tfrac{1}{2}\ O_2 = Fe_2(SO_4)_3 + H_2O$$

Then ferric ions act as leachants on sulphide minerals:

$$Fe_2(SO_4)_3 + Cu_2S + 2\ O_2 = 2FeSO_4 + 2\ CuSO_4$$

or $2\ Fe_2(SO_4)_3 + CuFeS_2 + 3O_2 + 2H_2O = 5FeSO_4 + CuSO_4 + 2H_2SO_4$

The first two reactions then become cyclic. These reactions can procede without the presence of bacteria, but enzymes of _Thiobacillus ferrooxidans_ catalyze the second reaction and accelerate the whole leaching process.

Other bacteria such as _Thiobacillus thiooxidans_ are found living in sulphide environments and are believed to attack sulphide minerals directly. They greatly contribute to sulphide oxidation by producing intermediate sulphur oxidation steps, such as for example:

$$S_2O_2^{2-} + \tfrac{1}{2}\ O_2 = S_2O_3^{\ 2-}$$

$$S_2O_3^{2-} + \frac{5}{2}\ O_2 = 2SO_4^{\ 2-}$$

Most sulphide mine waters contain the active autotrophic bacteria, and use of these waters together with dilute sulphuric acid automatically provides excellent conditions for leaching. The optimum conditions for flourishing of bacteria and operation are: pH between 1.5 and 3.5, temperatures between 25 and 40° C, avoidance of exposure of solutions to sunlight, and adequate oxygen supply obtained by aerating the solutions and by periodic draining of ore piles.

Bacterial leaching is used today for recovery of only two metals - copper and uranium. It is being increasingly introduced in low-grade copper mines, ore heaps, dumps or in already exhausted deposits. The process is attractive because it requires practically no labor and very low capital investment, but it is also much slower than other commercial leaching processes. It is particularly attractive for countries with warm climates.

In dump leaching, the process consists of pumping acid solutions at pH 2 containing ferrous iron in concentrations of 1 gr/l and ferric iron, also at 1 gr/l, to the top of the dump containing low-grade copper sulphide minerals and collecting the leach liquor at the base of the dump. Typically such liquors will contain about 1 gr/l of Cu in the form of copper sulphate. These solutions can then be processed by cementation or the SX/EW process. At preseht, the Chuquicamata mine in Chile is planning a vast operation of this type for the late 1980's.

Bacterial leaching can be applied not only to the ores but to concentrates as well. By leaching concentrates, the expensive smelting process can be avoided since the SX/EW process which follows will produce copper cathodes without smoke and ambient contamination. In Canada, for example, there exist specialized companies in bioleaching, which apply this technology to different metals and different products. In some cases it has been established, for example, that bioleaching is cheaper than roasting.

Bacterial leaching has also found acceptance in the Soviet Union and China. In the Soviet Union, copper is being bacterialy leached in one of the Tadjikstan copper mines, while in the Republic of China a subsidiary of Metallgesellschaft is trying to recover gold from copper bearing ores at the Tong Lu Shan mine in the Wuhan region.

This technology is very cheap but requires a lot of time. In abandoned mines or in deposits which are of very low grade it can be applied with great success when an initial circulating load has been built up to permit periodic stripping of liquors for copper and return of leachants back to the circuit.

At Chuquicamata, plans are now in development to install such a circuit to leach old mine dumps which contain about 0.3 to 0.4% Cu in the form of sulphides. A SX/EW plant will follow and recover probably as much as 500 tpd of copper. In other Latin American countries, where present copper ore grades are too low for commercial production by traditional means, such technology can be also successfully applied in situ, after a convenient fragmentization of bedrock.

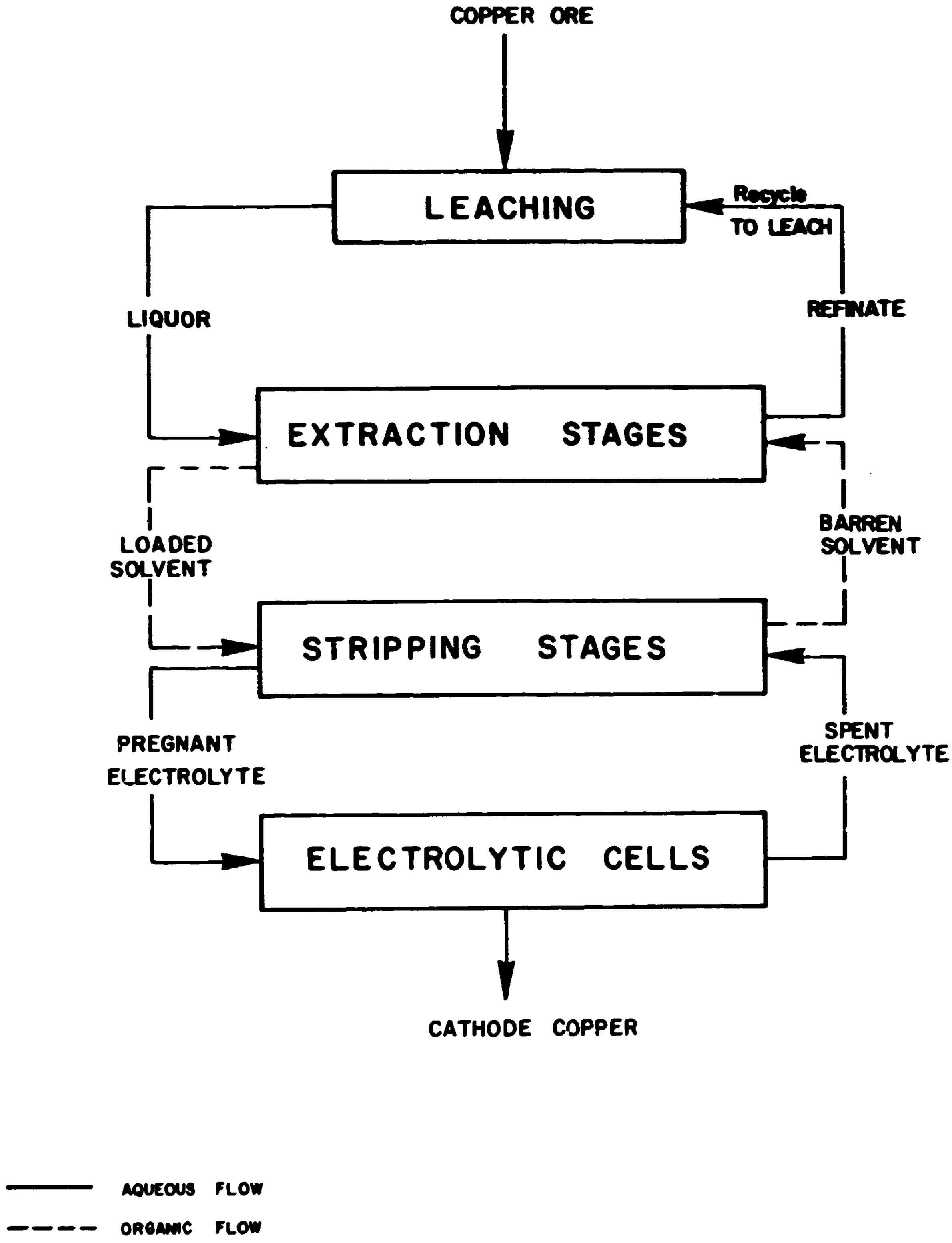
3.3 SX/EW PROCESS

The Solvent Extraction - Electrowinning Process is considered today the most direct and effective process for recovery of cathode copper and it is increasingly used in all copper operations where effective leaching of copper ores can be materialized. Its popularity is due to its relatively low investment cost, fast execution of plant construction and more importantly because of its low operating costs, which require minimum supervision and energy consumption. By using the leaching, solvent extraction and electrowinning sequence of operations, excessive comminution of ore, which is an energy and materials intensive operation, is avoided. Also, the flotation step, expensive because of its sophisticated equipment and reagents consumption, is eliminated. Finally, and most importantly, the expensive, energy intensive and ecologically inconvenient smelting step is also avoided. In sum, this means that equipment such as grinding mills, classifiers, flotation machines, smelter furnaces and all their controls are replaced by relatively simple mixers and settlers, while grinding media and flotation reagents consumption are replaced by organic solvent extraction media. Also, the final product, which is copper cathode, is obtained in its purest and most desirable commercial form at 99.9% purity, the same as is obtained in classical electrolytic refineries.

More importantly, the SX/EW Process is very flexible in its applications and can be run practically at any scale, which makes it very convenient for application in developing countries.

In Diagram 10, we give a schematic flowsheet for a SX/EW copper plant. A leach pregnant copper solution, which comes from an in situ, heap or vat leaching installation, containing normally between 1 and 3 grs/l Cu and between 1 and 3 grs/l of sulphuric acid, is introduced into solvent extraction units,

DIAGRAM 10: SCHEMATIC FLOWSHEET FOR EXTRACTING COPPER BY SX/EW PROCESS.



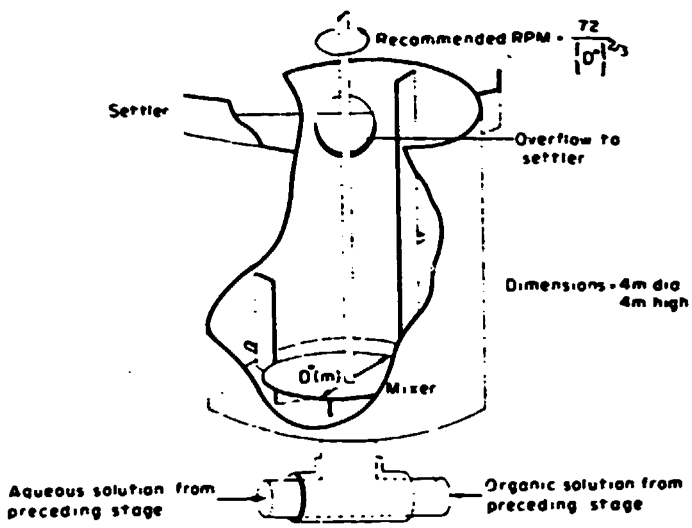
which consist of a mixer and a settler. In the mixers, the pregnant solution is mixed with an organic chemical of the chelating type, which in the case of copper solvent extraction is known under the names of LIX reagents (produced by General Mills Chemicals) and KELEX (produced by Ashland Chemical Company).

These organic reagents are characterized by selective solubility of copper ions and almost complete repelling of all other impurities. Thus intensive agitation in mixers produces the quantitative transfer of copper ions from dilute leach solutions to an organic phase, while all impurities remain in the barren refinate, which is recycled back to the leaching operations.

The pregnant organic phase is separated from the refinate in specially designed settlers, details of which are shown in Fig. 5 and 6. To make this operation quantitatively effective, generally three stages of extraction are carried out countercurrently: while pregnant leach solutions with the highest copper content are fed into the first mixer, the cleanest barren organic from strip operations enters through the third mixer to enhance final extraction of remaining copper. It is in this way that while from first to third mixer the copper concentration in leach solution diminishes, the organic phase becomes progressively more and more laden with copper, starting from the third to the first mixer. This, then, produces a conveniently highly loaded organic discharge from the first settler and maximally stripped refinate from the third settler, as indicated in Diagram 11.

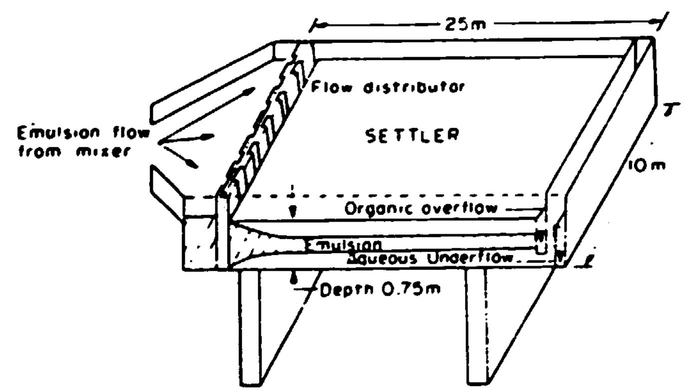
The loaded solvent normally contains about 2 grs/l of copper, while the refinate recycled to leach contains between 0.1 and 0.2 grs/l of copper and 3 to 5 grs/l of sulphuric acid. In this way, copper recovery at this stage is about 80 to 90

Figure 5



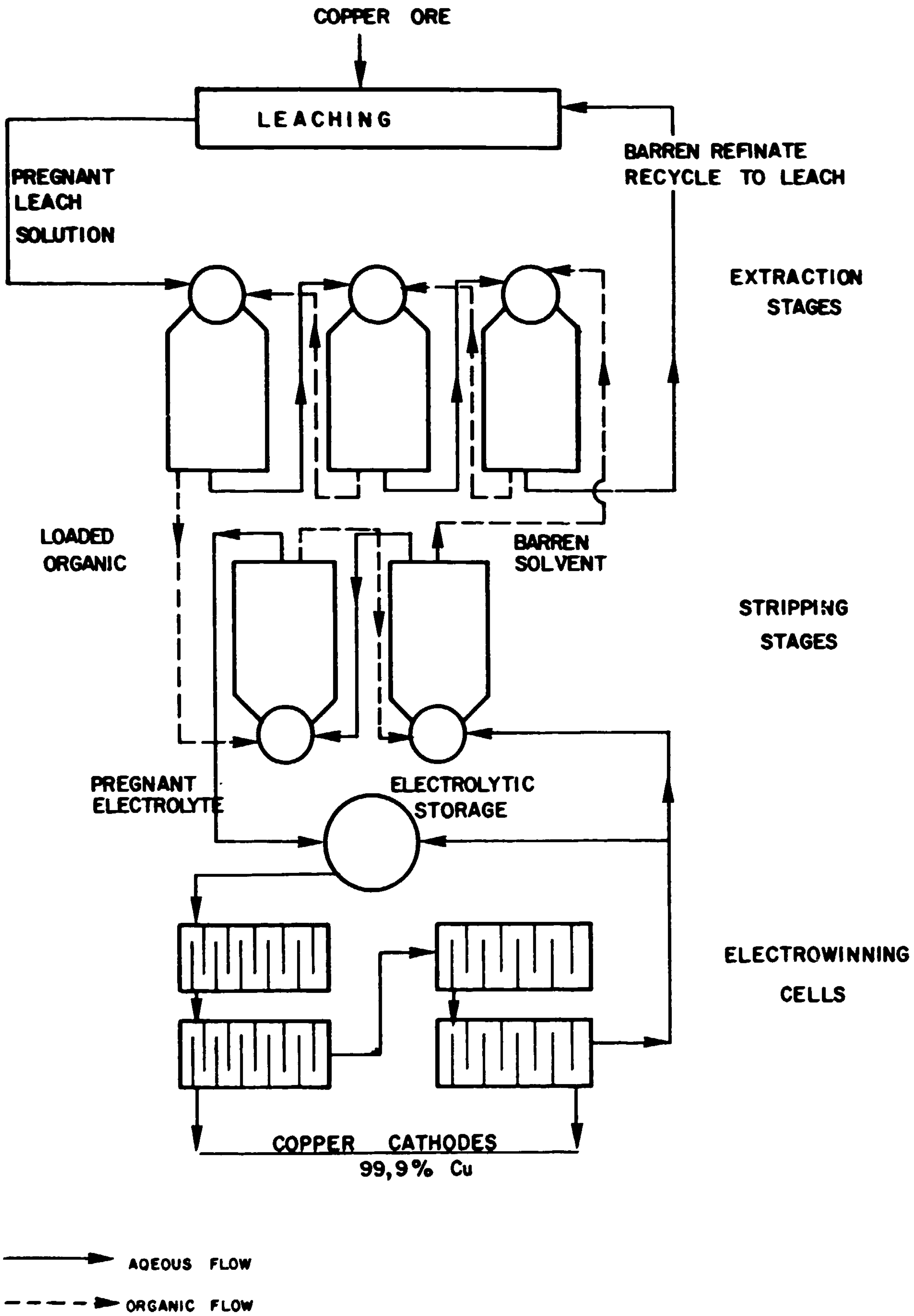
(a) Mixer

Figure 6



(b) Settler

DIAGRAM 11 : SX/EW PROCESS PLANT



percent. This greatly depends on the type of organic chemicals used, their selectivity and absorption capacity. The general absorption mechanism of these organic chelates is based on replacement of hydrogen atoms of two solvent molecules with an absorbed copper atom. For efficient treatment, organic solvents are always dissolved in an organic carrier, normally kerosene, in concentrations between 5% and 20% by volume. This is done to obtain a low viscosity liquid. A modifier is usually added to improve reaction rates or phase separations.

The most known brands used in solvent extraction of copper are LIX 63, which is an alfa-hydroxy oxime, and its modifications LIX 64N and LIX 65 N. Then there are KELEX 100 and KELEX 120, which are substituted hydroxy quinolines. These reagents are used in quantities which are 70 to 100 times higher than the quantity of copper absorbed. KELEX 120 has the highest absorption rate of 70.

In this way, in order to treat a 2 grs/l copper solution, 14 to 20% by volume solutions of organics should be used. Carriers of these solutions should have a relatively high flash point for safety's sake.

After the extracting stage, loaded organic solvents enter into stripping stages. As shown in Diagram 11, these are carried out in the same type of equipment used in extraction, and also the process is carried out counter-currently. To strip copper content from organics, a strong solution of sulphuric acid is used (150 grs/l). Stripping may require 2 or 3 stages and retention time in mixers is between 90 and 120 seconds.

As a result of these operations, a pregnant electrolyte of 40 to 50 grs/l of copper and 135 to 150 grs/l of sulphuric acid is obtained and is then fed to electrolytic cells for electrowinning. Spent electrolyte from the refining stage contains normally between 25 and 35 grs/l of copper and 150 to 185 grs/l of sulphuric acid and is recirculated to the stripping stage.

As already indicated, the direct production cost of this process, which includes copper leaching, extraction and stripping stages and electrowinning, is around 30 cents per pound of cathode copper. In new operations where amortization and debt servicing charges must be added, these costs may go up to 45 c/lb. Still, this is considerably below normal processing costs for sulphide copper minerals, which worldwide average about 64 c/lb.

In Table 2, the list of known operations of this type are given, including their production capacity and costs.

The impact of SE/EW technologies on overall corporate copper production costs was really impressive. Here is how it is reflected at some major plants in terms of cents per pound of copper produced.

Mine	Location		1980	1981	1982	1983	1984	1985
Ray	Arizona		107	102	104	78	73	75
Tyrone	New Mexico		81	88	99	92	85	70
Twin Buttes	Arizona	ox	65	65	66	64	49	--
Bagdad	Arizona		65	78	88	97	--	75
Pinto Valley	Arizona	ox	--	47	38	37	35	36
		sul.	104	89	105	--	79	75
Gibraltar	BC		63	73	77	85	66	62
Chuquicamata	Chile	ox	81	95	54	52	48	46
Lo Aguirre	Chile	ox	--	99	82	74	72	71
Mantos Blancos	Chile		92	85	63	61	63	60
Las Cascadas	Chile	ox	113	89	80	74	72	61

For most of the mines the overall production cost is reported, i.e., comprising sulphides and oxides. When only oxides are considered, then the ox abbreviation so indicates.

TABLE 2

SUMMARY OF COPPER OPERATIONS USING SX/EW PROCESS

PLANT	COUNTRY	ANNUAL CAPACITY tons per year		PRODUCTION COST c/lb
BAGDAD	USA, Arizona	6,500		37 - 38
CHINO	USA, New Mexico	19,500		19 - 23
CYPRUS-JOHNSON	Arizona	4,300		49 - 50
MIAMI	USA Arizona	4,000		59 - 61
MORENCI	USA Arizona	40,000		in construction
PINTO VALLEY	USA Arizona	7,000		35 - 37
RAY	USA Arizona	28,000		53 - 60
SAN MANUEL	USA Arizona	22,500		in construction
TWIN BUTTES	USA Arizona	30,000	closed	50 - 52
TYRON	USA New Mexico	20,000		36 - 40
BATTLE MOUNTAIN	Nevada	6.500		45 - 50
BINGHAM	USA Utah	20,000		52 - 62
GIBRALTAR	BC, Canada	5,100		30 - 35
CHUQUICAMATA	Chile	50,000		46 - 47
	Expansion to	250,000		25 - 30
EL TENIENTE	Chile	5,000		25 - 30
LO AGUIRRE	Chile	14,000		34 - 35
LAS CASCADAS	Chile	20,000		59 - 61
CERRO VERDE	Peru	24,000		35 - 40
CANANEA	Mexico	14,000		40 - 45
NKANA	Zambia	6,800		35 - 40

3.4 SEGREGATION PROCESS

Mixed sulphide-oxide ores are always a kind of a metallurgical problem because oxides do not float well in sulphide circuits, and additional steps such as leaching and precipitation are needed to promote recovery of oxides along with or separately from sulphides. This gave origin to procedures such as leach and float processes and the LPF (leach - precipitation - float) process.

On the other hand, as already discussed, smelting of sulphides or other copper concentrates is an energy-intensive and expensive operation. It requires heating of concentrates above 1,150° C and besides generates ecologically damaging SO_2 gas.

In looking for a solution to these problems, the original segregation process was discovered and applied in the 1930's in the Belgian Congo, now Zaire. The process is relatively simple: a mixed sulphide-oxide ore, refractory to other metallurgical treatment, is mixed with small quantities of coke and common salt and is then heated in a rotary furnace at about 700°C. At this temperature copper sulphide minerals decompose to oxides while sodium chloride generates active chlorine, which volatilizes copper oxides in the form of copper chlorides. However, due to the presence of small coke particles, these chlorides cannot escape the furnace and precipitate (by action of CO around coke particles) in the form of metallic copper on coke particles. In this way, segregation of copper from the original ores takes place on coke particles. The remaining unaffected sulphides and the segregated oxides in the form of metallic copper are then repulped and floated together into a bulk copper concentrate.

Typically, a sulphide copper ore unaffected by oxidation gives recoveries of around 90 percent. If affected by oxidation, these recoveries can drop to only 70 to 80%. The segregated ore, after processing, gives recoveries between 88 and 90 percent. The economics of the process depend on the cost of recovery of this additional copper - which is essentially the cost of heating the ore to 700°C, its chemical attack and the use of oxidation resistant equipment to carry out this process.

The economics of this process worked relatively well before the energy crisis, and an industrial operation known as the TORCO PROCESS was developed in Zambia, according to the flowsheet shown in attached Diagram 12. The ore is heated in a fluid bed reactor and the hot granular overflow is falling into a segregation reactor, into which coal and salt are also fed. The segregated material is then discharged into a quench launder, which feeds a flotation circuit for metal recovery. So far, two such plants have been built, one at Akjoujt in Mauritanua and the other at Rhokana in Zambia. Their operation has suffered from some difficulties and plants have worked with great interruption periods.

However, what now attracts attention to this process is the possibility of treating copper concentrates by the segregation process to produce metallization of copper at much lower temperatures - typically at 700°C instead of 1,200°C. This could result in considerable savings of fuels. The general flowsheet for such a plant, which would be particularly suitable for Central Africa deposits, would be a dry aerofall mill grinding to desired particle size, then application of the segretation process and flotation of copper concentrate high in metallic copper.

This technology has not yet been sufficiently studied but probably is worth another try. A general flowsheet layout for the TORCO process is given in attached Diagram 13.

The advantage of this process is that it can be carried out practically on any scale from 500 to 10,000 tpd and thus fits practically any size of orebody in excess of 2-3 million tons of ore. Here is for example the cost evaluation for a 4,000 tpd operation expressed in terms of US dollars per metric ton of ore:

Labor and supervision	\$ 1.20
Power	\$ 1.40
Fuel for drying and segregation	\$ 3.85
Reagents for segregation and flotation	\$ 1.10
Maintenance and Misc.	\$ 1.00
Total Direct Cost	\$ 8.55
Indirect Cost includ. amortization	\$ 3.00
Total cost per ton of ore	\$ 11.55

For a 500 tpd operation, costs may be closer to \$ 16 - \$ 18 per ton, while for a 10,000 tpd plant they will probably drop to around \$ 9 to \$ 10 per ton. This means that if we have, say, a copper ore assaying 2% Cu which can be recovered by this process with 85% efficiency, i.e., yielding 37.5 lbs of copper per metric ton of ore, then at a 4,000 tpd operation such copper can be obtained at 31 c/lb, which when are added the smelting, refining and transportation charges of say 20 c/lb, this will still produce 51 c/lb copper.

In case of a 500 tpd operation, such costs will be around 45 c/lb, which together with smelting and refining charges will increase the total cost to 65 c/lb, or approximately the present price of copper. In other words, a higher grade deposit would be required for a profitable operation.

In case of a 10,000 tpd operation, costs are likely to be reduced to 24 - 26 c/lb because of the economy of scale, and the final product can be delivered at a 44 to 46 c/lb total cost.

Economically, this alternative looks attractive, provided that it can be carried out without major handicaps. The problem with the segregation process is that it is a rather sophisticated project requiring fine tuning of technology, which is sometimes difficult to attain in developing countries where relatively little specialized work forces exist. Besides, the process has its problems such as intensive corrosion problems, which if not properly handled may bring plants to frequent shutdowns and production losses. Thus, further testing and research is necessary to make this process work satisfactorily.

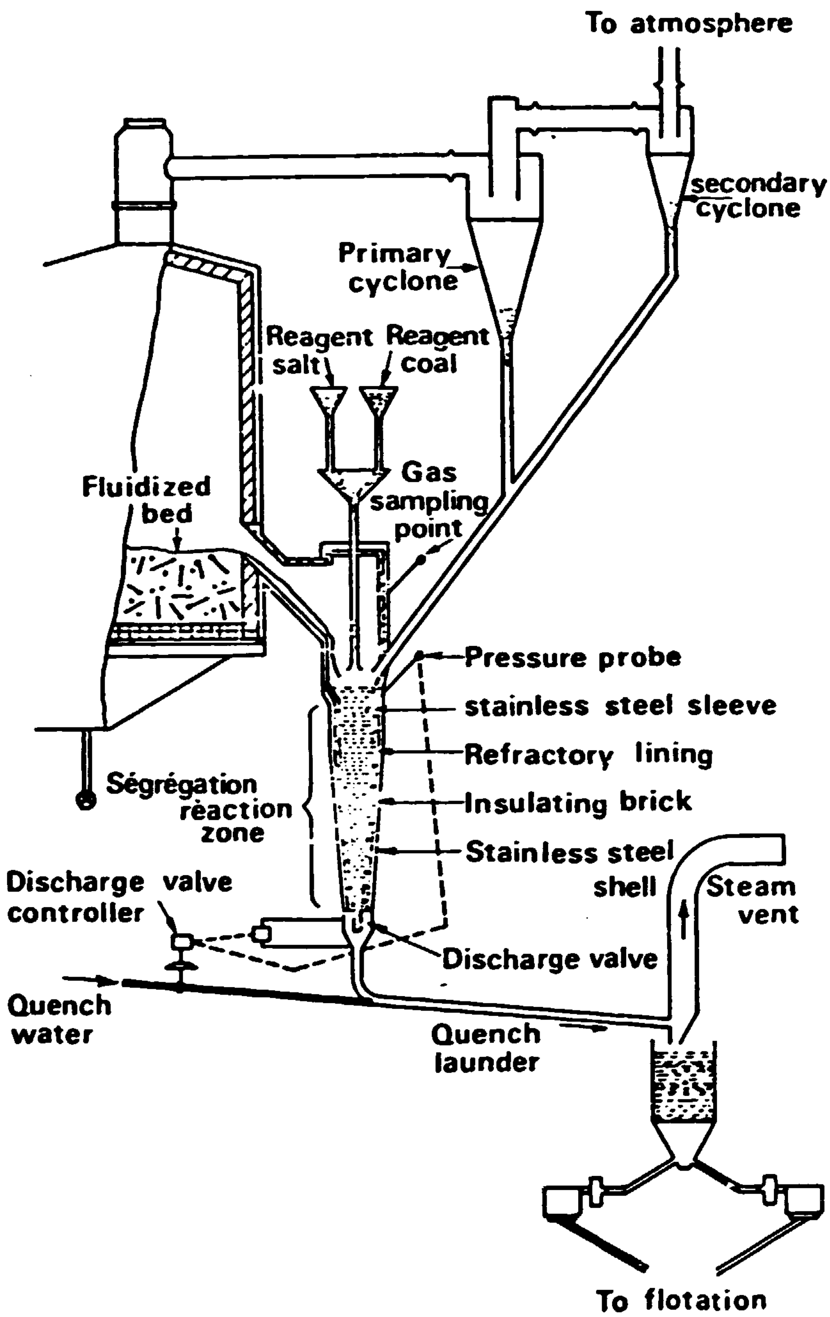
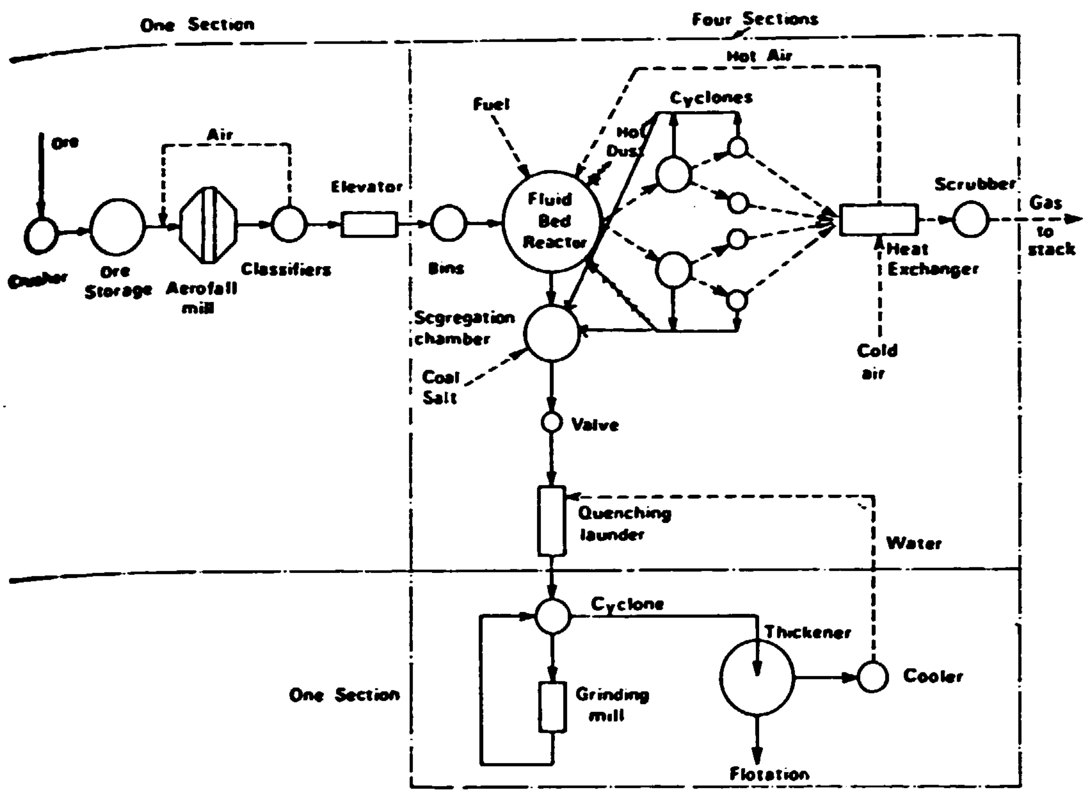


Diagram 12 : TORCO PROCESS UZED IN ZAMBIA



Diagr.13: Flow sheet of TORCO plant at Akjoujt (Mauritania)

3.5 OXYGEN TECHNOLOGY

In terms of scientific research and technological advance, however, the most important results in copper production technology have been achieved in pyrometallurgy. In attached Diagram 14, we can follow the advances of copper smelting technology in the 20th century along with the increase in copper consumption and production.

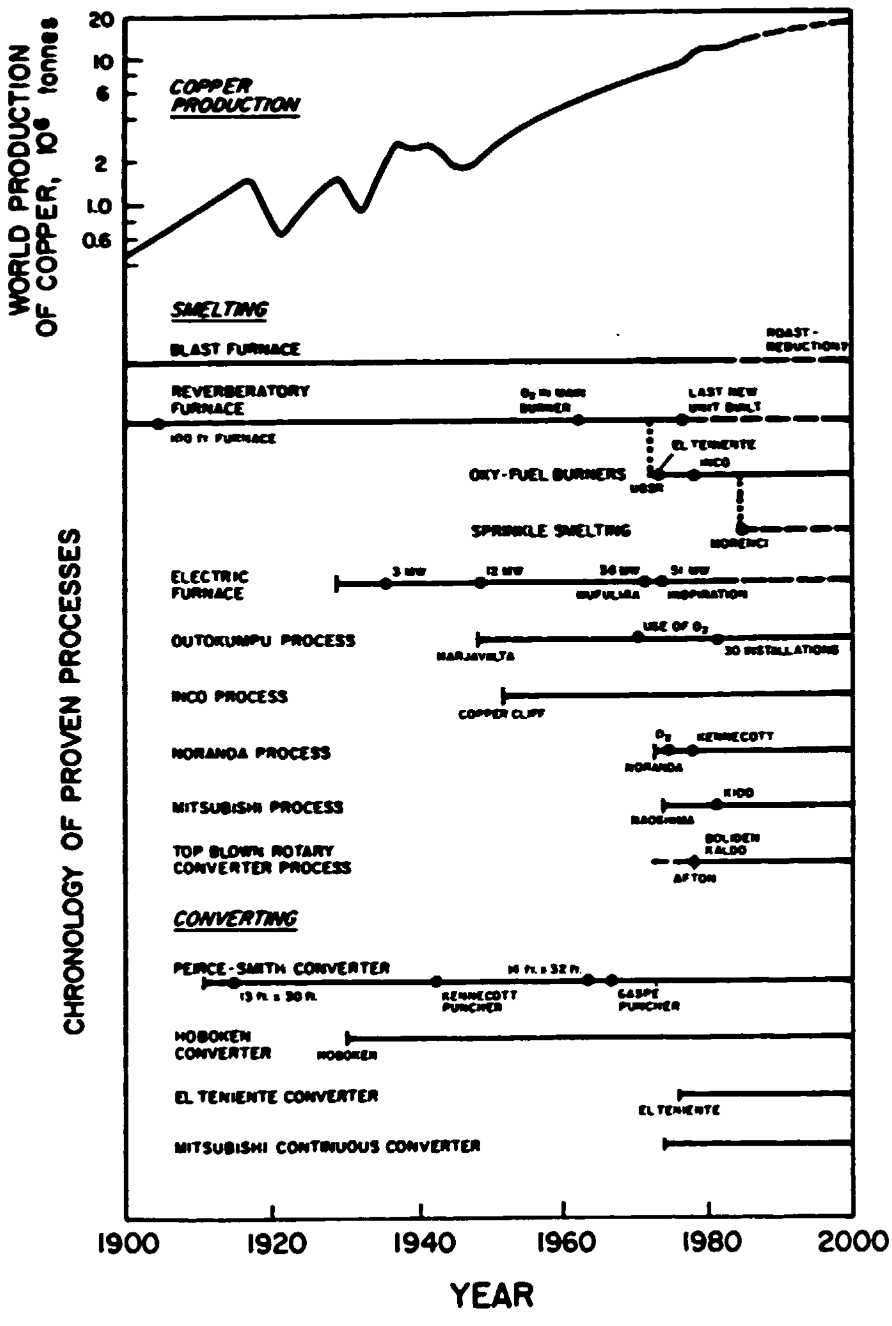
The old blast furnace which served so much for recovery of copper from oxide and then sulphide ores has now given way to roast reduction. Reverberatory furnaces,which permit smelting of sulphide concentrates, received an important innovation in the early 1960's when oxygen enriched air started to be blown through burners. This gave origin to oxy-fuel burners, as developed at El Teniente, INCO, in the Soviet Union and now at Morenci where sprinkle smelting is being industrially tested. The last modern reverberatory unit was built in the mid 1970's, and if it were built now it would be of a modified type or would be replaced simply by other developing processes.

Electric smelting furnaces still go strong with the newly built smelter at Mufulira and Inspiration. Their capacity has grown from 3 MW to over 50 MW, but they obviously require cheap hydroelectrical power or thermal electric power at low cost.

The most successful new smelting process is doubtlessly Outokumpu Flash Smelting, as introduced after WWII and which has today to its account some 30 installations. Success of this process was also enhanced by introduction of oxygen technology in the 1970's, which made Japanese smelters so efficient.

The other successful process is the INCO Flash Smelting furnace, installed however only at two places: in Sudbery, Canada, and Almalyk, Uzbekstan USSR). Its potential for applica-

Diagram 14


WORLD PRODUCTION
OF COPPER, 10⁶ tonnes
20
10
6
2
1.0
0.6
COPPER
PRODUCTION
CHRONOLOGY OF PROVEN PROCESSES
SMELTING
BLAST FURNACE
ROAST-
REDUCTION
REVERBERATORY
FURNACE
LAST NEW
UNIT BUILT
EL TENIENTE
INCO
OXY-FUEL BURNERS
USSR
SPRINKLE SMELTING
ELECTRIC
FURNACE
3 MW
12 MW
36 MW
INSPIRATION
OUTOKUMPU PROCESS
USE OF O2
HARJAVALTA
30 INSTALLATIONS
INCO PROCESS
COPPER CLIFF
NORANDA PROCESS
O2
KENNECOTT
NORANDA
MITSUBISHI PROCESS
KIDD
TOP BLOWN ROTARY
CONVERTER PROCESS
BOLIDEN
KALDO
AFTON
CONVERTING
PEIRCE-SMITH CONVERTER
13 ft. x 30 ft.
KENNECOTT
PUNCHER
GASPE
PUNCHER
HOBOKEN
CONVERTER
HOBOKEN
EL TENIENTE CONVERTER
EL TENIENTE
MITSUBISHI CONTINUOUS CONVERTER
1900
1920
1940
1960
1980
2000
YEAR

tion is still substantial, although other alternatives have become available lately.

Next to develop was Noranda's one-step smelting process, whose success is critically limited to use of oxygen to obtain all possible advantages. So far it has been used only by Kennecott at the Utah Bingham Smelter.

From the second half of the 1970's we see also the appearance of a new Japanese smelting process developed by Mitsubishi. This process was tested and introduced at Naoshima and then at the Kidd Creek mine in Canada.

The final addition to smelting technology was the top blown rotary converter process, which processes cement or metallic copper concentrates and which is used at Boliden and Afton.

All these smelting processes are normally accompanied by converting in one form or another. The traditional Pierce-Smith convertor with its minor modifications (punchers) still holds the predominance it has maintained since the beginning of this century. However, in the 1930's it was modified to give origin to the Hoboken convertor. The El Teniente convertor, named after the mine at which this technology was developed, was developed in the mid-1970's, approximately at the same time when the Mitsubishi continuous convertor was developed.

In the next pages we will analyze the most important innovations offered by these new technologies and their likely evolution in upcoming years, with a special emphasis on potential uses by developing countries.

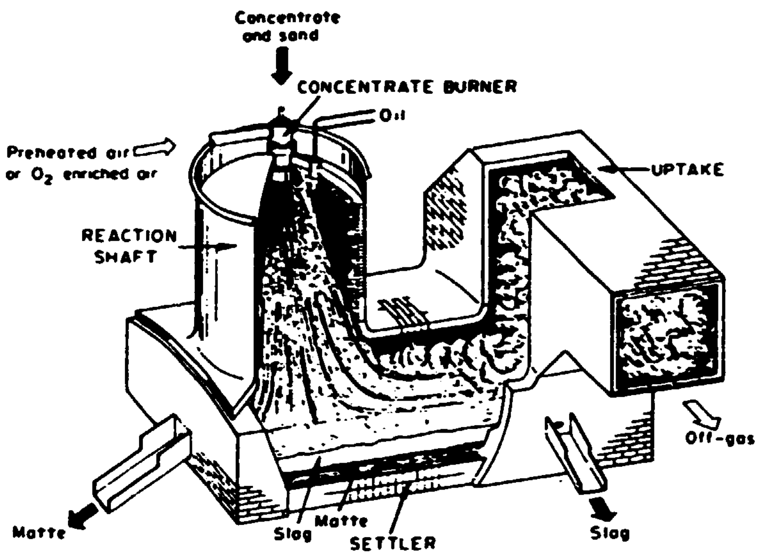
3.51 Outokumpu Process - The attached figures 7 and 8 give us cutaway views of Outokumpu and Inco flash smelting furnaces. Outokumpu is an older process and the first which introduced the flash principle. It originally operated on preheated air but then also introduced oxygen enriched air in order to eliminate fuels. Outokumpu produces mattes with about 70% Cu, 22% Fe and 8% S, which require further converting in order to obtain blister copper. Inco's oxygen flash smelting is lower on copper in mattes - about 48% Cu, 24% S and 26% Fe.

In this way, it can be considered that the Outokumpu Process combines roasting, melting and partial converting into a single process. When only preheated air (to 450°C) was used to supplement the heat generated by the exothermic oxidation of FeS, mattes assayed then only 45-50% Cu, and fuel oil needed to be added to finish the reaction. With the introduction of oxygen enriched air, the process became completely autogenous and the copper content of the matte increased to 65-70%. With a higher grade of matte, the required converting capacity and energy consumption fell sharply by as much as 40 to 50 percent. Also, addition of oxygen reduces the volume of gases and increases their SO_2 content from the normal 10-15% to as much as 30%.

The flexibility of the flash smelting process in terms of treating concentrates of varying composition and controlling matte grade is based on the fact that the degree of oxidation in suspension (flash) smelting can be regulated rapidly and easily by changing the ratio of concentrate to oxygen in the process air.

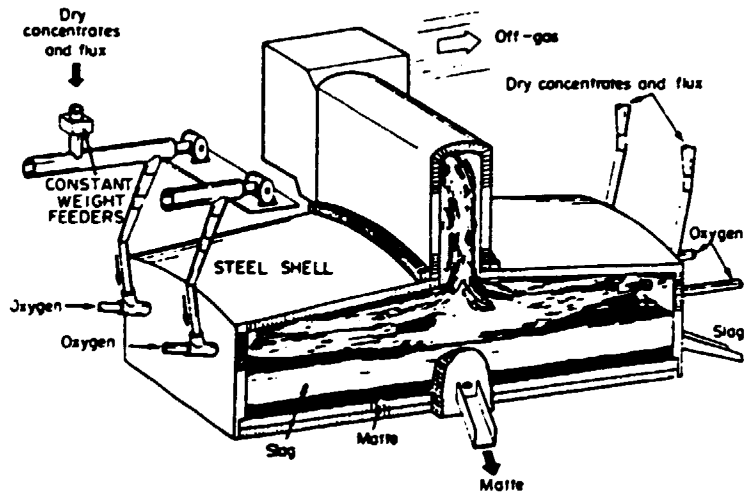
3.52 Inco Process also combines roasting, melting and partial converting in one autogenous operation. The use of technical oxygen (about 96% purity) instead of air to combust labile sulphur and iron sulphide in the feed eliminates the need for heating up nitrogen, which is the major cause of low fuel

Figure 7



Cutaway view of Outokumpu preheated blast flash-smelting furnace.

Figure 8



Cutaway view of INCO oxygen flash-smelting furnace efficiency in pyrometallurgical processes. Concentrates are injected through two burners in each wall of the furnace and combusted in a horizontal stream of oxygen. The grade of matte resulting from autogenous oxygen flash smelting depends on concentrate composition and feed rate and the proportion of secondary materials added to the sulphide feed. This process is also very flexible and can treat feeds of different composition. Slags of this operation are of around 0.8% Cu and can be discarded directly. Matte grades obtained go up to 55% Cu.

3.53 Noranda Process is a continuous process designed to produce either blister copper or copper matte directly from sulphide concentrates. Originally, the idea was to produce blister copper directly from concentrates, thus combining roasting, smelting and converting steps in one reactor. However, it was established that certain impurities, such as As, Bi and Sb, if not properly eliminated in the smelting step through slags, tend to penetrate up to anodes which makes them marketably unacceptable. Thus, the present form of this process, as applied at Horne and at Bingham, produces in a first stage a 70-75% Cu matte, which is then fed into traditional Peirce-Smith convertors.

The conceptual operation of the Noranda Process is given in Fig. 9, and the flowsheet of its pilot plant test unit is given in Diagram 15.

As shown in Fig. 9, in the Noranda Process pelletized concentrate feed and flux are counter-currently fed from the top of the furnace while the oxygen enriched air supplies the necessary heat to complement the reaction heat balance. Without oxygen enriched air, the Noranda process is not economically viable. The melting bath in which reaction occurs is maintained in a highly turbulent state, which results in a very efficient heat and mass transfer and high specific smelting rate of about 30 tons per m^2 per day. The melt is oxidized to high-grade matte of 70-75% Cu with oxygen enriched air (34% O_2)

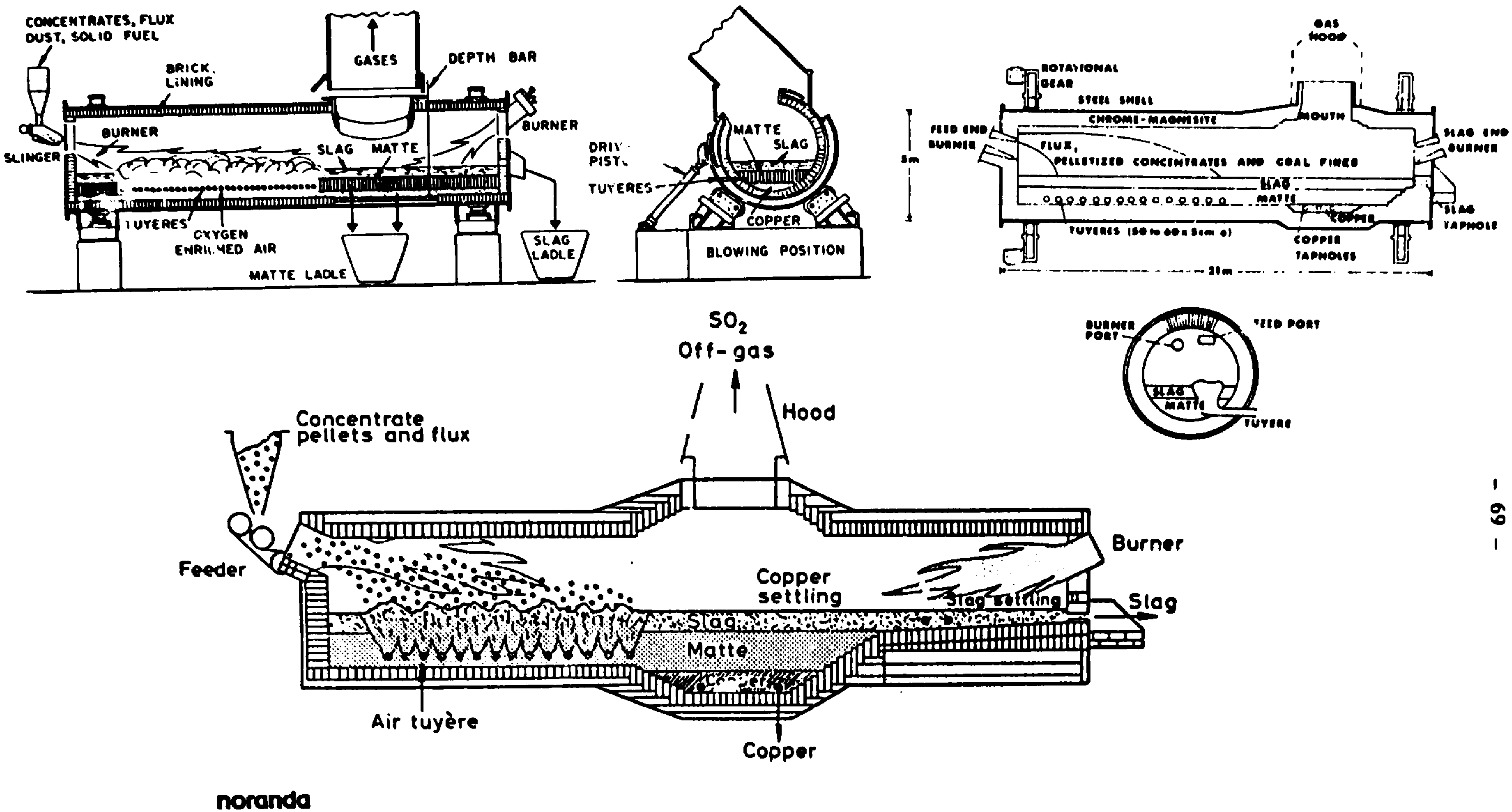
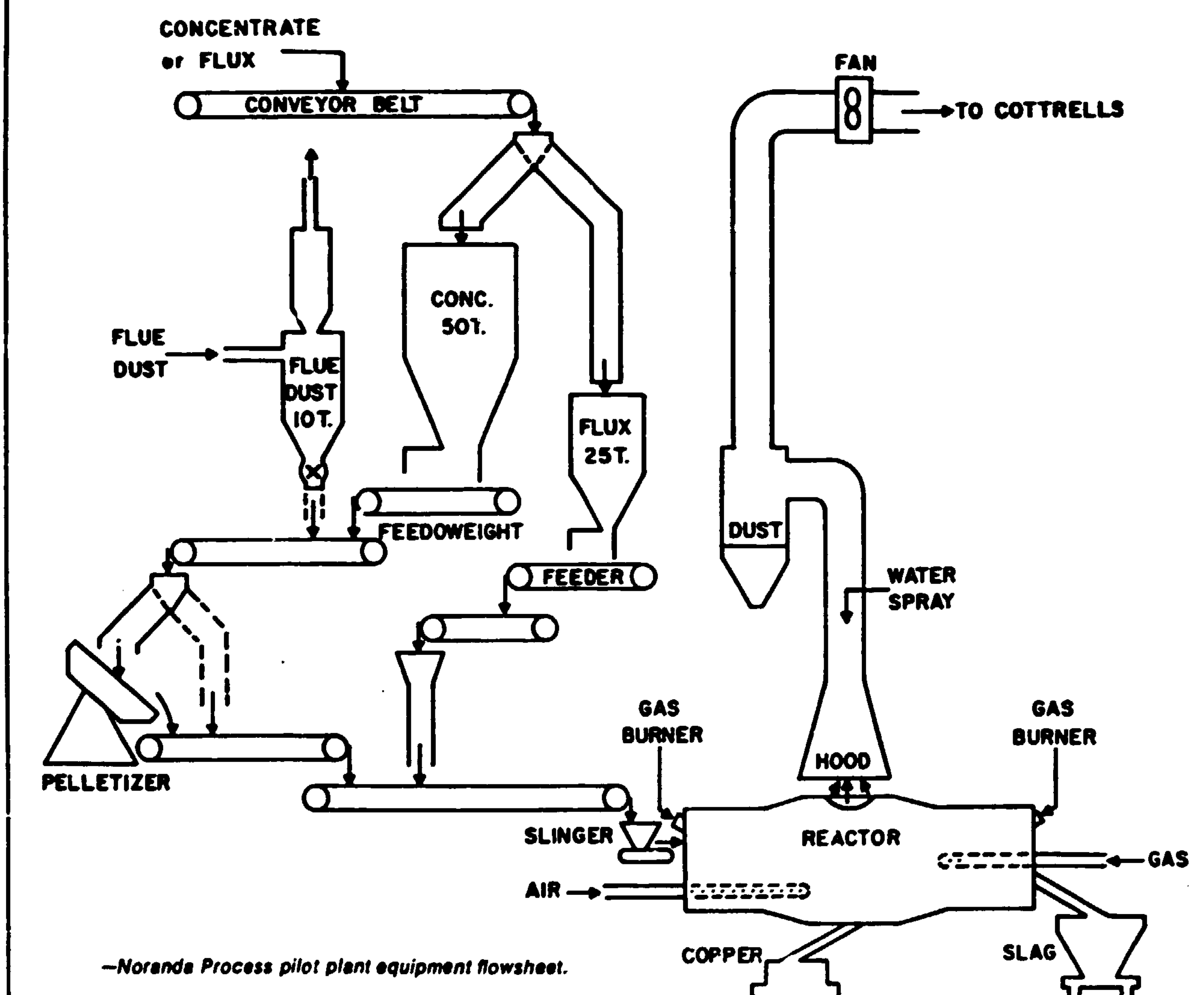


Fig.9– Schematic drawing of Noranda Process reactor.

DIAGRAM 15



—Noranda Process pilot plant equipment flowsheet.

through side blown tuyers. Overall oxygen efficiency is 100 percent.

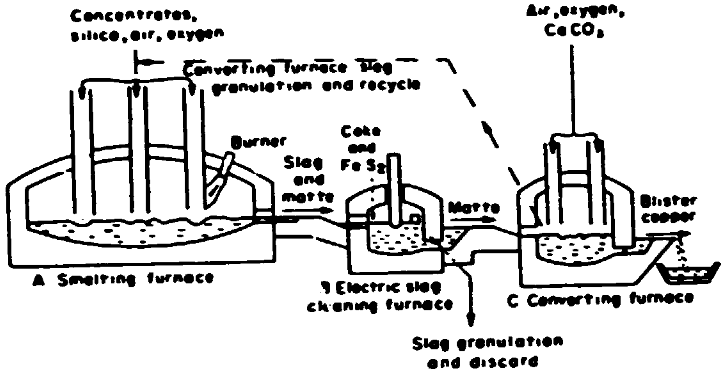
The process is relatively insensitive to changes in grade of feed material, including secondaries, and is very flexible for operation. The basic controls are reduced to oxygen/concentrate ratio and flux/concentrate ratio to maintain the Fe/SiO_2 ratio in the slag at 1.8/1.

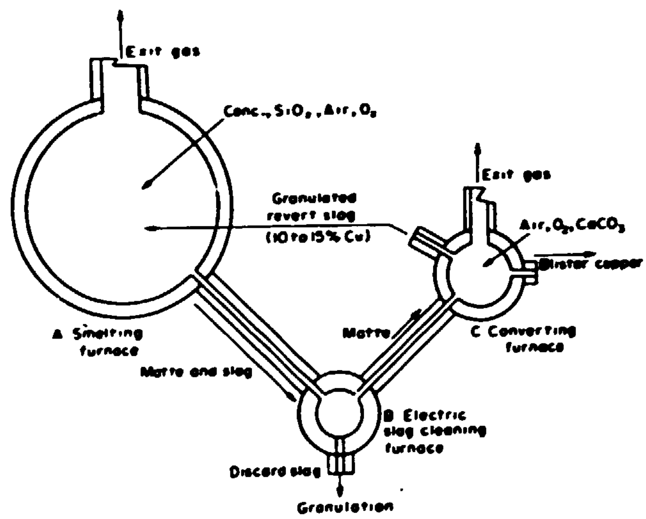
3.54 Mitsubishi Process - The Mitsubishi continuous copper smelting process was developed in Japan at the Naoshima smelter and apart from Naoshima is used today at the Kidd Creek mine. Its principle of operation is shown in Diagram 16.

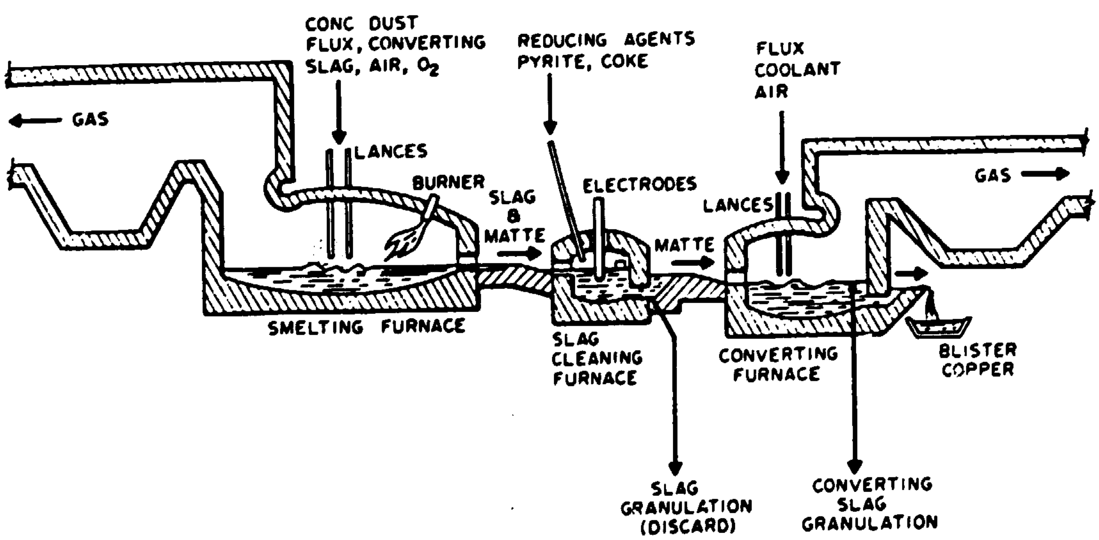
This is a multistep process which produces blister directly from concentrates, using for this three interconnected furnaces and thus dispenses with the conventional converter aisle and transport of materials by crane.

Dried fluxed concentrates and oxygen enriched air (30-35% O_2) enter the smelting furnace through non-submerged vertical lances and are smelted to produce matte of 65% Cu. The produced slag emulsion flows by gravity along an enclosed launder into an electric slag-cleaning furnace for settling and discard of slag. Fuel is required in the smelting furnace to supply the heat deficit. High-grade matte flows from the slag cleaning furnace via a syphon top hole and interconnecting launder to a converting furnace where it is continuously oxidized to produce blister using oxygen-enriched air of 26-28% O_2. There is no intermediate white metal layer in the furnace. Oxygen efficiency in the converting furnace is 85-90%, a figure comparable with Peirce-Smith converter operations. Because of the use of oxygen, off-gas strengths from the smelting and converting furnaces are typically 14-17% and 17-19% SO_2, respectively.

DIAGRAM 16







Mitsubishi continuous copper smelting.

The normal oxygen enrichment level in the smelting furnace is not sufficient for autogenous smelting. Increasing the level of enrichment allows fuel quantities to be reduced. Recent tests at Naoshima indicate that with an increase of oxygen content of the air from 28% to 39%, throughput of these furnaces can be increased by 50% while fuel requirements can be reduced 70%. Autogenous smelting can be reached with a 65% enriched air.

3.55 El Teniente Process - After unsuccessful trial at using conventional convertors to smelt concentrates along Noranda's process lines, El Teniente developed a new technology which consists of oxygen smelting of concentrates in reverbs by using excess heat generated by oxidation of matte. The slag blowing stage is carried out continuously by charging reverb matte of 48% Cu and concentrate of 40-45% Cu in an approximately 1 : 1 proportion. The heat generated by oxidizing the seed matte with air enriched to 32% O_2 is sufficient to smelt concentrates autogenously and produce a high grade matte of 74-78% Cu, which is tapped and blown to blister copper in conventional Peirce Smith-convertors.

El Teniente Process is schematically presented in Diagrams 17 and 18, while details on equipment used can be seen in figures 10 and 11.

Essentially, the innovations included in the El Teniente process are: (1) introduction of oxy-fuel burners, which allows replacement of standard long flame air burners with higher temperature short flame burners located on the roofs of reverberatory furnaces. This leads to a more efficient heat transfer and decreases the volume of off gases. (2) Simultaneous reverberatory matte converting and autogenous concentrate smelting in El Teniente Modified Convertors by blowing oxygen enriched air and producing high grade white metal.

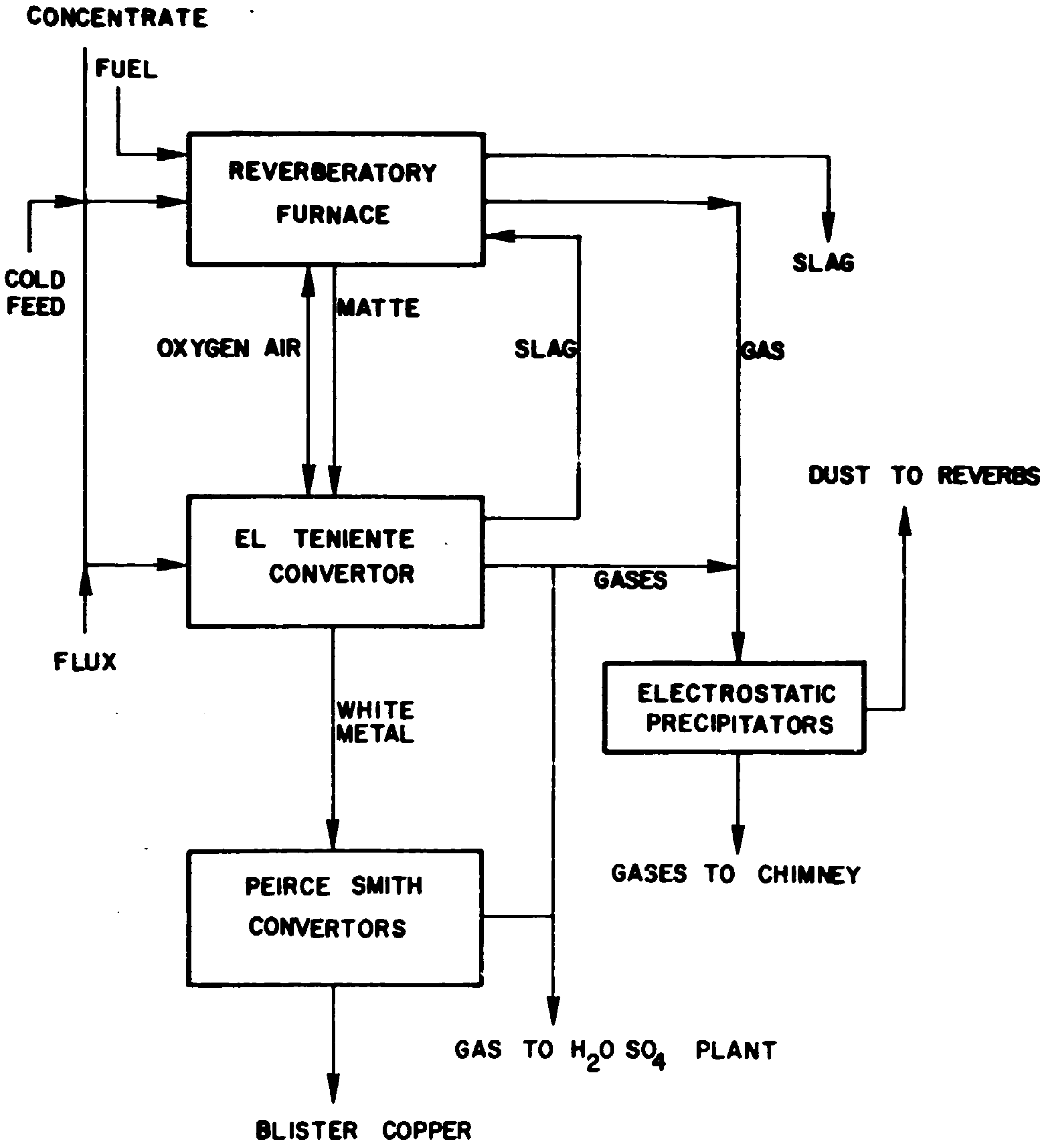
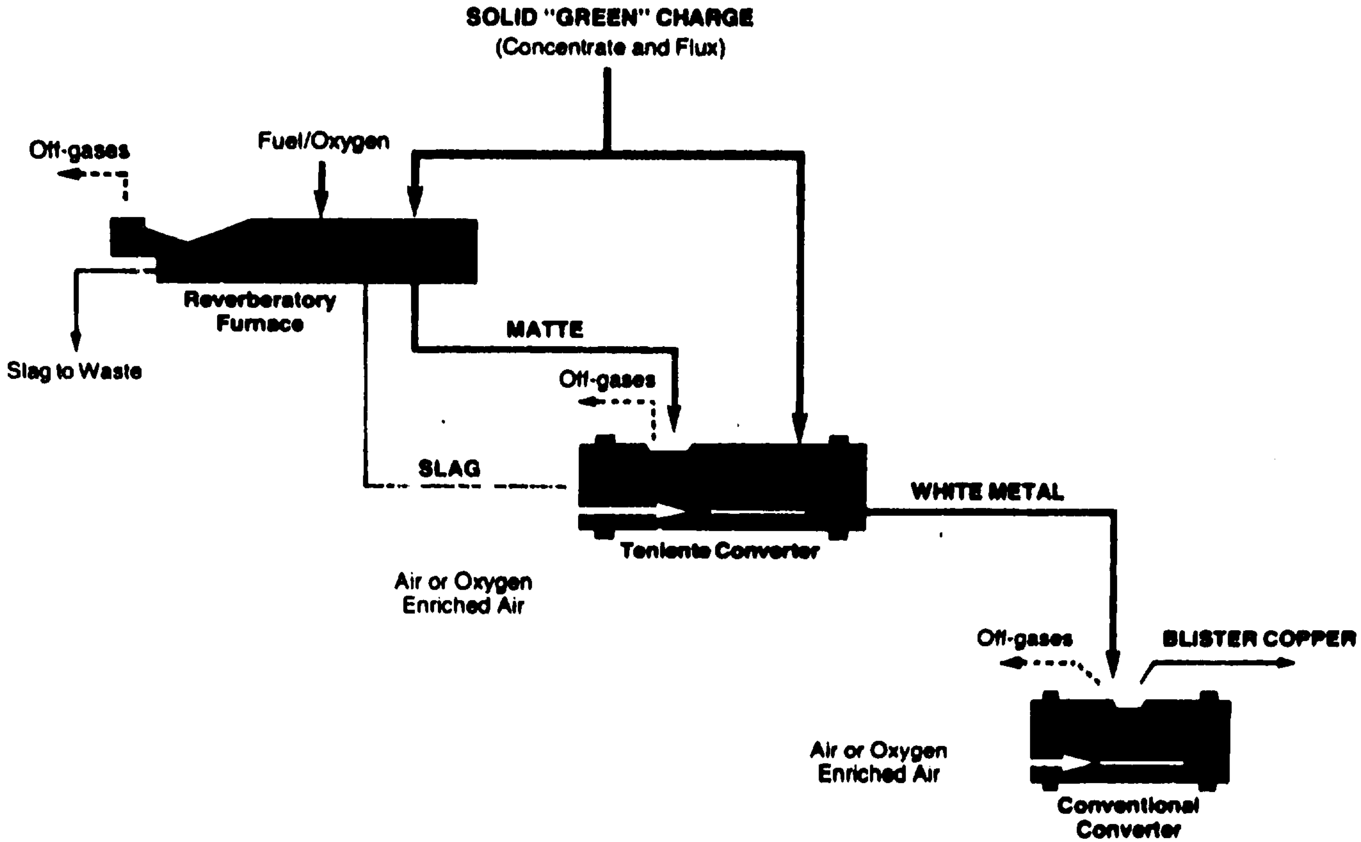


DIAGRAM 17: SCHEMATIC PRESENTATION OF EL TENIENTE PROCESS IN CALETONES

DIAGRAM 18



Caletones smelter schematic operation flow-sheet

Oxy-fuel technology description

The reverberatory oxy-fuel technology allows for partial or total replacement of standard long flame air-fuel burners by higher temperature short flame burners located on their roofs. This improvement on the conventional reverberatory smelting technology results in a more efficient fuel usage.

According to this technique, industrial oxygen and fuel oil are used for combustion purposes. Nevertheless, the oxy-fuel system can be expanded to other fuels-solids or gases-or it can be used combined with air-fuel burners.

The oxy-fuel burners are mounted vertically through the furnace roof and they are located directly above the green feed charge of concentrates and cold dope. This arrangement provides for a more even distribution of heat in the furnace and results in a more efficient utilization of energy. Because of this increased smelting efficiency, higher smelting rates can be easily achieved. By varying the number of oxy-fuel burners used, furnace throughput can be scheduled according to the overall smelter capacity requirements.

In Figure 2, a typical cross-section at one of the Caletones reverberatory furnaces is shown, with the oxy-fuel burners positioned so that the flames hit the base of the charge banks.

Normal daily reverberatory operation conditions consider a green charge (concentrate and reverts) moisture between 7 to 8%. No flux addition is required and nearly 600 to 700 tons per day of converter slag are returned to the furnace. Approximately, 1000-1200 tons of concentrate per day are smelted in each reverberatory furnace.

Matte and slag are normally tapped at slightly higher temperatures than with conventional concentrate reverberatory smelting.

As usual, copper and iron sulphide concentrates (chalcopyrite, covellite, pyrite, bornite, etc) react according to chemical decomposition reactions and 20 to 35% of the sulfur oxidizes to SO_2.
A high decrease in off-gas volume due to the use of oxy-fuel burner technology, results in a considerably higher SO_2 concentration of 5 to 8% by volume. This concentration will vary depending on the amount of air infiltrated into the furnace. Conventional reverberatory furnace smelting, using no oxygen enrichment, results in a SO_2 off-gas concentration of 1.5 to 2%. On the other hand, oxidation of iron sulphide (Fe S) takes place in a small proportion due to magnetite presence in the converter slag, which is returned to the reverberatory furnace.

Operation of oxy-fuel burners differs little from traditional smelting. Operational controls are mainly to maintain an adequate fuel to industrial oxygen ratio in the burners and to keep the burners properly cooled. Due to productivity increases, furnace charging has to be checked more frequently. Material handling activities are consequently intensified and a larger amount of molten material have to be tapped from the furnace.

No special furnace design modifications are required for the implementation of the new

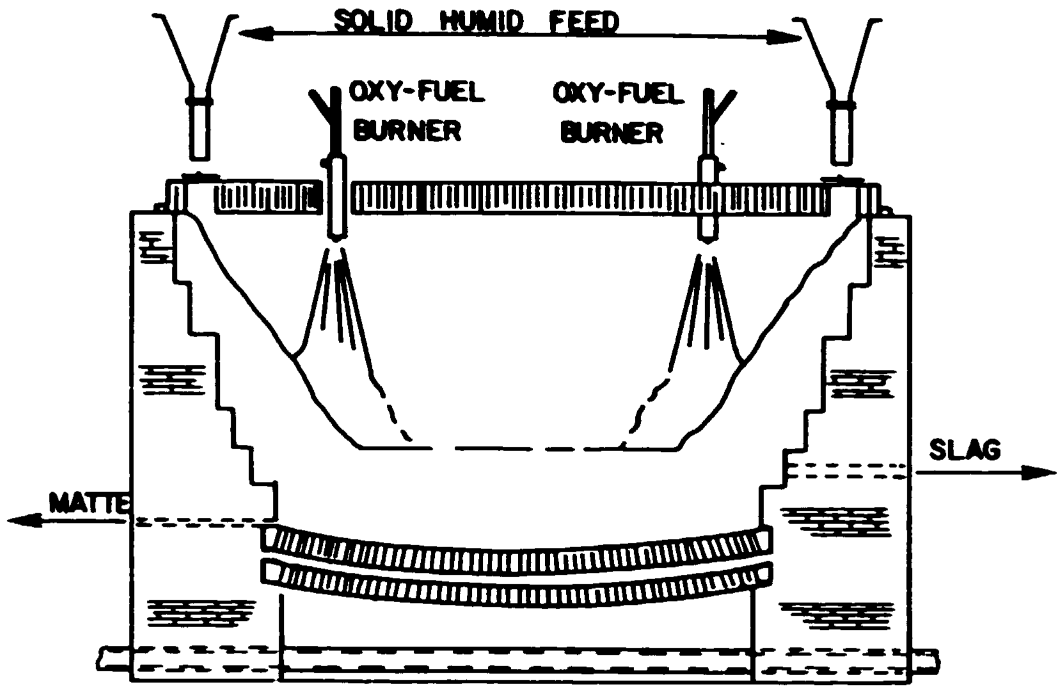


FIG. 10 POSITION OF OXY-FUEL BURNERS IN REVERBERATORY FURNACE.

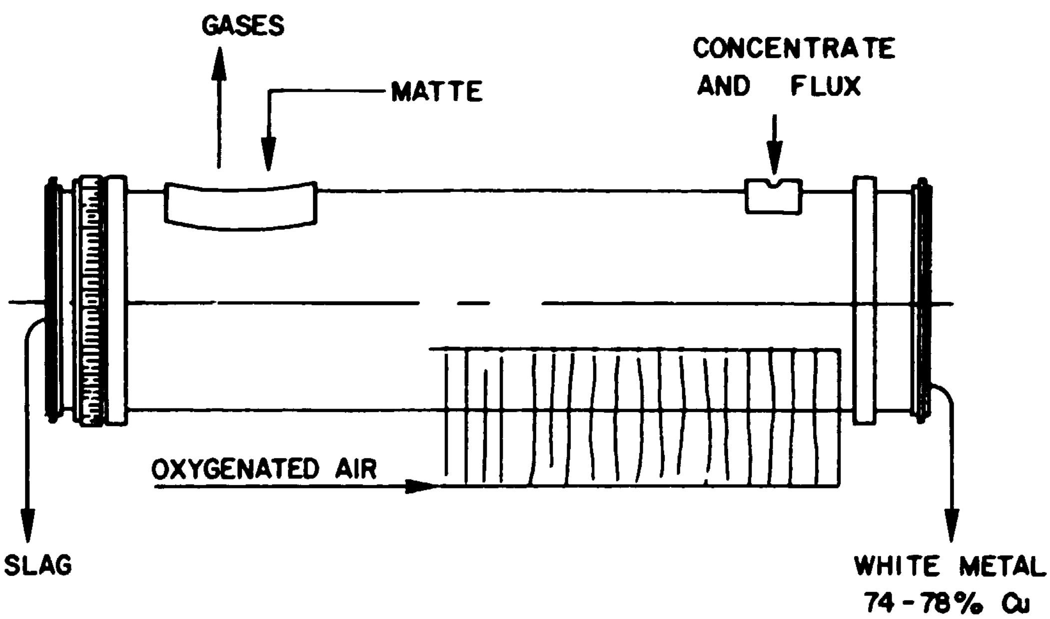


FIG. 11 EL TENIENTE CONVERTOR

These modifications have increased plant production capacity by about 40 percent, reduced fuel consumption by 50% and led to recovery of high grade sulphur dioxide gases. The process is simple and easily adaptable to old smelters with a minimum of expense.

3.56 Discussion - So far the only oxygen smelting process which has won unanimous acceptance is the Outokumpu Process, which has to its credit some 30 operating smelters. The advantages of oxygen smelting are generally weighed against the cost of oxygen production, including investment for a new tonnage oxygen plant. On the positive side, the advantages of oxygen use are not only in economizing on fuels but also in increasing capacity of existing smelters. This may be very important when new investment into an oxygen plant is weighed against not only more economic production costs but also increased production output. In new installations the cost of an oxygen plant should be compared with the economies which come from the necessities of a considerably smaller smelter.

An evaluation of costs for Outokumpu Oxygen Smelting at Ashio, Japan, led to conclusion that with a 21% oxygen air (i.e., normal air from the atmosphere) there was a necessity to consume 2.66 c worth of fuel oil for each pound of copper produced in the form of 48% Cu matte. By increasing the oxygen content of air to 41%, the cost of added fuel was reduced to only 0.37 c/lb, which together with the 0.48 c/lb cost for oxygen, reduced the total cost to only 0.85 c/lb, i.e., by almost three times. Also, there was a bonus in obtaining 60% Cu matte instead of 48% Cu matte.

Under the same price conditions for fuels and energy to produce oxygen, the Inco type Flash Smelter would have had a 0.44 c/lb cost and 50% Cu matte - this because of using 96% O_2 oxygen instead of enriched air. In other words, the Inco cost would be about one half of that of Ashio.

Finally, in the Mitsubishi Process as applied in the Nioshima Smelter, the following comparative figures are available: with a 30% O_2 enriched air, the fuel cost is 1.65 c/lb of copper while the oxygen cost is 0.27 c/lb, which makes a total of 1.92 c/lb. With a 50% O_2 enriched air, the fuel cost is reduced to 0.66 c/lb but oxygen cost increases to 0.86 c/lb, which makes for a total of 1.52 c/lb. In both cases the matte grade is 65% Cu. In this case, then, higher oxygen content reduces cost by about 21 percent. But note that at any rate Mitsubishi smelting costs are higher than both Outokumpu and Inco by from 2 to 4 times (see Table 3).

Obviously, while fuel costs are more or less comparable, the oxygen cost may vary considerably depending on where and with which energy it is produced. It is definitely different if you can install a tonnage oxygen plant on cheap hydroelectric power (in the best case of your own) or if you have to produce oxygen with imported oil as a fuel.

In Chile, the El Teniente Process has received unanimous approval and furnaces have been modified not only at El Teniente, Chuquicamata, El Salvador and Las Ventanas, but also will be introduced at Paipote and probably the Chagres smelter. On the other hand, it is interesting to observe that in the latest expansion of its smelting capacity, Chuquicamata will also introduce an Outokumpu Flash Smelter, in spite of the fact that this smelter will be at a 10,000 ft elevation and thus have less oxygen in the air and also the energy used to produce it will be exclusively thermal.

Apart from this, the El Teniente Process has alreadh aroused interest in Mexico (La Caridad), Yugoslavia (Bor), Zambia, the Philippines (Passar) and other countries.

TABLE 3

COST COMPARISONS BETWEEN CONVENTIONAL AND OGYGEN SMELTING

Costs based on : (1) Power cost - 3.5 c/KWH
(2) Bunker C Oil - $ 225 / MT
(3) Coal at $ 40 /MT

	OUTOKUMPU FLASH SMELTER-at Ashio		INCO FLASH SMELTER at Copper Cliff	MITSUBISHI CONTINUOUS SMLETER at Naoshima	
Percent Oxygen Enrichment	21	41	100	30	50
Power required for oxygen production c/lb Cu	0	0.48	0.44	0.27	0.86
Additional fuel oil required (Bunker C) c/lb Cu	2.66	0.37	0	1.56	0.57
Coal fuel added c/lb Cu	-	-	-	0.09	0.09
Total cost of energy per lb of copper	2.66	0.85	0.44	1.92	1.52
% Cu in matte	48	60	50	65	65

The Noranda Process has not been so successful so far. Apart from three furnaces installed at Bingham and the fourth at Horne, there is little new progress reported. The Mitsubishi Process also hasn't shown much progress after being applied at Naoshima and Kidd Creek. It has been commented that transfer of the liquid metal from one furnace to the other produces problems and limits capacity of operations, which are generally rather cumbersome.

All told, developing nations have very realistic possibilities to improve their costs and expand production capacity at old smelters by a careful consideration of oxygen technology already in operation with a proven record of production efficiency. Table 4 clearly shows how this change has worked in Chile.

At El Teniente, where oxygen technology was first introduced, by injecting oxygen into reverbs, the plant capacity increased about 40% and the fuel cost decreased from $ 25 per ton of concentrate to $ 12 per ton. With this, the direct variable cost decreased from $ 32 to $ 21 per ton. At Chuquicamata, similar changes, although not yet fully introduced, have already decreased the smelting cost (variable) from $ 33 to $ 26 per ton, and further introduction of El Teniente technology and Outokumpu Flash Smelter will bring down smelting costs probably to $ 20 per ton or even lower.

Contrary to this, El Salvador, which is still on old technology (it is already working on introduction of oxygen technology), has variable smelting costs roughly double those of Chuquicamata and El Teniente, at $ 52 per ton of concentrate. True, there is no oxygen cost while refractory and other costs are minimized with use of oxygen, but the overall situation is considerably worse. Cost of fuel oil is from 2 to 2.5 times higher than at Chuquicamata or El Teniente.

TABLE 4

CHILEAN COPPER SMELTING COSTS

(in US dollars per ton of con)

	Chuquicamata	El Teniente	El Salvador
Installed Capacity tpy concentrates	1,000,000	800,000	265,000
Conc. grade % Cu	37.8	38.0	34.0
DIRECT COST			
Variable Cost :			
Fuels	15.35	12.06	29.85
Oxygen	3.34	2.76	---
Refractories	1.83	1.67	0.37
Air	2.17	1.17	0.30
Electric Energy	1.00	0.42	0.07
Others	2.42	2.93	21.53
Total	26.11	21.01	52.12
Fixed Cost	9.92	14.41	4.65
TOTAL DIRECT COST	36.03	35.42	56.77
INDIRECT COST	14.93	17.51	13.95
TOTAL COST / TON	50.96	52.93	70.72
Cost per lb of copper cents	6.11	6.32	9.43

Overall, Chile has very low smelting costs at 6 c/lb, which only compare with Japanese smelting costs, which are even lower. But in comparison with the USA, Canada and Europe, Chilean smelting costs are 3 times lower. This is because, first, oxygen use in Chile is more advanced and, second, it has no ecological protection cost.

4.0 LEAD AND ZINC TECHNOLOGIES

As a general rule, lead and zinc ores appear together in the earth's crust. Also, mostly these ores are complex, i.e., accompanied by numerous other metals, such as copper, silver, gold, bismuth, cadmium, antimony and several others. At Cerro de Pasco, now Centromin, some 21 different products are being recovered from the local ores. This complexity of lead-zinc ores has led to numerous flowsheets for rational recovery of different metal components in the different ore combinations. The major types of ores so far have been lead-zinc-copper ores, lead-zinc ores, copper-zinc ores, and lead-copper ores. In each specific case, special flotation reagents and techniques are used to separate metals to the fullest extent. Also, since each orebody is unique as to its mineralization and impurities content, practically in each case some modifications have had to be made to tailor reagent formulae and flotation conditions for optimization of results. It is, therefore, almost impossible to discuss in general terms the flotation recovery of these ores without referring to specific cases. The only general principle which has been established so far is that when lead, zinc and copper are present together in the ore, copper and zinc should always be separated first because they interfere with each other. In fact, even the smallest amount of copper ion in the pulp activates zinc, and once activated zinc is difficult to separate. In this context, in such a combination of metals, lead and copper are generally floated together into a collective concentrate and then separated by classical means, while zinc is kept depressed and only later reactivated and floated into a separate concentrate.

Apart from this classical formula, few other generalizations can be made and, as already told, each specific ore deserves a specific study and reagent formula for separation of its components. In this sense there are no general rules, and thus we cannot speak here about any specific improvements in flotation technology, except for using more sophisticated equipment and larger flotation cells, which has the same importance as in flotation plants for copper and other metals.

One thing that is true, however, is that no matter how efficient flotation technologies are, separation of principal metal components of ores, i.e., lead, zinc and copper, still requires considerable improvements since much of each metal is being lost in concentrates of other metals. Typically, the overall recovery of metals in such complex ores, when calculated on their recoverable content into a finished concentrate, rarely exceeds 80%. These recoveries are even lower if sulphides are mixed with oxides. In fact, flotation recoveries do not present any difficulties as far as bulk flotation concentrates are concerned. Metal losses start principally in selective flotation.

Given that bulk flotation recoveries easily surpass 90% and that their differential flotation doesn't offer sufficiently encouraging recoveries, the center of research and new development in these last years has moved from flotation to pyrometallurgy. Processes for smelting bulk concentrates have been designed with the subsequent possibility to separate components.

4.1 IMPERIAL SMELTING PROCESS

This process is designed to recover lead and zinc simultaneously from lower grade bulk concentrates. The solvent action of the metallic lead makes it possible to recover also the silver, gold and bismuth content of concentrates together with a substantial part of the copper. It is ideal for treatment of lead-zinc-copper bulk concentrates because it offers an overall recovery of some 95% of the metals. Presently, some 13 plants of this type exist in the world with a proven record of production and efficiency. They generally use a blended feed with a specific ratio of lead and zinc for optimum technology. This makes them suitable as regional custom smelting plants. These plants use waste heat sources to preheat blast furnace air and coke and are reasonably energy efficient. Only copper and cadmium recoveries are poor in this process. In one case, the ISM process didn't work, which is the case of BMS, the New Brunswick lead-zinc smelter, which led to research studies for sulphate roasting of these concentrates.

4.2 SULPHATION ROAST LEACH PROCESS (SRL)

This process is being tested at the CANMET miniplant, which is a continuous process development unit in New Brunswick and whose purpose is to find viable ways to process local bulk concentrates that do not respond satisfactorily to the Imperial Smelting Process. New Brunswick bulk concentrates assay 25 to 30% Zn, 0.8 to 6% Cu, 6 to 11% Pb, 18 to 22% Fe, 36 to 37% S, 1 to 2% gangue materials, with 150-290 grs/t silver and 0.2 to 3 grs/t gold. They are roasted in a first stage in a fluosolid roaster by sulphation roast and then leached in diluted sulphuric acid to obtain zinc, copper, cobalt and antimony sulphates in solution. In the same reaction, lead, silver and gold are precipitated. The leach solution is treated to precipitate copper (by cementation), cobalt and antimony, and zinc is then electrolytically recovered. The lead sulphate precipitate is leached with a strong sulphuric acid at 95°C, and then conditioned with calcium and sodium chlorides to obtain soluble lead chloride, which is subsequently precipitated with calcium carbonate as lead carbonate. The lead carbonate is calcined to obtain lead oxide, which goes into conventional shaft furnace for smelting.

This process, although chemically complicated, is economically justified. It provides 96% recovery for zinc, 95% recovery for both lead and copper and silver recovery of 85 percent. This compares with 85-87% recovery for zinc, 75-80% recovery for lead, 80% for copper and only up to 50% for silver when the bulk concentrate is retreated by flotation to obtain individual products.

The pilot plant operates at 10 tpd of concentrate and the Canadian Government has already invested C$ 18m for its construction. Its economic results are completely comparable to those obtained by the Sherritt Gordon pressure leaching process. In both cases, the net income increases from 12 to 20 percent.

In respect to lead smelting Diagram 19 gives us a summary of all major processes for lead smelting. The direct smelting of lead sulphide concentrates offers significant advantages over the conventional roast-reduction blast furnace route. In this context, in these last years extensive research and development activity has been developed in the four indicated approaches, but in spite of all their energy efficiency and potential elimination of ecological problems, few industrial installations have yet been built.

4.3 OUTOKUMPU PROCESS

The Outokumpu Flash Smelting Process, widely used for smelting copper and nickel concentrates, is now being adapted to flash smelting of lead concentrates. Details of this process are shown in Fig. 12. In fact, this process was developed some 20 years ago but was discontinued because of some problems due to high fume carryover. The present design, shown in Fig. 12, has solved these problems. Lead concentrate, limestone, pyrite and silica fluxes are dried and fed pneumatically to the distributor or burner of specialized design, mounted on the roof of the reaction shaft. This distributes the feed and provides a suspension in oxygen or oxygen-enriched air, which smelts as it descends the reaction shaft. Normally the objective is to

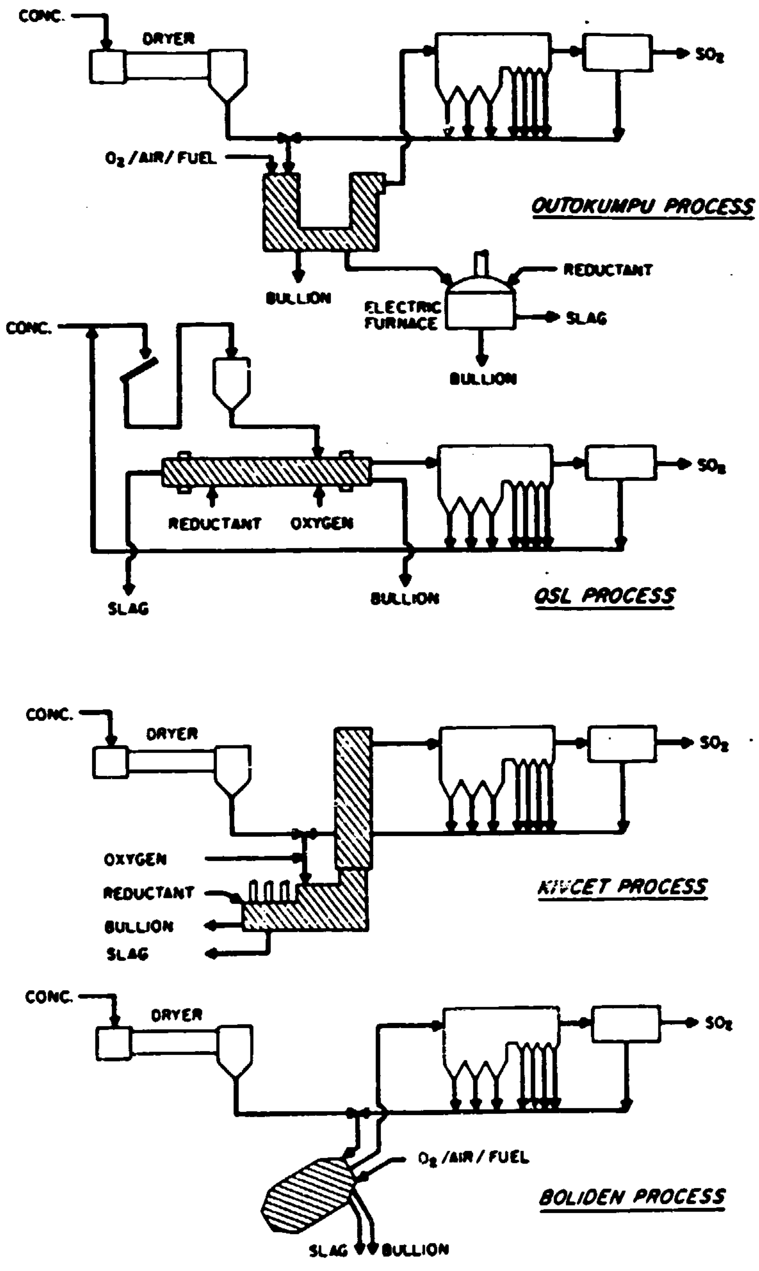


Diagram 19 : Direct Lead Smelting Processes

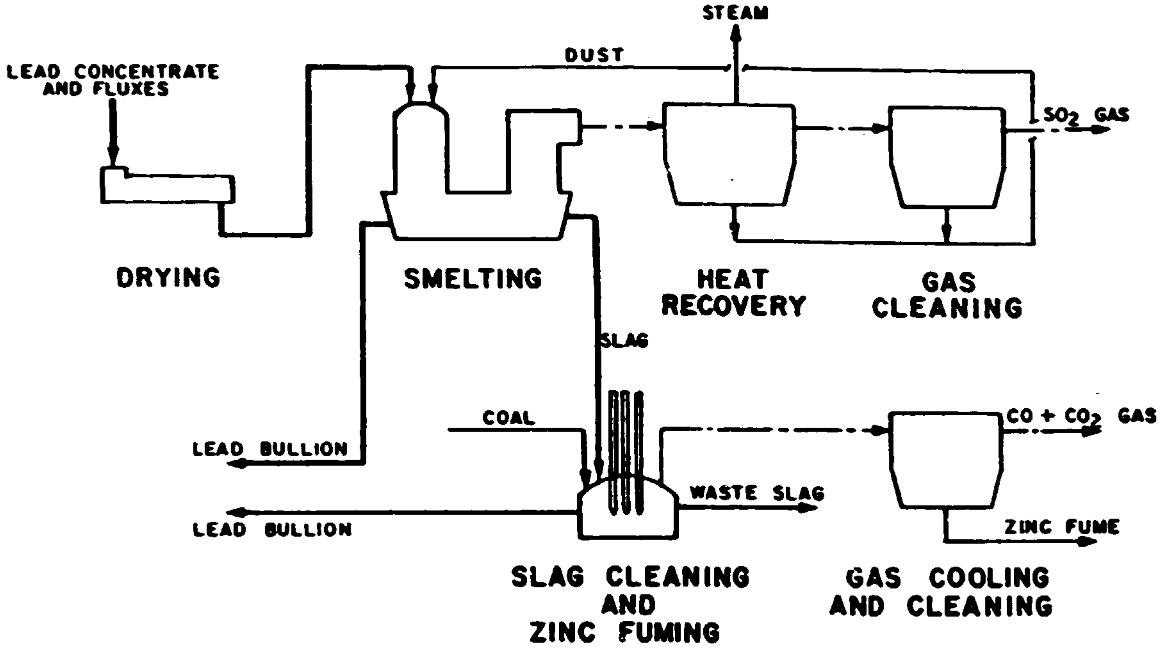


FIGURE 12. OUTOKUMPU LEAD FLASH SMELTING PROCESS

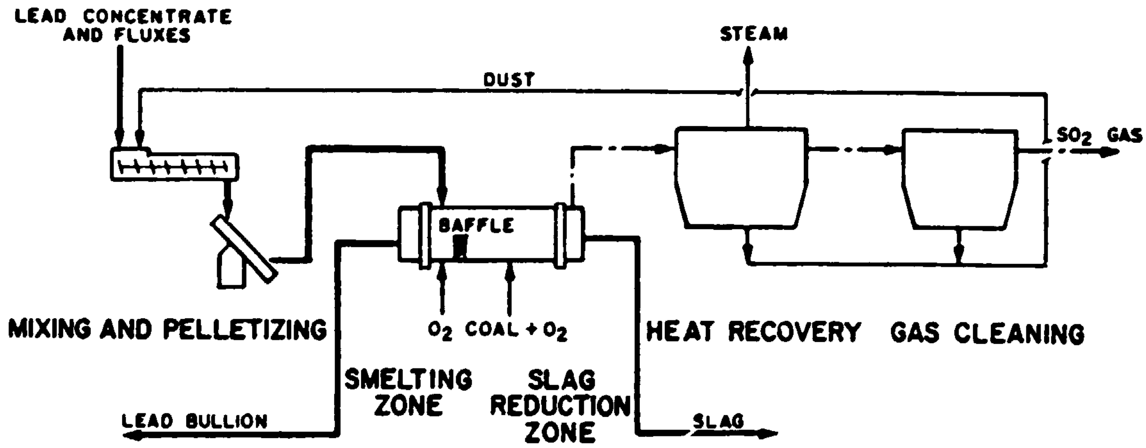


FIGURE 13 QSL LEAD SMELTING PROCESS

smelt autogenously using commercial oxygen so as to minimize offgas volume and dust to carryover. This produces offgases containing up to 70% SO_2. Material carried over to the gas cleaning system is oxidized and reports as sulphates.

Flash furnace slag, containing 20 to 40% Pb, is continuously skimmed to an electric furnace where it is reduced using coal injection. An excess of coal - 10 to 20% - is used in the reaction, which provides high reaction rates and reduces the electric furnace size. Discard slag contains between 1 and 3% Pb.

Electric furnace offgas is shock cooled in a wet scrubber, from which the recovered dust contains metallic lead and zinc. A certain amount of zinc is volatilized in the electric furnace, at a minimum 20 to 25%. Zinc removal can be increased to recover zinc and up to 80% Zn volatilization has been achieved in the pilot plant, with waste slag running 3-4% Zn and less than 1% Pb.

4.4 QSL PROCESS

The other very promising lead smelting process, just recently developed, is the QSL Process, which stands for its inventors Quenau, Schuman and their sponsoring firm Lurgi of West Germany. Details of this process can be perceived from Figures 13 and 14 and Diagrams 20 and 21 . Its advantages are that this is the only continuous lead smelting process, and that all reactions happen in the same reactor.

Concentrates, fluxes and reverts are pelletized and charged to the reactor, without prior drying. The reactor is a horizontal magnesite-lined kiln equipped with injectors (tuyeres) along the bottom. The kiln rotates through 90° to clear the injectors when the operation is suspended.

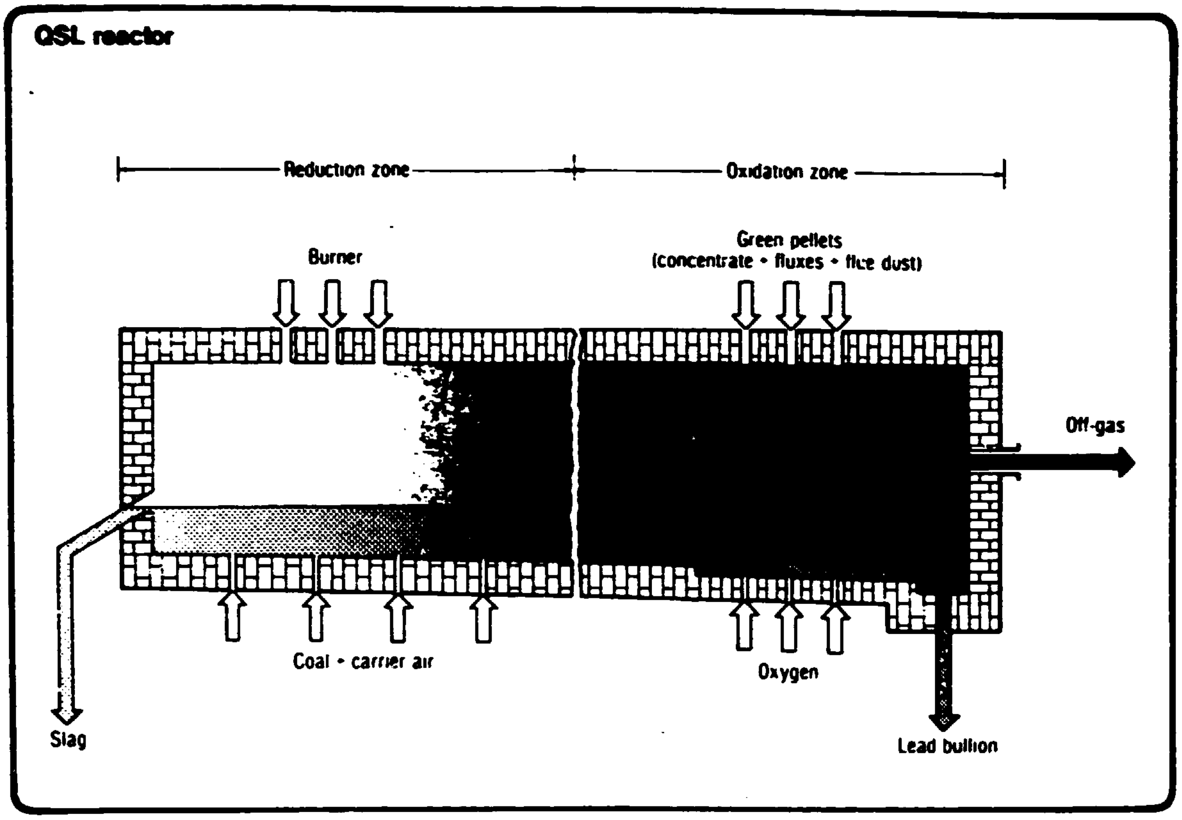


Fig. 14 QSL kiln-reactor for continuous lead smelting

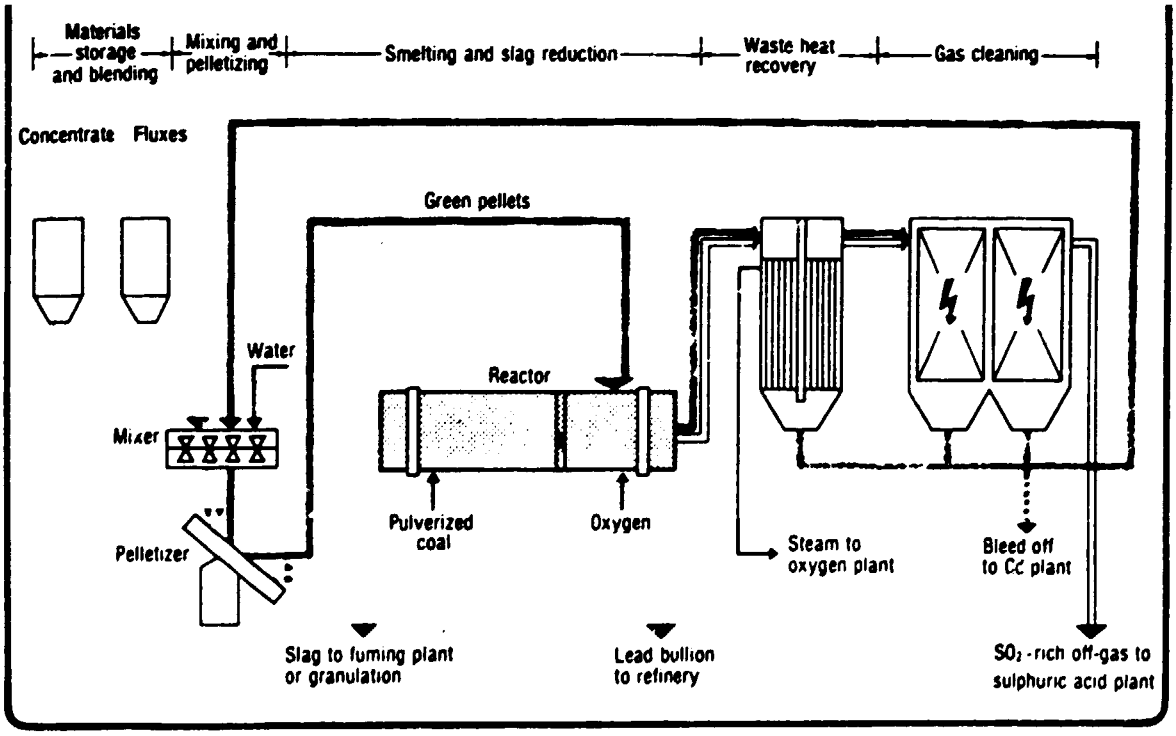


Diagram 20 - QSL Lead Smelting Process

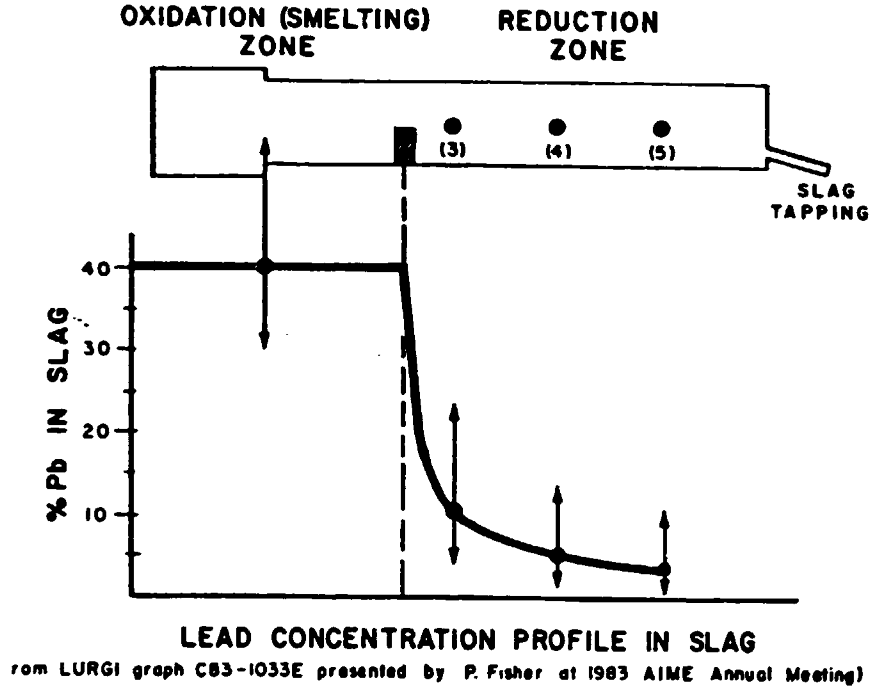


Diagram 21 : Lead concentration profile in slag

The pelletized charge is dissolved in the molten bath and oxidized by commercial oxygen injected from the bottom. This produces a low sulphur bullion, the PbO slag with 40-50% Pb and 15 to 20% SO_2 offgas. Lead content of the slag is reduced by injection of a pulverized carbonaceous reductant prior to being continuously skimmed and granulated. A series of burners are located over the skimming end to raise slag temperatures from 950°C to 1150°C.

Bullion is collected at the bottom of the kiln and flows back to the bullion well. Sulphur content of the bullion is only 0.2%. Diagram 21 clearly shows how the lead content is continuously reduced in slag along the kiln length.

A QSL demonstration plant has now been operating at the Berzelius lead-zinc smelter in Duisburg, West Germany, since 1981 It has processed over 20,000 tons of concentrates ranging from 50% to 75% Pb and 1 to 7% Zn.

The process advantages claimed are: (1) its continuous operation in a single reactor; (2) great flexibility in respect to lead and sulphur content of concentrates; (3) use of heat of oxidation of sulphides for smelting the charge; (4) low capital and labor costs, which are 30% lower than in conventional smelters; (5) high possibilities for automation of operations.

4.5 KIVCET PROCESS

Another emerging technology is the Kivcet lead-zinc process, developed in the Soviet Union and marketed by the West Germany's Humbolt Wedag AG. This technology was developed in Siberia at Ust Kamenogorsk in a 25 tpd pilot plant, and already two plants have been sold in the West: one to Bolivia for Karichipama and the other to Italy for Sardinia.

As shown in Figure 15, lead concentrate, flux and returns, after being dried in a rotary dryer, are fed under high pressure simultaneously with oxygen into a shaft furnace through a special burner. Flash smelting commences shortly after the injection of the charge, raising the temperature to about 1400°C. Smelted charge is collected at the hearth of the electric furnace where coke breeze is added to maintain a reducing atmosphere. Slag, with less than 3% Pb and 3% Zn, is skimmed and discarded. Lead bullion containing about 0.1% S is tapped continuously for refining by conventional methods. Shaft offgas is high in SO_2 - 30 to 55% - and suitable for sulphur fixation as liquid SO_2. The process has the advantage of being able to recover byproduct zinc by fuming from the electric furnace compartment. In the case of the Bolivian plant, where lead concentrates fed contain 7.5% Zn, an estimated 85% of the zinc will be recovered as oxide.

This process was seriously considered and evaluated by the two largest Western lead producers, Cominco in Canada and Broken Hill in Australia. Both companies, however, appear to be convinced of using the flash smelting and electric furnaces from this process, while they consider zinc fuming in other units. This is probably because such units already exist at Trial and Port Pirie. Also, it is believed that separation of the smelting unit from the fuming zinc section will provide more flexibility for the process. This process is very similar to the Outokumpu Process.

4.6 DISCUSSION

In the treatment of complex lead-zinc sulphide ores two new basic approaches have emerged in these last years: one, which tries to start pyrometallurgical treatment of bulk concentrates right from the beginning, without previous separation of individual concentrates, and the other, which improves technologies for treatment of the individual concentrates.

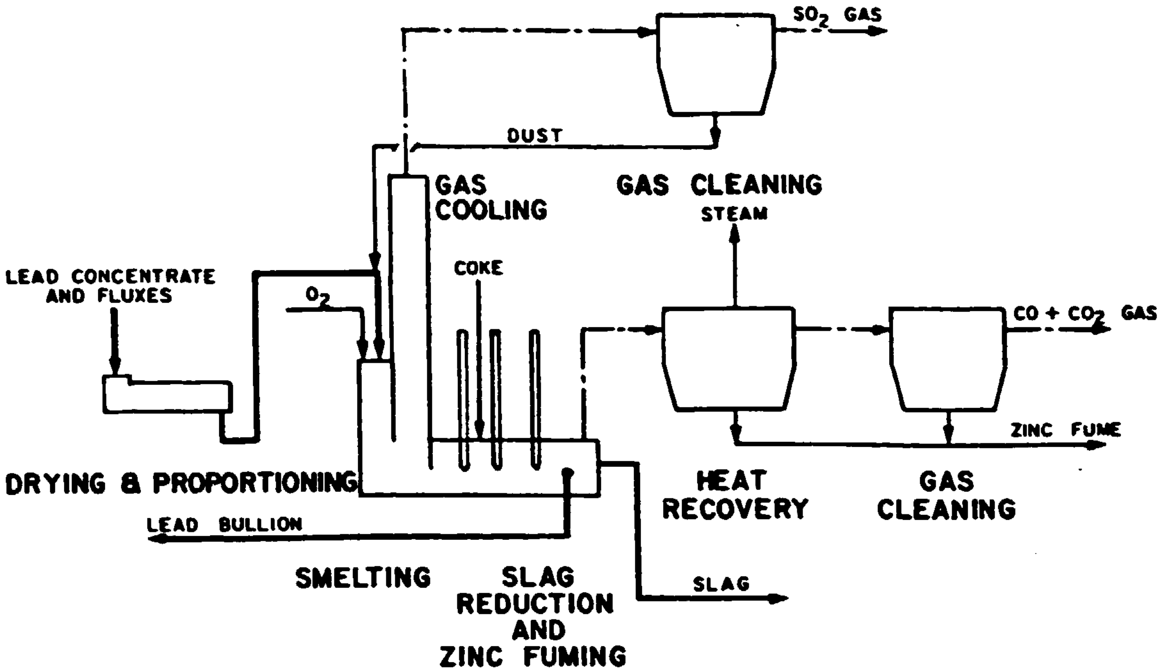
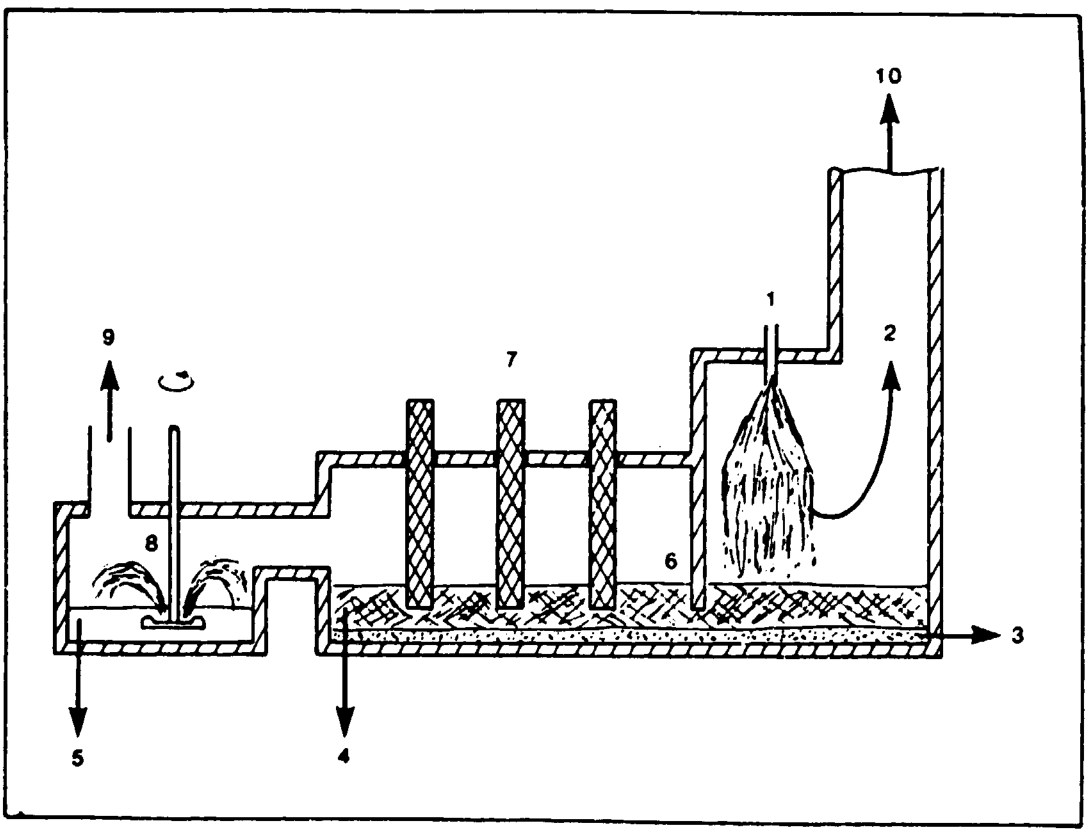


FIGURE 15. KIVCET CS LEAD SMELTING PROCESS



KIVCET PROCESS

1. Concentrate burner
2. Fume-laden gas
3. Lead tap
4. Slag tap
5. Zinc tap
6. Water-cooled under-flow wall
7. Electrodes
8. Lead-splash condenser
9. CO off-gas
10. To electrostatic precipitator

Figure 16 : KIVCET FLASH SMELTING FURNACE

In the first case, excessive loss of metals in their flotation separation is avoided, and typically 90-95 percent metal recoveries are obtained against the average 80 percent recoveries by other classical methods. The most outstanding in this respect is the Imperial Smelting Process with 13 industrial installations to its credit so far. However, in some cases the Imperial Smelting Process is not quite suitable for solving all problems, and new, chemical processes are being developed for the same purpose. Today, these are the Sulphatation Roast Leach Process, being developed by Canmet in New Brunswick, and the High Pressure Leach Process, under development by Sheritt Gordon, also in Canada. We consider the High Pressure Leach Process interesting, but not entirely suitable for developing countries because of the high technology and investment costs involved.

In the area of direct smelting of concentrates, like with copper, we have two types of new processes in development: those which use bath smelting, such as the Boliden Kaldo (TBRC) Process, and the QSL Process. The QSL Process looks to us considerably more attractive and suitable for developing countries because of its relative simplicity and flexibility. Also, it has potentially lower costs.

The other group of direct smelting furnaces are the flash smelting technologies as developed by Outokumpu and Kivcet. Both technologies seem to have their own merits, and consideration of their application in developing countries should be studied from the point of view of investment and production costs as well as from the point of view of effective transfer of technology and further follow up in potential troubleshooting. Developing countries normally have little previous experience with new high-tech technologies, often rely on turnkey jobs and then experience high losses in production when operational or control problems appear.

At any rate, it should be clear that these emerging technologies should be still convincingly proven in full scale industrial plants. This experience should be preferentially acquired first in industrially developed countries where ample technological and scientific infrastructure exists. Moreover, the very nature of the problems and commercial considerations strongly advise that developing countries use only an industrially proven and confirmed technology.

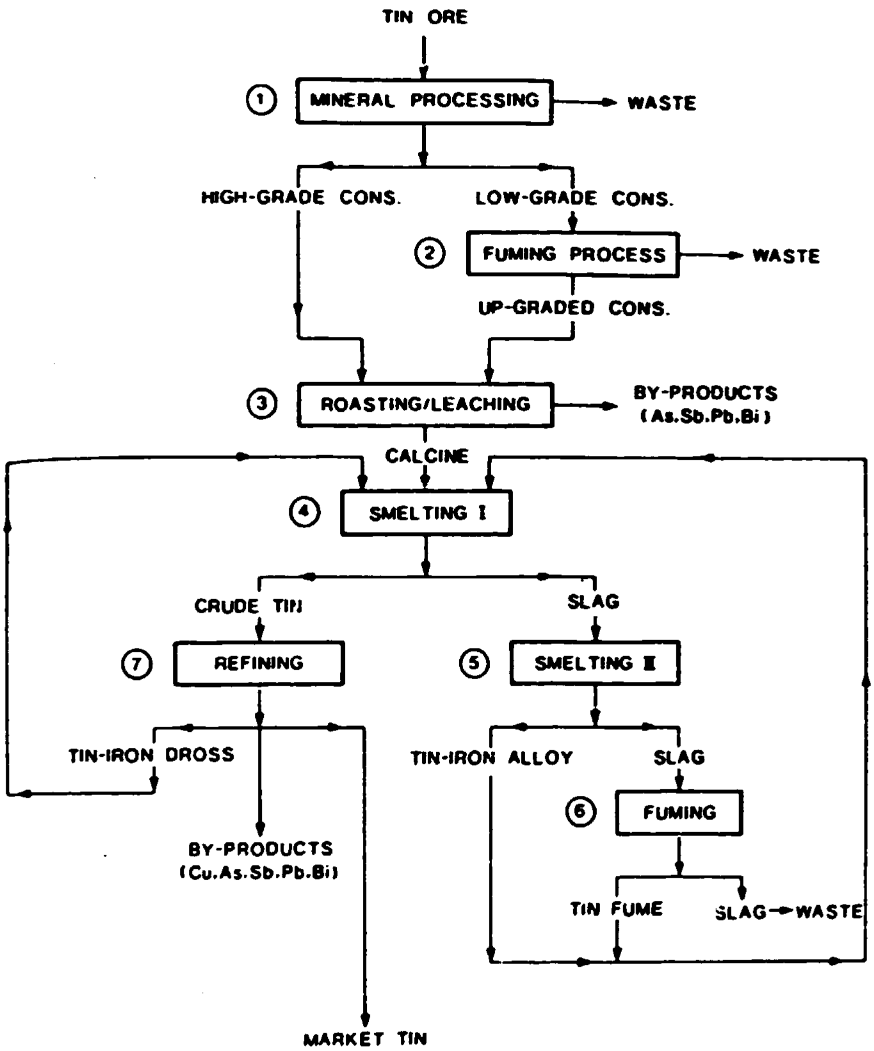
5.0 TIN TECHNOLOGIES

5.1 PROCESSING CHANGES

In a relatively recent survey, T.R.A. Davey indicates that a generalized flowsheet from tin ore to finished metal is as indicated in Diagram 22. Up to 1970, the almost universally used flowsheet consisted of mineral processing to a high-grade concentrate, containing over 60% Sn, followed when necessary by roasting/leaching to reduce the content of "dirty" elements (Pb, Bi, As, Sb) before two stage smelting. Only exceptionally was the second-stage slag fumed for additional tin recovery. If it contained substantially above 2% Sn, it might be subjected to yet a third smelting stage, to produce a tin-iron alloy for return to the first stage. The discard slags from the second stage smelting rarely contained in fact less than 1% Sn as usually was claimed in the literature.

During the 1970's, an increasing proportion of lode tin has been mined, due to the gradual exhaustion of sources of alluvial tin. It has become increasingly difficult to obtain high-grade tin concentrates at a high recovery from lode material, and not only was the average tin content decreasing, but the concentration of associated sulphide impurities, particularly of Pb, Bi, As and Sb, started to increase.

Rather than lose increasing amounts of tin by attempting to upgrade the concentrates, there has been a trend towards the fuming processes (item 2, Diagram 22), which can give a medium grade concentrate of 40-50% Sn at high recovery rates of over 90%. This compares with 50% or less recovery for obtaining concentrates of about 60% Sn by mineral processing methods. Furthermore, elimination of S, Pb, Bi, As and Sb by mineral processing cleaning stages often also entails unacceptable



Diag. 22 GENERALISED TIN FLOWSHEET

tin losses, and tends to be replaced by treatment at a smelter: roasting/leaching (item 3) before smelting, or refining processes of the metal after smelting (item 7).

Roasting (in air with or without addition of NaCl for chloridizirg) followed by leaching in hot water or diluted acid has long been practiced and is no innovation. However, the increased need to cope with greater impurity levels has led to new developments in refining, both electrolytical and pyrometallurgical.

In an endeavor to recover more of the fine tin produced by ever finer grinding to liberate cassiterite, particularly if this is intimately associated with sulphide minerals, flotation has been widely introduced (see next chapter), not merely to float sulphides away from cassiterite concentrates, but also to float cassiterite from the gangue minerals. Although the production of this flotation concentrate can boost tin recovery significantly, by 20% and more, the product is very low-grade, around only 20% Sn, and calls for new methods of treatment. In some cases, the problem is solved by shipment of these low grade concentrates to classical smelters, where their impurities are diluted by blending with high-grade concentrates. In the future, these concentrates will probably be upgraded more by fuming, and impurities recovered as by-products by roasting and leachinj techniques or by refining processes.

Fuming, in favorable cases, may replace mineral processing altogether to produce a concentrate directly from the ore. But this certainly requires high-grade ores to start with. Fuming normally requires products of 7% Sn and more, but in the case of ores this does not apply if the ore itself contains combustible minerals sufficient for autogenous smelting. This refers to highly pyritic and pyrrhotitic materials, where sulphur constitutes high fuel values.

5.2 TIN FLOTATION

Tin ores have been mined since antiquity and successfully concentrated by gravity methods. Since the only tin mineral of major importance is its oxide, cassiterite (SnO_2), when the flotation process was developed at the beginning of this century for the concentration of non-ferrous base metals, and particularly their sulphides, it neglected beneficiation of tin ores.

The main source of tin was always placer deposits, which were normally exploited by dredges, which then passed the material over grizzlies and trommels for classification and then to cyclones for dewatering and desliming. Deslimed material was then concentrated in jigs and cones until high grade concentrate was obtained. Sometimes up to four stages of dewatering in cones and concentration in jigs were necessary to obtain the necessary purity of the final concentrate (60% Sn).

With the gradual exhaustion of high grade placer deposits, mining of hardrock deposits started, which eventually led to beneficiation of lode deposits. At the beginning, gravity methods were applied for their concentration, but as mineralization changed to fine grained highly disseminated cassiterite, metallurgical losses in gravity concentration increased and recoveries dropped to 50 percent and less. The other problem of lode deposits has been that they contain sulphides of other metals, such as those of copper, iron and zinc. These sulphides follow tin in its concentrates and must be removed before smelting. At this stage, flotation technology was introduced for cleaning sulphides from cassiterite concentrates. This eventually led to the flotation of cassiterite itself.

Development of an effective flotation process to recover fine cassiterite has long eluded the industry. However, in the last decade or so considerable progress was made. The idea is principally to concentrate fine cassiterite particles in a low-grade concentrate. These particles normally escape gravity concentration methods and report to tailings, which after conditioning with specific flotation reagents are treated in flotation cells, leading to the additional recovery of tin which increases the overall recovery of the metal by about 20 percent, i.e., to 65% or 70% instead of between 48 and 50 percent before.

Since cassiterite is an oxide, it should be treated for flotation with reagents which are normally used for flotation of non-metallics or oxides of heavy metals. In this respect, particularly successful have been proven oleic acid, cetyl sulphate, hydroxamate, phosphoric and arsenic acids and Aerosol 22, which is a sulphosuccinamate.

Of all reagents so far used, the biggest success has been achieved with Aerosol, which has been tested also on a pilot plant and industrial scale. In a Cornwall plant in the UK, through the treatment of a complex ore containing 1.26% Sn, 2% Zn and 0.4% Cu, gravity concentration produced a 40% Sn concentrate, and flotation produced a 30% Zn concentrate, a 5% Cu concentrate and a 14% Sn concentrate obtained from flotation of fine slimes. This increased overall tin recovery from 50 to 65 percent, and recovery for sulphides was about 70 percent. Flotation of the deslimed tin material, which is 80% minus 53 microns, is carried out in an acid circuit at pH 2.4 using a sulfosuccinamate collector and citric acid as a modifier to upgrade fine fractions from 0.8% Sn to 10% Sn and more. The flotation concentrate is then upgraded to 14% Sn using high-intensity wet magnetic separators to remove turmaline and other slightly magnetic material.

As much cassiterite is being lost today in fine slimes and as ores supplied to present mills have an increased content of the metal in them, flotation is becoming one of the most promising new technologies for recovering this material and for improving plant technology. Much research is still necessary to be done in order to achieve a greater selectivity of flotation reagents and contribute to higher grade products. We understand that some of this research is already being carried out at the Oruro Mining and Metallurgical Research Center in Bolivia and could be expanded to the other tin producing countries.

5.3 PYROMETALLURGY OF TIN

About 75% of all tin is found in placer deposits, which are mainly found in Thailand, Malasia, Indonesia and Central Africa. These deposits, although low grade, undergo wet gravity methods rather satisfactorily, producing high grade concentrates with a satisfactory recovery of 65 to 75 percent.

The other quarter of tin production comes from the so called lode deposits which are found in Bolivia, Australia, Russia and China. These deposits, although of higher grade, are complex in mineralization and produce impure concentrates which are expensive to treat, as discussed in the previous chapter. This leads to a natural tendency to improve our technology at the pyrometallurgical stage if it cannot give satisfactory recoveries in the beneficiation stage.

In this context, great attention has been paid in these last years to the development of a cheap and effective method for tin recovery from slags by the fuming process, as discussed in the tin first chapter. In the past, this was justified by relatively high tin prices which would permit such treatment. Today, because of the catastrophic fall in tin prices, this alternative remains in doubt. At any rate, it should be explored for better days.

Here the main problem is that in a smelting process the tin content of slags critically depends on the iron content of concentrates: the higher the iron content of concentrates the higher is the corresponding loss of tin in reject slags, and this is a problem which had become to be regarded as almost inevitable. Fuming or recovery of tins from slags by volatilization has been found so far to be the most promising step in this direction. For this purpose, tin is converted into one of its most volatile products, halides or stannous oxide, and is eliminated by distillation. So far, the only economic way to do it has been with chlorides. Much of this research has been done in the USSR by Kolodin and other scientists.

In Figures 17 and 18, we give examples of a conventional tin smelting circuit and of a smelting circuit for low-grade tin concentrates.

In the early days, tin volatilization from stannous slags was attempted in long kilns, but was abandoned because of poor results. Then Phelps Dodge developed a commercial installation in a Pierce-Smith type convertor fit with special tuyers. Liquid slag was fed in from a crane, and was blown with a mixture which consisted of a pyrite suspension in air and light fuel oil. Blowing time was that required to add the necessary quantity of pyrite for reaction with the slag. The reject slag from this process contained only 0.5% Sn.

Then came the Kolodin furnace, shown in Figure 19, which is the most widely used today. This is basically a water-jacket shaft with a water cooled cast iron hearth. It operates with six tuyers on each of the shorter sides to blow a coal/pyrite mixture (today coal is often replaced by oil) into the bath of liquid slag, a little under 1 metre in depth, with a given quantity of air to produce reactions the same as in the convertor described above.

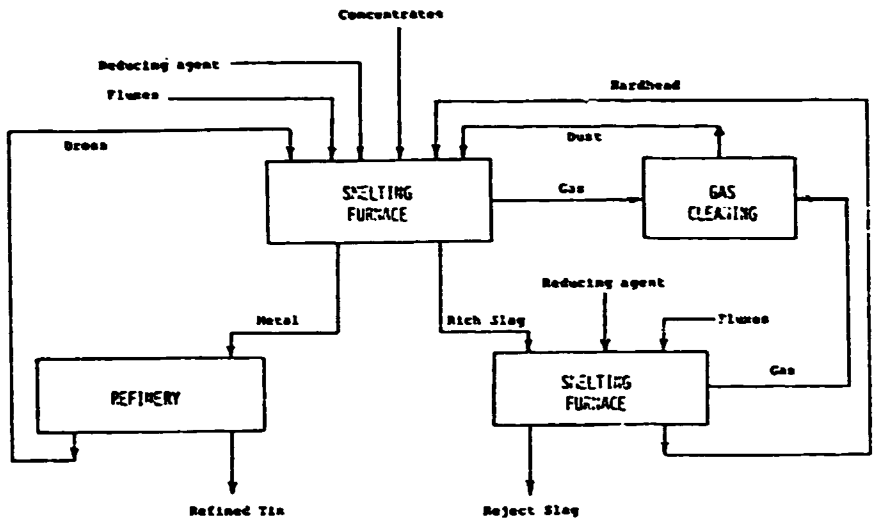


Figure 17 The conventional tin smelting circuit.

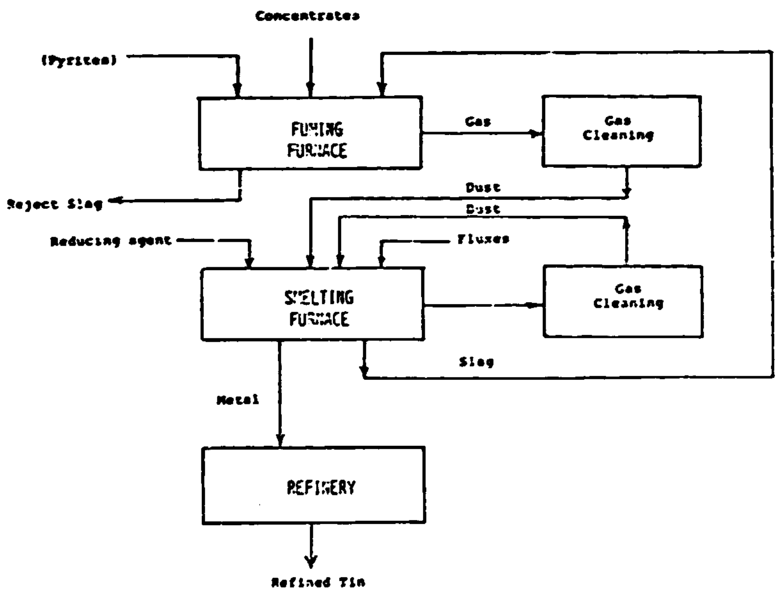


Figure 18 Smelting circuit for low grade concentrates.

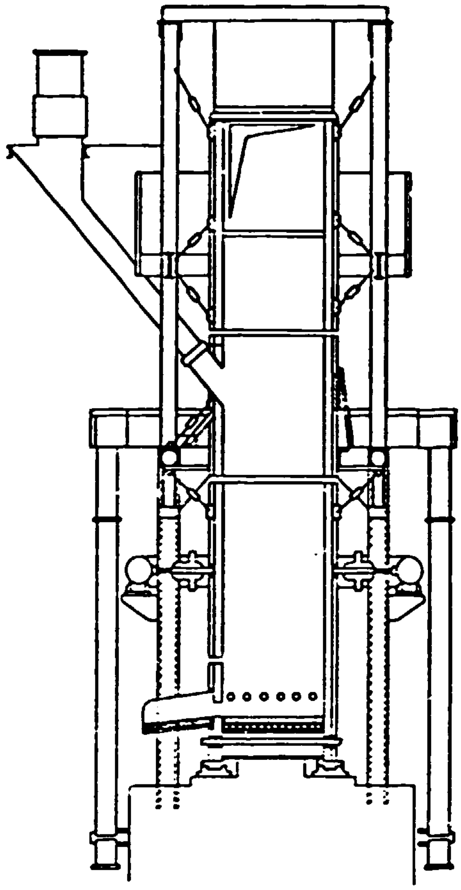


Figure 19 The stationary tin fuming furnace as designed by Kolodin.

ANNEX I. Technological Alternatives for the Fabrication of semi-finished and finished products of copper

I. COPPER WIRE ROD MANUFACTURING

Copper's major field of application is the electrical sector, where the high electrical conductivity, good physical properties, relatively high strength and corrosion resistance of copper are of great importance. In this sector copper is overwhelmingly used in the form of wires, cables, magnet wires, etc. Growing quality requirement and efforts of decreasing production costs resulted in revolutionary changes in the main technological routes.

Up to 1965, the standard processes involved the separate operations of wire bar casting, hot rolling and pickling. Although during this period the technologies were steadily advancing, particularly with respect to rolling mills and introducing continuous casting of oxigen-free billets, significant contraints were imposed by the discrete nature of the bars /typically weighing about 100 kg/, as well as the need both to reheat the bars for hot rolling and to relatively short lengths of rod during drawing.

Within the last decade, this procedure has largely been displaced by continuous processes. These feature the tandemisation of operations spanning melting of a cathode in-feed and coiling of bright rod in length, in effect, restricted only by the available handling facilities. Bearing in mind the relatively short time during which this dramatic change has occured, it is remarkable that by the end of 1984, the number of continuous processing units for re-draw rod, either installed or expected to be installed was around 100.

The success of the continuous processes is mainly due to bold innovations and improvements in design of equipment.

An important contributory factor has been the steady increase in the availability of high-purity cathode copper from electrolytic refineries. This situation has promoted the development of furnaces for the continuous melting of cathode, for direct transfer to a holding furnace associated with a continuous casting machine. Consequently, it has become possible to produce copper rod of both the tough-pitch variant and virtually oxygen-free, suitable for the most critical electrical conductor applications.

A) CONTINUUS-PROPERZI

The Properzi non-ferrous metals casting process was developed first for zinc and lead, and was later adapted to produce aluminium rod. In 1960 the company built its first copper rod caster, which had a capacity of 10 tph. This plant was sold to the USA's Southwire. A second caster was installed in the USSR in 1962. As rolling torques and loads are similar for aluminium and copper casting, Properzi was able to draw on its experience in aluminium rod casting in the development of the copper caster.

The "second generation" of copper casters was developed in the mid-1960s. These casters had capacities of 25 tph. Three were installed in Sweden, Greece and Italy. This range of casters was particularly aimed at high-capacity producers. Following requests from smaller producers, Properzi developed a "third generation" of casters which had smaller capacities and required low capital investment. Costs were reduced by the replacement of continuous shaft furnaces with reverbatory furnaces.

This new range of casters, including the modified "second generation" casters, had capacities ranging from 5 to 30 tph.

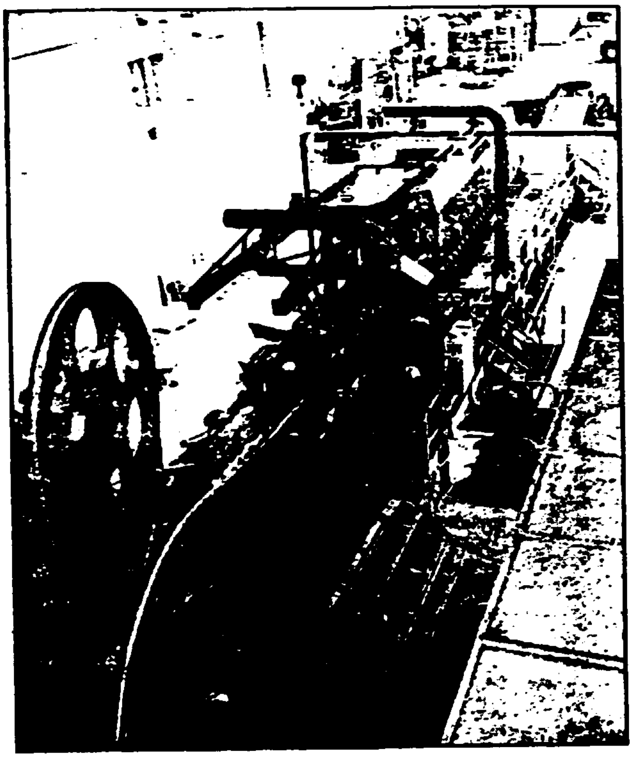


Figure 1. A Continuus-Properzi continuous copper rod casting line

There were two main types : casters with capacities up to 10 tph with either one or two reverbatory furnaces (designed for small and medium-sized producers); and casters with capacities over 10 tph with vertical shaft furnaces.

In 1977 Properzi developed a "micro" mill which was capable of cold rolling from a diameter of 8-10mm down to 1.5mm at a speed of 45 metres per second. The mill was designed to replace the traditional drawing process for small-diameter copper rod.

At present there are four standard Properzi copper casters which are built by Continuus-Properzi : Model Cu/2500-8/19 has a capacity of 25 tph and produces 6.35-8mm dia rod; Model 7E-Cu/1800-8/13 has a capacity of 14 tph and produces 8mm dia rod; Model 6E-Cu/1400-8/13 Mini has a capacity of 7 tph and produces 8mm dia rod; Model 6E-Cu/1400-8/13 Mini/S has a capacity of 4 tph and produces 8mm dia rod.

In the casting plant, the well-established two-wheel casting process is utilised. The lower casting wheel and the upper "idle" wheel are both encircled by a continuous steel belt which closes the ring mould fitted to the periphery of the lower wheel, to form the casting chamber and determine the shape of the bar . The casting ring mould has an outside diameter of 1,500mm.The caster's cooling system includes two headers which are fitted with separately adjustable valve sprays to allow variations in cooling.

The cast bar is guided through a conveyor on to an automatic shear unit which is fitted with a pinch roll to feed the shearing head. The bar is cropped continuously and cropped ends are collected in a water cooling tank. Immediately downstream of the shear unit is a trimming unit.

Following trimming, the bar passes through a brushing unit. This unit is fitted with four rotary steel wire brushes which remove the thin surface oxide layer and any burrs left by the trimming process. The bar is then fed to the rolling mill by means of a pinch-roll.

B) THE SOUTHWIRE PROCESS

The first Southwire Continuous Rod (SCR) line completely designed for copper went into operation at Carrollton, Georgia, in 1965, a development of the company's copper bar caster, which had its first run in March 1963. Over the years the system has been improved by continuing development of the shaft furnace, molten metal handling, casting, rolling, pickling, coiling and packaging of coils, and can be designed to meet the requirements of individual customers.

The normal run circle for a SCR copper system is 16-40 hours , with changeover time for new belts of about 30 minutes. Raw material is charged into a melting ASARCO shaft furnace, and the molten metal is then transferred to a holding furnace, by means of a covered launder. Again via a launder the molten copper flows to the pour-pot mounted on the casting machine, from which it is transferred by the pouring spout to the grooved periphery of the casting wheel. A steel band encloses much of the wheel's circumference, thus forming the casting cavity in which the molten metal solidifies. After solidification, a cast bar leaves the cavity by means of an adjustable stripper shoe mounted on the casting machine above the casting wheel which extracts the casting from the wheel during start-up.

An extractor conveyor then alters the movement of the bar from the vertical to the horizontal plane, so that the bar

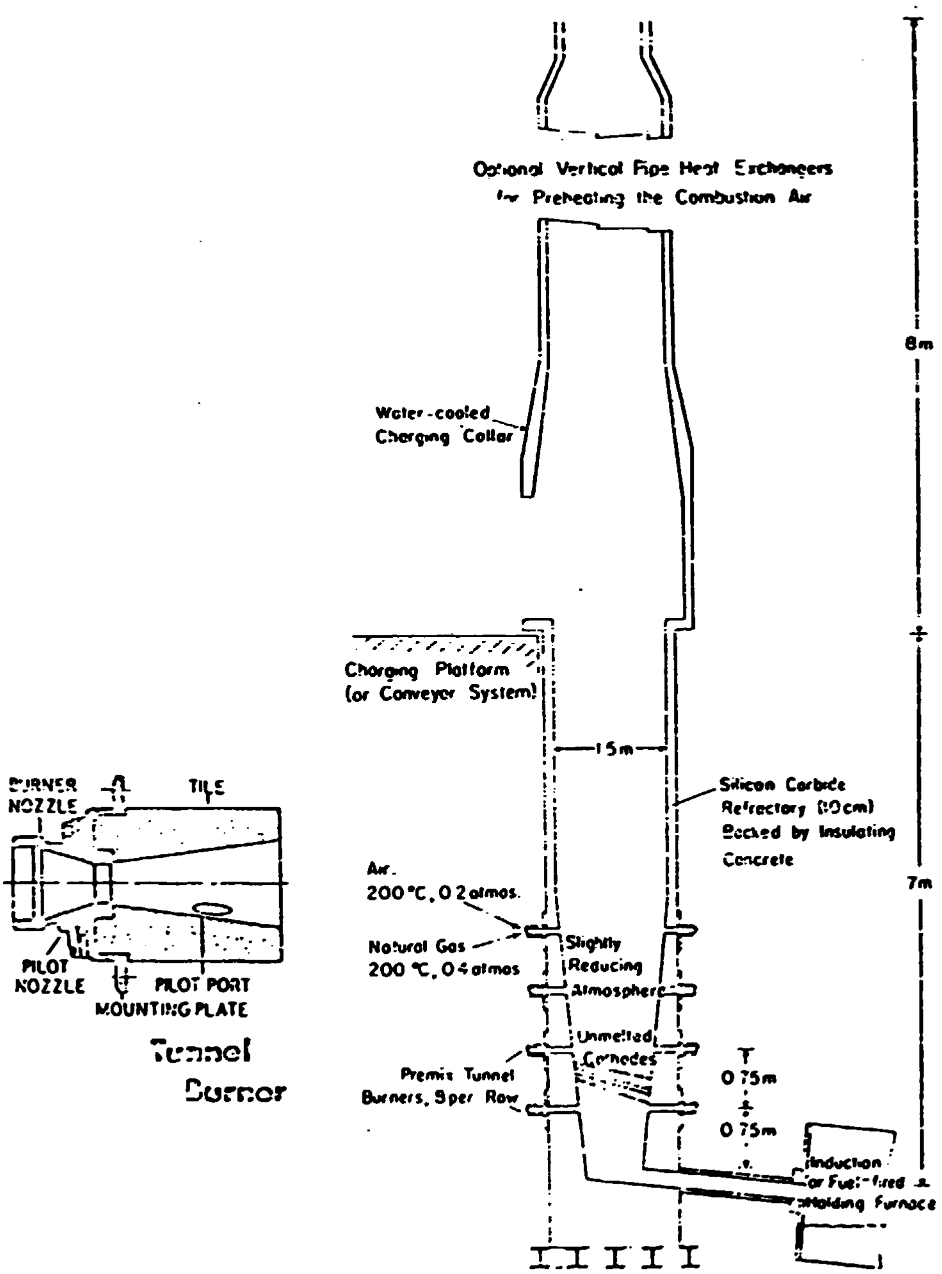


Figure 2. Diagram of ASARCO shaft furnace and tunnel burner

will be in the proper orientation for subsequent processing. Pneumatically operated presser rolls are lowered on to the bar as it passes over the extractor conveyor and pinch rolls, which maintain tension in the bar and to guide the lead end into an in-line shear, are positioned at the end of the extractor conveyor. The in-line shear squares the lead end of the bar and, upon start-up, crops the cast bar into lengths suitable for remelting until the physical and metallurgical quality of the bar are suitable for rolling.

After passing a bar handling and loop control table, to sense and correct variations in casting and rolling speeds, the bar enters a preparation device consisting of four scribing knives and four wire brushes, and then a pinch roll unit which positions and feeds it to the first stand of the rolling mill. During its passage through the mill the rod is protected from oxidation through a soluble oil protective atmosphere which also cools it, though the solution is kept at an elevated temperature to prevent overcooling.

On leaving the mill, the rod is subjected to a 3-phase in-line pickling process, which occurs as it travels within a compartmentalised delivery tube. In phase one, a mild acid solution is circulated through the tube to remove light oxide on the rod as it leaves the last finishing stand, while phase two consists of water rinsing, cooling the rod and removing any acid residue, with pressurised spraying boxes and air wipers. The third phase prevents surface oxidation by a water-soluble wax coating applied automatically. Hydraulically-driven pinch rolls then direct the rod to the coiler, with the diameter of each coil being determined by the rotational speed of the coiling head.

C) OUTOKUMPU CASTING PROCESS

In 1969, an upwards casting system was developed and a copper caster incorporating this system was brought on stream in 1970.

In this casting process, cathodes are melted in an induction furnace and the molten copper is charged into a holding furnace through a launder. From this furnace metal is drawn continuously through a vertical die cooler. A cooled graphite mount-piece is positioned into the melt and the upper end of the die is encircled by a water-cooled copper jacket. Cast strands are cooled as they are drawn upwards and are coiled into 2-3 tonne coils by coilers located at the end of each strand. The submerged die withdrawal unit is located above the holding furnace and the die coolers are fitted to a horizontal steel support. Two pairs of pinch rollers for each strand pull the cast rods through the coolers. The diameter of a cast strand can vary from 8-25mm depending on the required dimensions of the finished product.

After the casting operation, the unpickled rod is bright, unoxidised, and ready for cold working. Cold working is undertaken on a specially-designed tandem line where several rollers are installed behind one another. Since the rod reductions are small, the wire is worked gently in order to prevent rolling defects, which are prevalent in hot rolling. Cast rod can also be further processed by means of drawing. The rod diameter determines whether cold rolling or drawing is most suitable.

An important characteristic of this casting process is that speed of rod withdrawal is limited. Consequently, to maintain economic output levels, a caster has a number of strands working simultaneously.

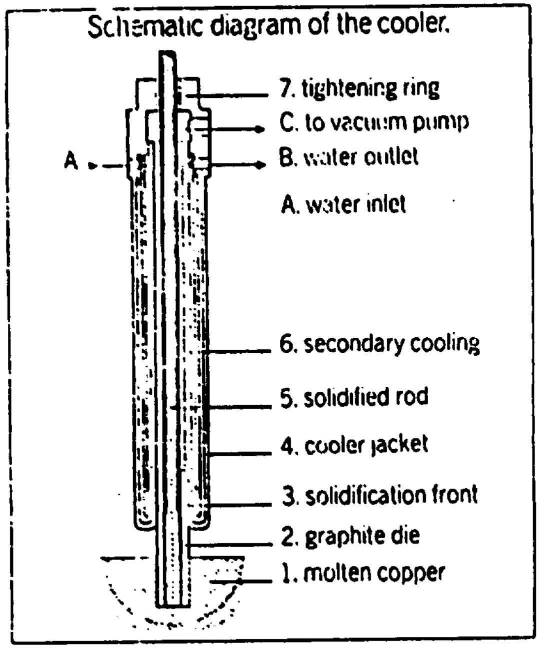
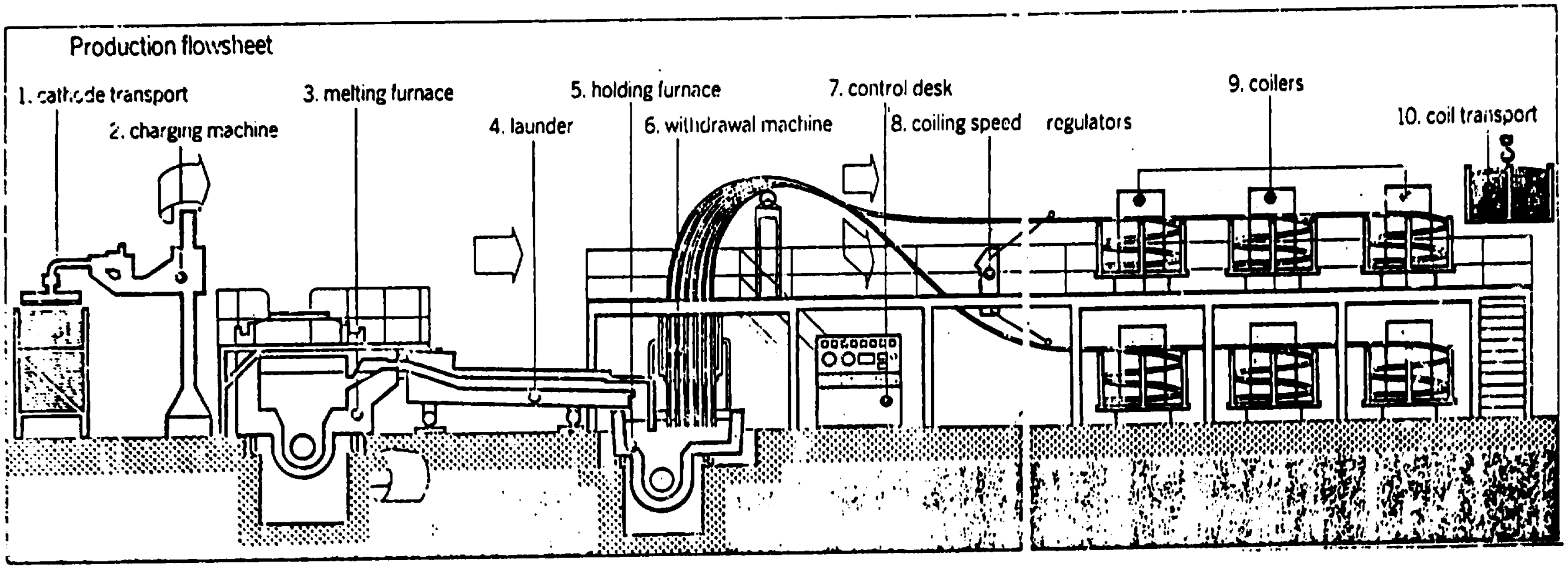


Figure 3. OUTOKUMPU upcast system



A pricipal advantage of the process is that complex equipment is not required as the process is very simple. This makes it possible to build and operate economically very small plants. It is also possible to add strands to a caster to increase capacity.

The casting line is completely automated and a 12-strand caster, for example, producing 2 tph can be controlled by one operator. Plant capacity is dependent on the number of strands, but as an approximate guide, each strand of a caster can produce 1,000 tpy. At present, there are some 20 Outokumpu casters in operation worldwide with capacities ranging from 3,000 to 30,000 tpy.

Outokumpu has production-scale experience in : oxygen-free copper (HCOF, OLP, and OHP copper), brasses, nickel-silvers, bronzes, copper-nickel alloys, precious metals and alloys, zinc, cadmium. The shapes and dimensions of the up-cast products can be varied over a wide range; not only rods (wires) but also tubes, strips, and various special profiles can be cast. The use of multi-strand machines makes the casting of small cross-sections economical. The smallest dimensions cast are : wire 3 mm diameter, strip 4 mm thick, tube of 20mm OD, 1.5mm wall thickness.

The production capacity of a plant depends on the dimensions of the product, the number of strands, and the rate of withdrawal, the latter generally being dictated by the nature of the metal.

Different types of furnaces can be used. Normal channel-type induction furnaces are used for oxygen-free coppers and certain alloys. Coreless induction furnaces are used for precious metals and some difficult alloys. Resistance and gas-heated furnaces can also be used if required.

D) SECOR CASTING PROCESS

Clecim, the plantmaking subsidiary of Creusot Loire, began development work (as Secim) on a copper rod casting process in 1975-76. A 10-tph prototype caster was built in that year and following a testing period of 18 months an order was placed by Australia's Copper Refineries of Townsville, Queensland, for a 25-tph rod caster. This caster, which incorporates the patented grooveless rolling procedure (RER), developed by Copper Refineries, was brought on stream in late 1977. The trademark for the Secim rod casting process is Secor.

More recently Cosim, Clecim's Spanish subsidiary, has built a Secim rod caster at the Oviedo plant of SIA Santa Barbara. This Secor line has a capacity of 10.5 tph and was fully operational at the end of 1982, two months after the start of the commissioning period.

Secor process description

In both the Secor plants which have been built so far, molten copper is supplied from an Asarco shaft furnace. It then passes through a launder into a holding furnace, and then into a pouring furnace, from which it flows through a spout into the caster. Slag traps are incorporated in the two launders. The pouring furnace can be controlled either manually or automatically to feed the caster at a constant rate. Automatic pouring is achieved by means of a "bubble" tube which is immersed in the tundish of the caster and fed continuously with nitrogen or argon.

The caster is of the wheel-and-belt type and top-pouring is practised. After cooling in the mould, the cast bar is passed from the casting wheel to an exit table which feeds the rolling mill. To avoid problems in the event of re-start in casting following any breakdown, the cast bar does not pass above the casting wheel.

Pinch rolls are on the exit table, which are synchronised with the casting wheel speed, take the weight of the bar to prevent bar tension at the wheel exit. The pinch rolls convey the bar to the edge trimming unit and entry shear. The trimming unit removes any fins on the belt side of the bar and chamfers bar corners. The entry shear crops the bar at start-up until the cast bar is suitable for rolling. The shear also disposes of rod in the event of a cobble. Cropped bar sections are cooled on the crop bar conveyor and stacked for remelting.

Prior to rolling, the bar passes through a pre-pickling and scale-breaking chamber. The bar is rolled using grooves to form a round then grooveless rolls (Copper Refineries' RER system) to form a flat before the final round pass. The non-twist rolling mill has cantilever stands grouped in blocks of 2-3-4 or 5 stands. An intermediate shear is installed on the mill to reduce downtime should a cobble occur. Cooling in the mill is achieved by circulating coolant from a central station. After rolling, the rod passes through a cooling tube where it is cleaned with non-acid solution. The rod is drawn through the cooling tube by exit pinch rolls and fed through a waxing unit to the coiler. The wax prevents tarnishing of the clean finished rod. A guillotine shear is positioned in front of the coiler to cut the rod, if necessary, to avoid coiler jamming.

The Secor casting process offers seven principal advantages. The automatic level control system ensures consistent high-quality cast bar as well as labour savings and improved operator safety. Increased operator safety and labour savings are achieved also by the automated cropped bar cooling and removal system. Mill stands are driven separately to ensure reliable production of a wide range of rod diameters.

The RER grooveless non-twist rolling process permits improved interchangeability of rolls, reduced operating costs and capital cost, and increased productivity. The life of the tungsten carbide rolls is prolonged by 50 % by use of the organic solution cleaning process. Fine wire production down to 71 microns can be achieved without rod shaving. Finally, the low height of the caster reduces building installation costs.

E) CONTIROD (KRUPP-HAZELETT)

In the 1960s Belgium's Metallurgie Hoboken Overpelt (MHO) and Usine á Cuivre et a Zinc de Liege were involved in the joint development of a copper rod casting process. The companies planned to install a rod casting plant at the Olen refinery of MHO. Development work was undertaken on a Hazelett twin-belt casting plant at Olen. In 1970 it was decided to build a continuous casting and rolling plant next to the Olen electrolytic copper refinery. Production of copper rod using the MHO-Usine á Cuivre et a Zinc process began in 1973. The plant included Krupp-Hazelett casting and rolling equipment and was designed for a capacity of 100,000 tpy.
The trademark "Contirod" was patented by MHO for the cast copper rod products.

In this process, cathodes are melted in a shaft furnace and the molten copper is passed to an induction-heated holding furnace. From this furnace, copper is charged into a twin-belt casting unit. Rectangular bars are produced by the caster. A pair of pinch rolls guides the cast bar to a scalping unit which trims the edges of the bar by means of rotating tools. The pinch rolls act as a speed-sensing device for casting and rolling. A pendulum shear is located near the scalping unit to crop bar during start-up and casting breakdowns.

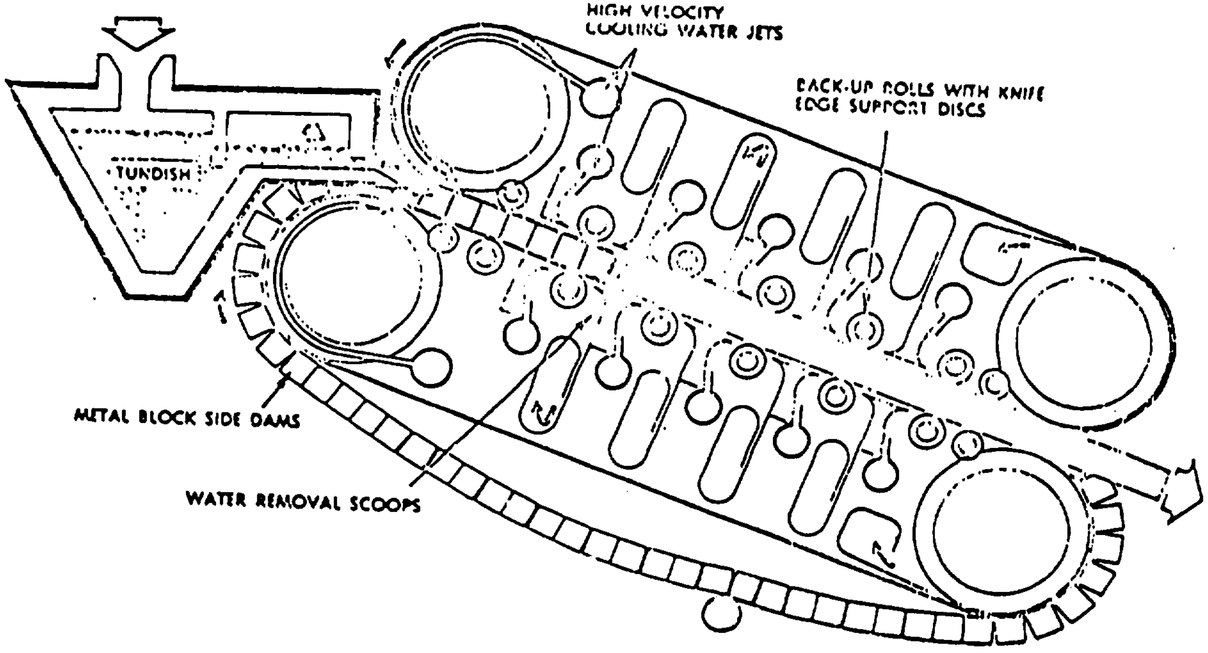


Figure 4. HAZELETT twin belt casting machine

Following scalping, the bar passes to a mill train consisting of 15 horizontal and vertical passes. The mill train is divided into four groups: roughing mill; intermediate mill; finishing mill with rotary shear; loop control unit and emergency shear.
The loop controls are designed to eliminate tension between passes to ensure production of high-quality rod.
After pickling in sulphuric acid solution, washing and soaping, rod is coiled to produce 5-tonne coils. Currently rod ranging from 6.35 to 22.5 mm is being produced on Krupp-Hazelett casting units with capacities ranging from 12 to 50 tph.

F) GENERAL ELECTRIC DIP-FORMING

In this process a cold, clean copper "seed" rod with a diameter of about 9.6mm is pushed upward through a graphite container filled with molten copper. The rod moves at 100 metres/minute and the copper depth is about 500 cm.
On emerging the rod is some 2.75 times its initial weight, with a diameter of about 16mm. When molten copper and seed rod are fed in continuously, a unit of this size will have an output of some 10 tph of hot rod, and by scaling up the diameter of the rod and the depth of copper, higher output rates are achieved.

The hot rod is cooled to 850 ^{o}C and hot rolled in a protective atmosphere then cooled to room temperature while still in protective atmosphere, and coiled.
Dip formed copper rod has a low oxygen content, clean dense surface, uniform single-phase internal structure and mechanical properties suitable for such applications as fine wire drawing.

At the same time, it has to be mentioned that this rod on the one hand is free from hydrogen embrittlement pheno-

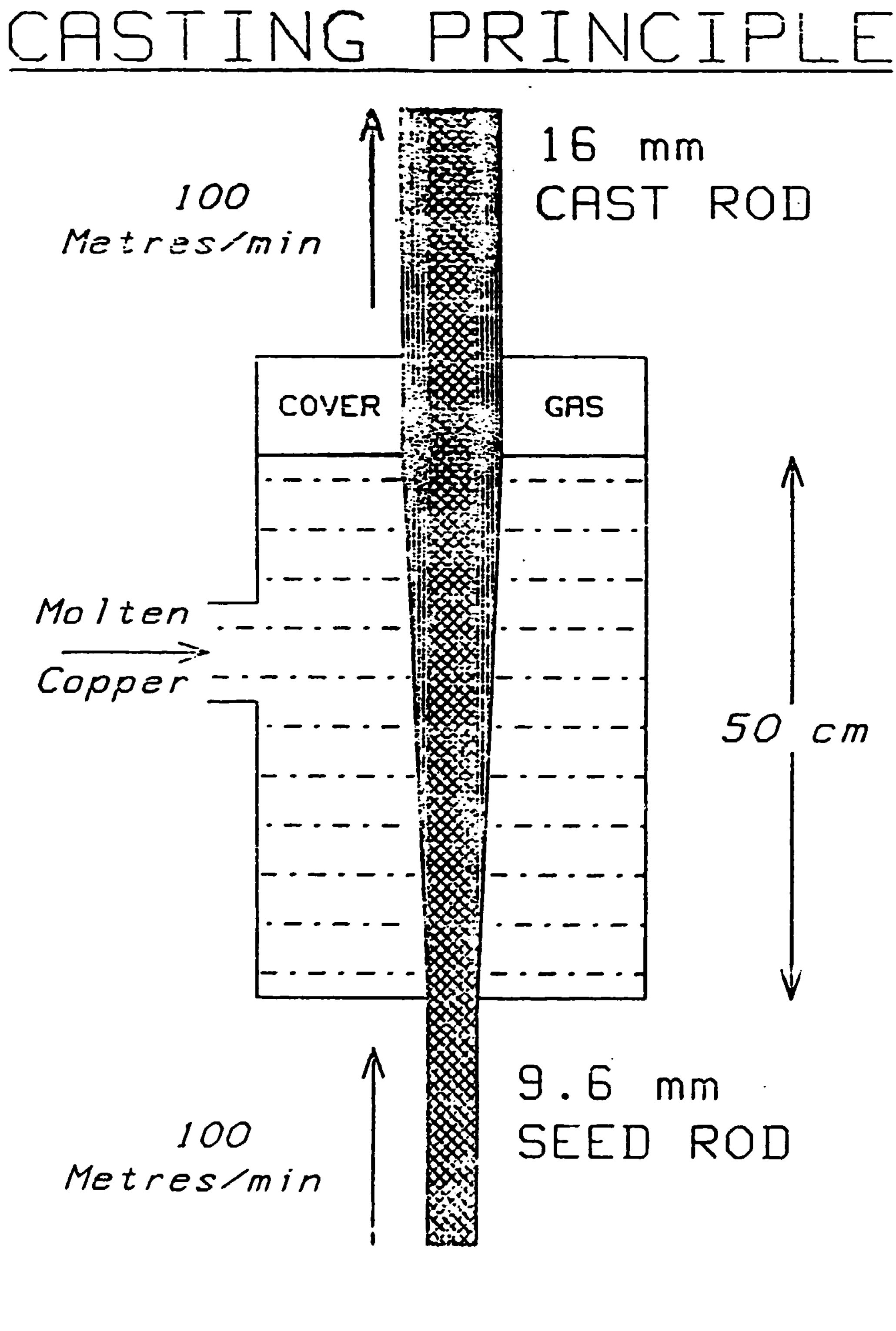


Figure 5. GENERAL ELECTRIC dip-forming process

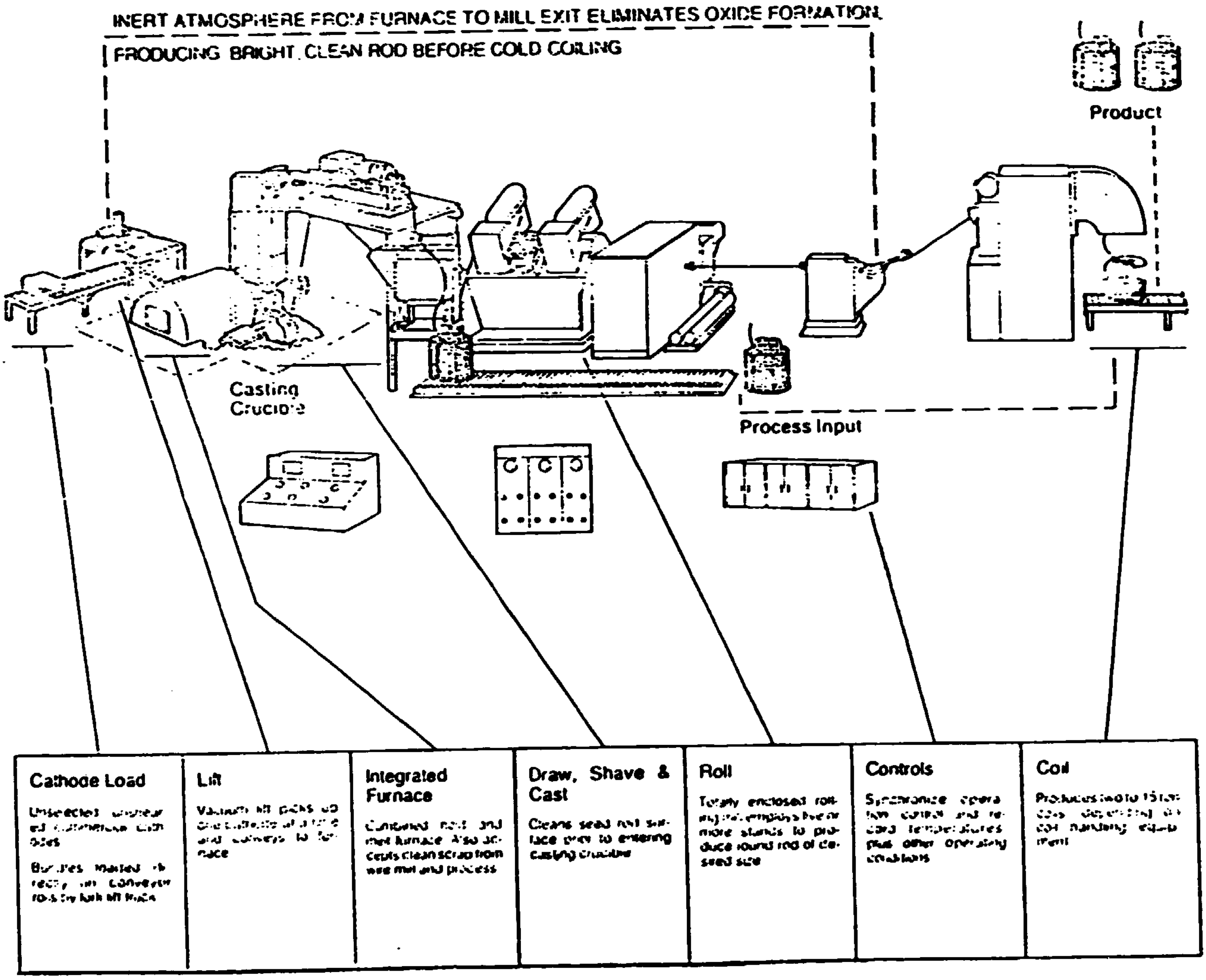


Figure 6. GENERAL ELECTRIC dip-forming process

menon, but on the other hand it has somewhat higher recrystallization temperature, which is always inherent to the oxygen-free copper. The dip formed rod is very much suitable even for special applications, where requirements are high.

G) GENERAL ELECTRIC LEVITATION CASTING (GELEC) PROCESS

The General Electric Levitation Casting (GELEC) process is basically a synergistic combination of an electromagnetic levitation field and a highly effective heat exchanger used in an upward casting mode. The result is a simple, low cost continuous casting process that overcomes problems of friction and adhesion at the mold-metal interface often found using other casting techniques. Among the advantages are high casting speeds, smooth continuous withdrawal of the cast product; excellent homogenity, grain structure and dimensional uniformity of the cast product; absence of imperfections or inclusions in the cast surface and extended operating life of parts in contact with molten metal.
It is particularly well suited for near or net shape casting of small diameter rods and other products from a variety of pure metals and alloys. For most applications, the fine equi-axed grain structure of the "as-cast" product is suitable for immediate drawing or forming operations without the need for hot rolling, annealing, or other processing after casting.

H) CONCLUDING REMARKS

It follows from the above that hitherto traditional wire bar route for the manufacture of re-draw copper rod will, increasingly in the future, only be retained in exceptional circumstances. Thus, for rod outputs in the range of 6-40 tonnes/hr, the Properzi, Southwire and Contirod continuous melting, casting and rolling processes are now firmly established.

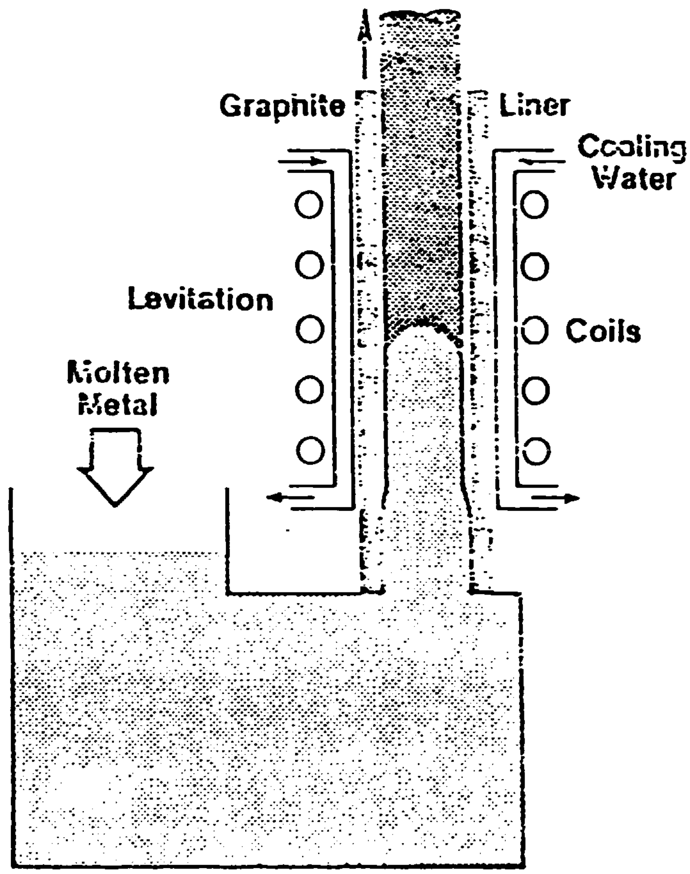


Figure 7. Schematic representation of GE Levitation Casting (GELEC) apparatus

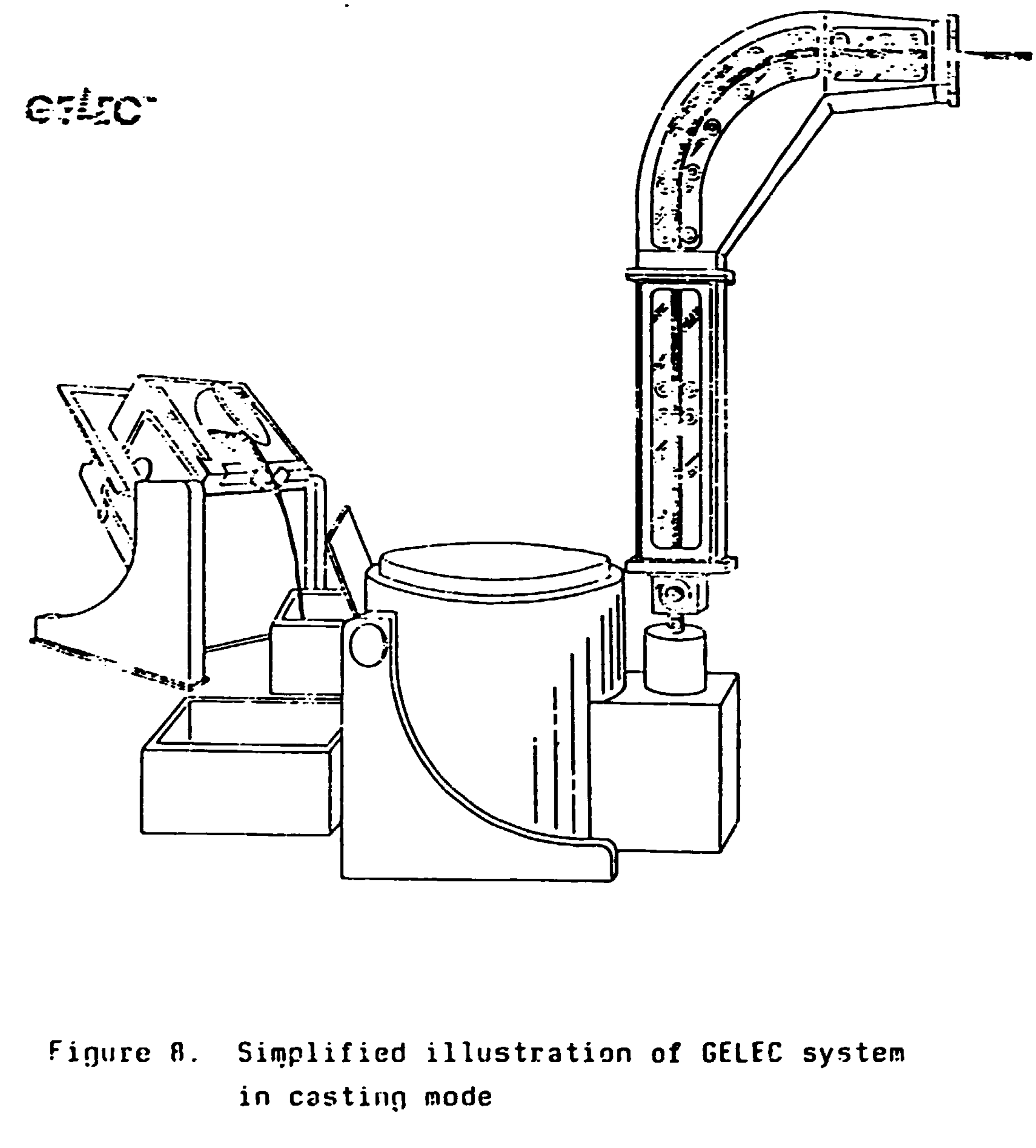


Figure 8. Simplified illustration of GELEC system in casting mode

A similar situation exists for the GE dip-forming process for outputs in the range of 3.0-11.0 tonnes/hr. Adoption of these processes has been mainly on economic considerations, including those arising from the use of a less expensive in-feed, namely cathode, and the omission of a bar reheating stage. For example, with the Southwire SCR system, the overall saving in energy consumption is estimated as about 1340 MJ/tonne, with an output of about 40 tonnes/hr. The continuous processes also give important advantages in "downstream" operations, among which are the absence of a pickling stage and the greatly reduced frequency of butt welding between coils. The latter allows the use of higher drawing speeds for the rod. Additionally, with appropriate operational techniques, the technical properties of continuously processed rod have proved at least as good and, frequently, markedly superior to those of rod made from wire bar.

However, it should be recognised that the equipment involved in the continuous processes is relatively complicated and its performance depends on the efficiency of a range of control and automatic devices not previously employed in the industry. This part of the technology is therefore still evolving, as is also that dealing with the material design and use of various components for which long-term durability is necessary under the arduous conditions associated with the process. For the user, the equipment represents a major capital investment and its utilisation factor is therefore important. On this point, claims have been made that certain processes are better than the others, but they are difficult to relate in terms of a given product quality.

For lower outputs than those of the processes with a tandemised hot rolling stage, continuous casting methods exists (Outokumpu), or are being developed (GELEC), which produce rod suitable for separate cold rolling to re-draw sizes. They provide increased flexibility with respect to the range of section shape and materials that can be cast as well as to consistency of product quality from the continuous nature of the operation. As far as can be judged, all these casting processes use a submerged graphite die technique. The macro-structure of the as-cast rod usually consists of large columnar grains. While this does not appear to cause any problems with rod ultimately reduced to wire sizes, concern has been expressed about the risk of fire cracking with rod subjected to more modest amounts of deformation before annealing.

II. MANUFACTURE OF COPPER AND COPPER ALLOY SHEET AND STRIP

A) CASTING

The manufacture of sheet and strip in the modern copper and brass mill begins with one of two basic casting operations. In the casting plant the metal is melted and either cast in the form of slabs which are subsequently heated and hot-rolled to coils of heavy gauge strip, or directly cast in strip form and coiled. The coils, in either case, will have their surfaces milled to remove any defects from casting or hot rolling. The next set of operations through which they are brought will provide the desired final gauge and temper by a series of cold rolling, annealing, and cleaning operations. Finally, they may be slit into narrower widths, leveled, edge rolled or otherwise treated, and packeged for shipment.

Raw materials from which the melt is prepared consist primarily of virgin copper, either electrolytic or fire-refined, selected clean scrap of known origin, carefully checked for composition, and special alloy elements such as virgin zinc, lead, tin or nickel. After the charge has been formulated, the raw materials are assembled in charge buckets and carefully weighed. These materials are discharged into hoppers which feed electric-induction melting furnaces. As melting of the charge proceeds, samples are taken from the furnace and sent to the spectographic laboratory for analysis. The composition is calculated by computer and returned to the printer on the melt shop floor within minutes. If necessary, the melter can then make additions to bring the melt exactly within the specified composition range.

The metal is protected from atmospheric oxidation by a cover of carbon or bone ash. When the composition and temperature have been determined to meet the requirements of the alloy being melted, the molten metal is transferred to a holding furnace.

For many years in copper and brass mill casting practice the molten metal was poured from the melting furnace into a pouring box which distributed it into long, rectangular molds. This method had some important disadvantages. The maximum weight of a casting, and therefore the length of finished coil was limited. Molten metal dropping from a pouring box to the base of the mold was subject to oxide scumming and entrapment. Metal splash caused the bottom end of the bar to be spongy. The casting varied from bottom to top in temperature and solidification rate with potential problems from shrink cavities, gas entrapment, surface laps, and mold coating defects.

During the 1960-1975 period, semicontinuous and continuous casting processes began to supplant the book molds. In each of these new methods, molten metal flows into a short, rectangular, water-cooled mold, which initially is closed at one end by a plug on a movable ram or a starter bar. The metal freezes to the plug and forms a shell against the mold surface. The ram is then steadily withdrawn, pulling the shell with it. As the shell exits from the bottom of the mold, cold water is sprayed on it, cooling it rapidly and causing the contained molten metal to freeze. In this manner a continuously cast slab of the desired length is produced.

Gases and non-metallic materials float to the surface of the shallow pool and remains there to collect at the top of the semicontinuous cast slab. This end has to be sawed off before hot rolling. The continuous cast slabs are free from this type of disadvantages.

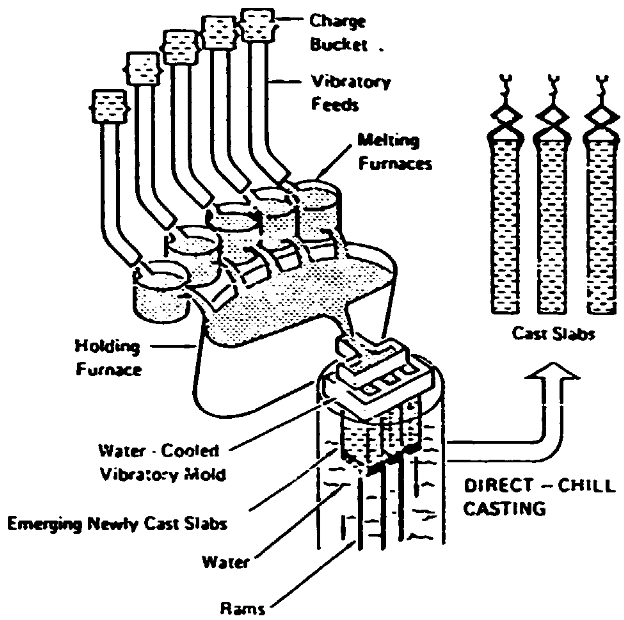


Figure 9/a. Schematic sketch of vertical direct chill (DC) casting of slabs

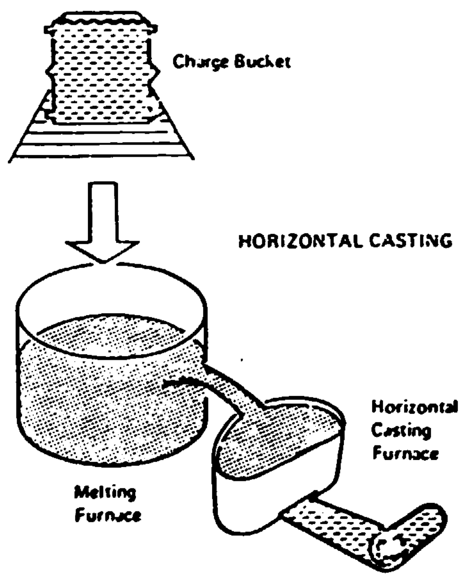


Figure 9/b. Schematic sketch of horizontal continuous casting and coiling of strip

The direct chill (DC) ting processes described above are done in vertical molds and are used to produce slabs of large cross section which are subsequently reheated, hot-rolled into heavy gauge strip, and coiled. Such large coils are the most economical to handle through the subsequent rolling and annealing processes at the mill and later by the user who is fabricating finished parts.

Some alloys contain elements which produce phases, or structures, which are difficult or even impossible to hot-roll. Such alloys must be cold-rolled, and the amount of reduction in thickness that can be achieved, before annealing becomes necessary, is small when compared to hot-rolling reductions.

The problem with alloys that are hard to hot-work is overcome with the horizontal continuous-casting method. It offers a means of producing relatively thin cast strips in long lengths which can be coiled in the cast state and later reduced by cold rolling. Tedious, costly cold breakdown rolling and the attendant annealing are avoided.

The horizontal continuous casting process provides a product of excellent quality. Typically, one low-frequency electric-induction furnace is used as a melter. As a charge of selected scrap and refined metal additions are melted and brought to the pouring temperature, samples for chemical analysis are taken. When the proper analysis is established and the pouring temperature attained, part of the metal is poured into a second, smaller electric-induction holding furnace. This furnace is constantly monitored to maintain the metal at the desired casting temperature. The casting mold is attached to the lower front of this furnace. It is graphite mold contained in a copper, water-cooled jacket.

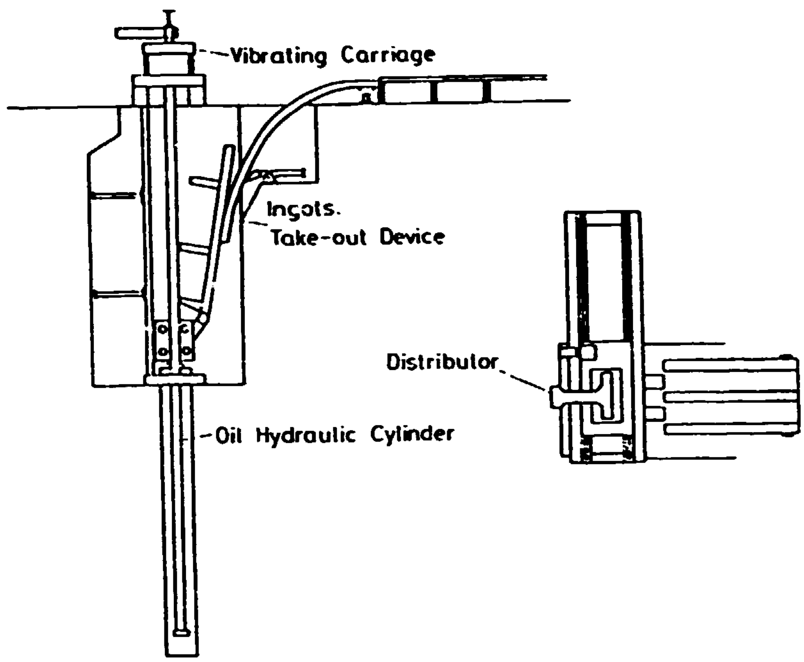


Figure 10/a. Semi-continuous casting machine

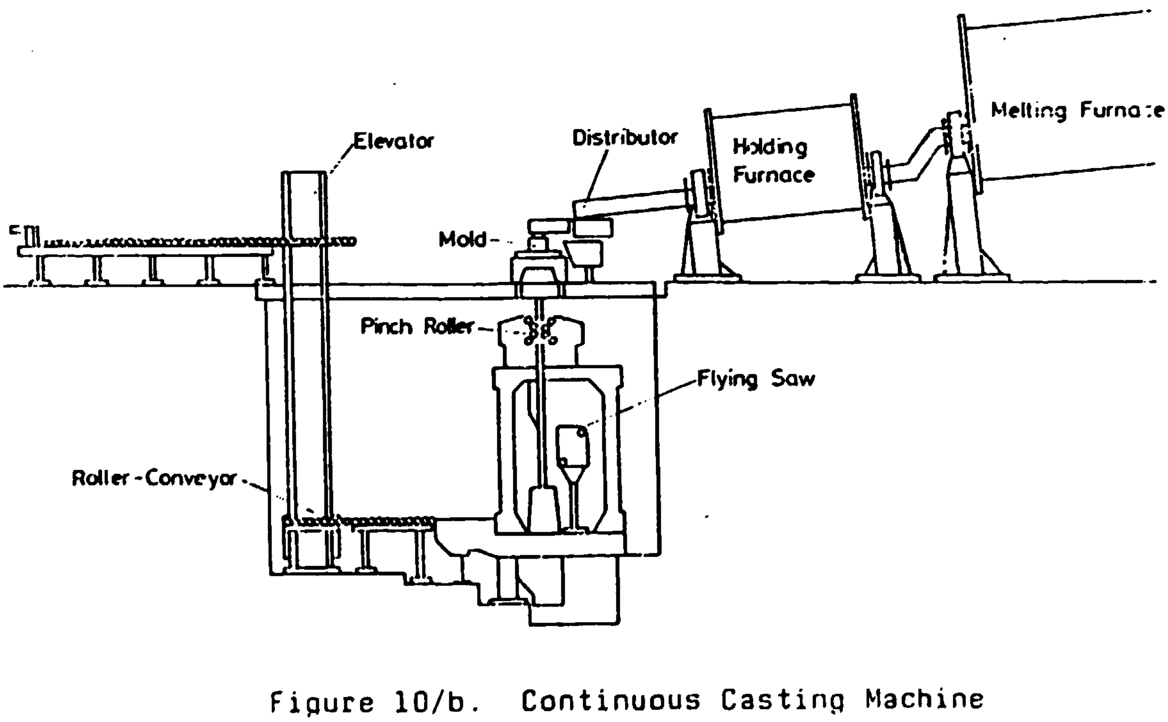


Figure 10/b. Continuous Casting Machine

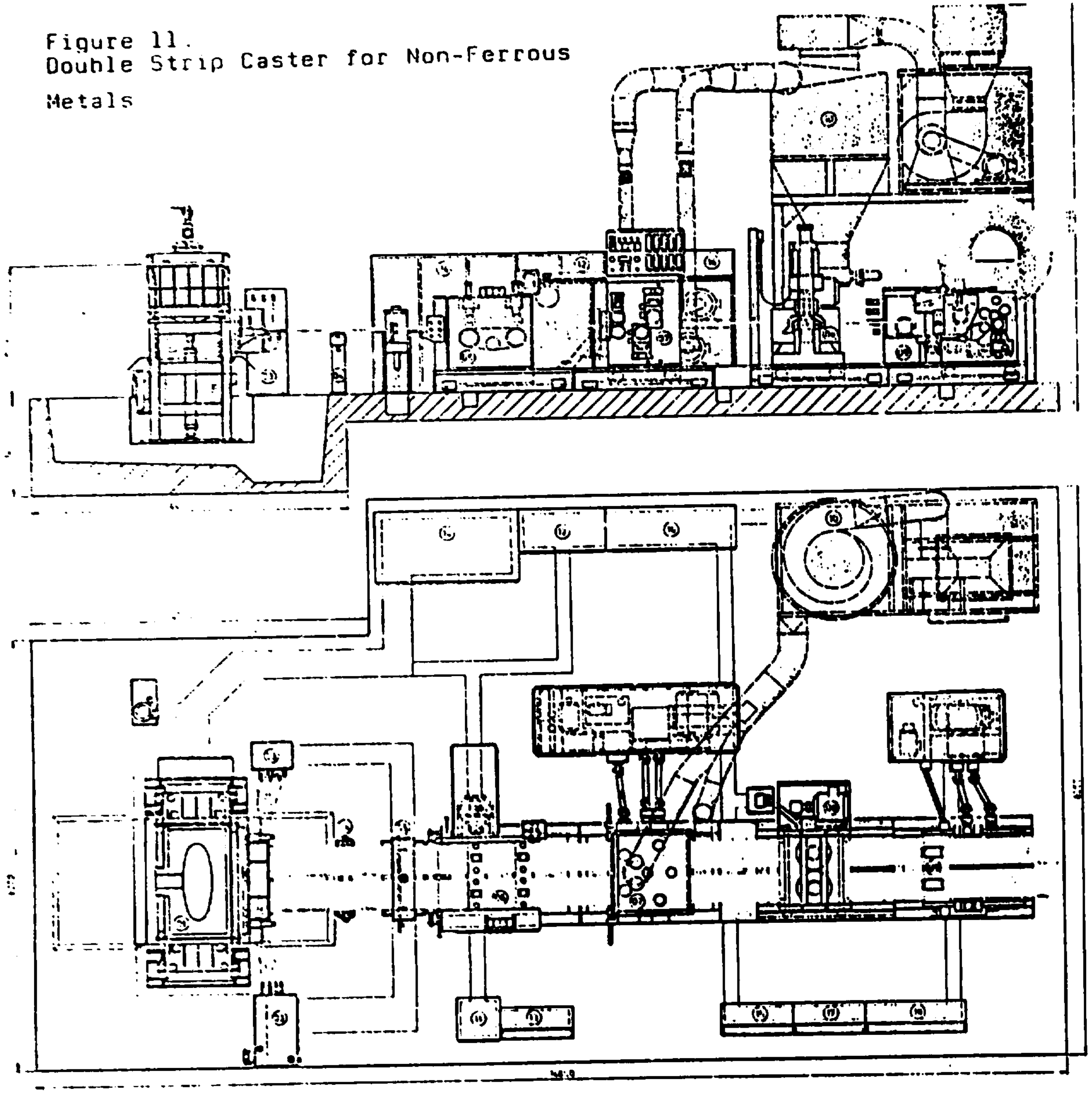
A silicon carbide plate in the front of the furnace contains a slot which opens into the mold. At the beginning of a cast, a starter bar is inserted into the mold and the metal freezes to it. The mold is only a few inches long.

Two stands of withdrawal rolls slowly withdraw the starter bar as the metal freezes in the mold cavity. The cast bar, frozen to the starter bar, is continuously withdrawn as the metal freezes in the mold. Although it is simple process, its practice requires that narrow tolerances on mold dimensions be held, and exceptional melt cleanness be maintained. Any dross or foreign materials that enter the graphite mold will quickly destroy it. Mold sizes range from 200 to over 660 mm in width and from about 12 to 20 mm in thickness. A saw in the withdrawal line cuts the bars off at the desired length, and they are coiled in preparation for subsequent processing. A sample for chemical analysis is cut from each bar end, so the composition at each end of each coiled bar is determined. This process lends itself to in-line coil milling and to maximum coil lengths, dependent only on handling equipment capacity and practical processing of the material itself.

The rapid chilling of the small amount of metal in the horizontal mold produces a fine, equiaxed cast grain structure. The metal drawn from the furnace as it solidifies always has a pool of molten liquid above it where gases and nonmetallic impurities tend to collect. The cast bar is free of porosity and of defects caused by solid inclusions.

The good quality of the horizontal casting shows up in the finished strip in terms of excellent formability. Phosphor bronzes cast this way develop high strength for which they are specified, coupled with the good formability needed in most of their applications.

Figure 11.
Double Strip Caster for Non-Ferrous Metals



01 Holding furnace with inductor
02 Cooler
03 Cooling water distribution
04 Roller support
05 Secondary cooling
06 Withdrawal device
07 Milling machine
08 Strip shear
09 Strip coiler
10 Chip removal for milling machine
11 Control cabinet for withdrawal device
12 Power cabinet for withdrawal device
13 Control desk for inductor
14 Power cabinet for inductor
15 Control desk for milling machine
16 Power cabinet for milling machine
17 Control desk for strip shear
18 Control desk for strip coiler

Leaded bearing-bronze, also cast by this process, offers improved quality for bushings, bearings, and thrust washers which must carry heavy loads under dynamic stresses without failure. Some smaller mills depend almost entirely on horizontal casting, regardless of alloy, because the process is readily adaptable to the casting of small quantities of several alloys.

CONCLUDING REMARKS

1) The mold casting method is not used in up-to-date mills any more in view of the small weight of the finished coils, inferior surface quality, shrinkage cavities, porosity, etc.

2) Semicontinuous cast slabs have excellent quality, but one end of the slabs has to be sawed off.

3) Vertical continuous casting process is free from the above mentioned disadvantages, it has a very high production capacity, but requires the highest capital investments of all the reviewed technologies.

4) Horizontal continuous casting characterized with low investment cost equipment, good quality, flexible capacity range by using several smaller capacity machines, which also enable casting of different alloys at the same time.
   It offers the possibility to eliminate hot rolling and to reduce costs of cold rolling of some "difficult" alloys, so this method requires the smallest capital expenditure for the mill as a whole.
   Cuts the energy, material and transport costs, requires few but well-trained personnel.
   This process, however, is not suitable for the production of very wide strips.

B) ROLLING

To ready the direct chill cast slab for hot rolling, the top or gate end is trimmed by sawing, and then it is conveyed into a furnace for heating. Slabs or bars of the same alloy are grouped together in a lot and processed through the furnace and the hot mill.

The roll stand used for hot rolling is a very sturdy mill having two rolls (two-high) whose direction of rotation can be rapidly reversed so the strip can be passed back and forth between them. The large horizontal rolls which reduce the thickness are supplemented by a pair of vertical edging rolls.
After the final rolling pass the metal is spray cooled and coiled.

The modern hot mill is operated from an air-conditioned pulpit overlooking the rolling stand and the conveyor run-out table. With the aid of television cameras, placed at strategic points focusing on the rolls, the furnace and the transfer buggy, the alligator shears at each end, and the coiler, the operator can control all these from his vantage point.
A schedule of rolling reduction for each pass through the rolls is established and recorded on a punched card. Operation of the hot mill is sequenced by computerized controls to insure uniform processing through the hot mill.

Following hot rolling the DC cast bars are coil milled, and after careful surface inspection are ready to be applied on orders for processing to final gauge, temper and width. Horizontally continuous-cast bars are milled in-line.

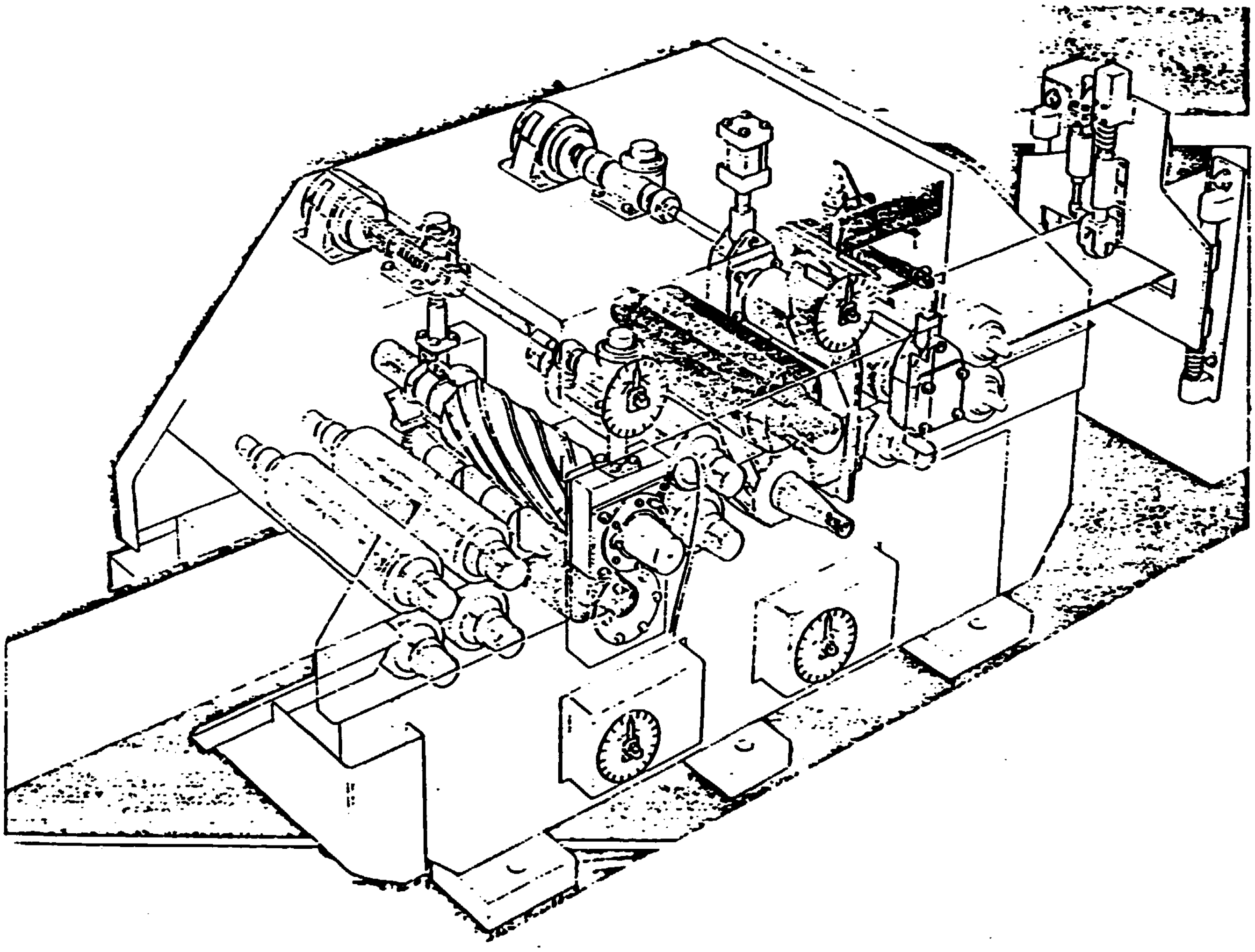


Figure 12. Slab and strip milling machine

Cold rolling of coppers and copper alloys into sheet and strip of excellent quality requires a combination of skillful workmanship, knowledge, and good rolling mills.
To keep cost as low as possible and competitive, the reduction in thickness to final gauge needs to be accomplished in the fewest operations compatible with quality requirements.

Continuous cast strips are usually rolled on reversible two-high breakdown mill with large roll diameter. On the mills heavy reductions are performed in every pass.
From 20mm to 6mm thickness the metal is rolled without tension between two upcoilers. Under 6mm winders and tension are applied.
In some cases combined two-high/four-high mills or a combination of one-way/reversible operation is used for roughing, intermediate and finish rolling; achieving savings in investment costs.

Small diameter work rolls are most desirable for providing maximum utilization of roll force in reducing metal to thinner gauges, but they lack the stiffness required. The wider the metal to be rolled, the longer the rolls, and the greater the tendency for the rolls to bend or spring.
To overcome the tendency four-high and cluster rolling mills are used for cold rolling in the brass mill.

Four-high rolling mills contain a pair of work rolls of relatively small diameter. A second pair of rolls, of large diameter, is placed above and below the work rolls in the stand to back them up and prevent from springing. This arrangement allows the advantage of the small contact area of small work rolls and the transmittal of high force through the large back-up rolls while maintaining the rigidity required for gauge control.

The minimum size of the work rolls is limited by the forces in rolling, which tend to bow them backward or forward during rolling.

For the very high capacity rolling of heavy gauge coils tandem mills consisting of 2-4 four-high mills are operated in some plants.

Cluster rolling mills, for example, Sendzimir 20-high mills were designed to counteract both the vertical and horizontal elements of the rolling forces and thus enable the use of minimum diameter work rolls. In cluster mills the work rolls are backed up by a cluster of rolls placed with respect to the work rolls so they contain the rolling forces and prevent bending or springing of the work rolls.

The traditional 20-high Sendzimir mills are very complicated and expensive, so rolling mill manufacturers started to design and manufacture new type of rigid rolling mills.
One of the most commonly used solutions is to upgrade the existing four-high mills changing the mechanical screwdown system to a hydraulic one. This is a very cheap way of modernization giving optimal results.

Converting existing two or four-high mills into Z-high cold rolling mill with smaller work rolls enables to roll thinner gauges and tougher alloys with extremely tight tolerances. This mill combines the advantages of a four-high and a Sendzimir cluster mill, and can be used as a four-high mill, too.

In new four-high mills hydraulic screwdown system is commonly used and the rigidity is increased by application of prestressed mill housing frames. Good examples for that are the genuine Fröhling mills. To improve cross-sectional shape of strip hydraulic roll bending devices are lately spreading all over the world.

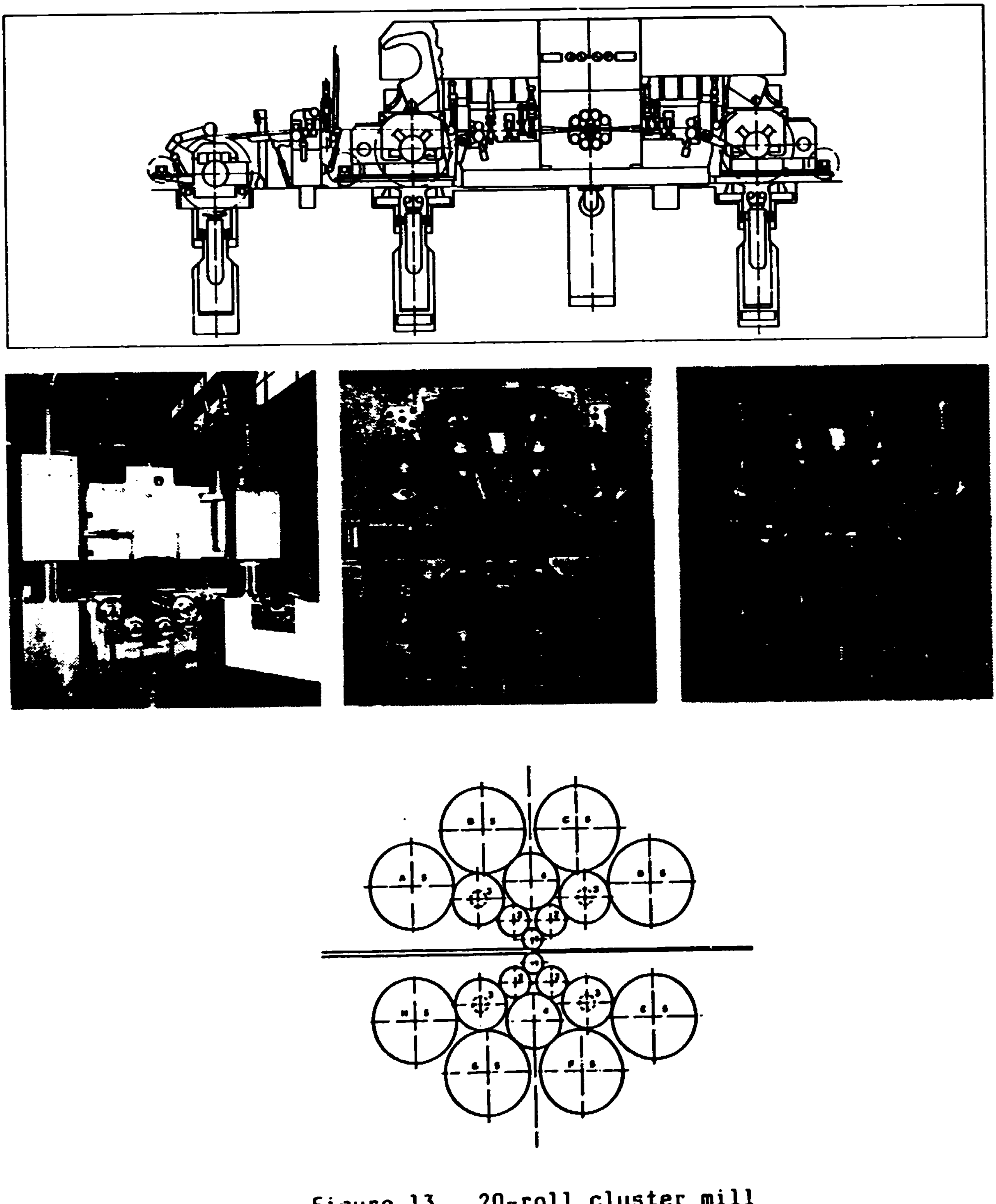


Figure 13. 20-roll cluster mill

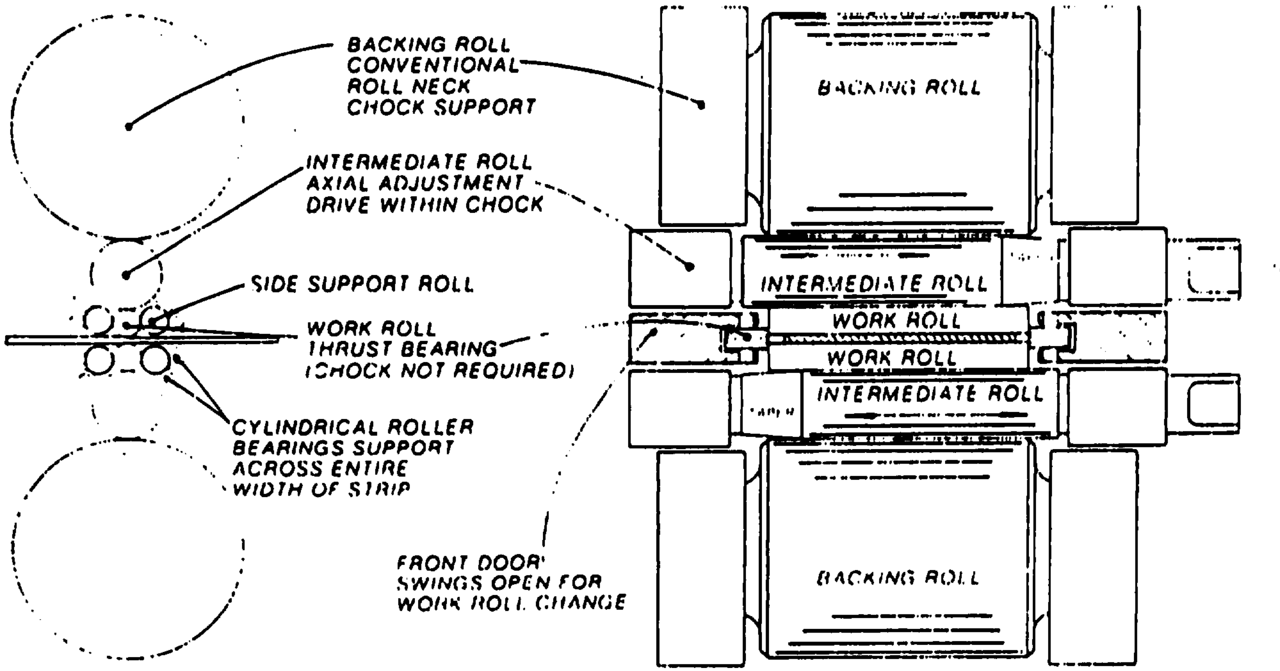


Figure 14.. SENDZIMIR Z-high cold rolling mill

For thickness control during high speed rolling, continuous measurement of this dimension is a necessity. Rolling mills are equipped with X-ray or contact instruments, which continuously gauge the metal and provide a continuous readout of thickness. There are also control devices which actuate the screws in the roll housings and automatically open or close the gap between the work rolls to adjust the thickness being produced as required. These gauges may also adjust back tension and forward tension applied by payoff and recoil arbors to effect changes in the thickness of the rolled metal.

In the last decade the Vollmer contact thickness and roll gap gauges are used most commonly on the non-ferrous cold rolling mills. These gauges are safe for the personnel, cheaper than the X-ray or beta-ray instruments, and require less and more simple maintenance.

C) ANNEALING

During cold rolling, hardening of the metal occurs. One reason for annealing is to soften the metal so it can be further reduced by cold working. In case of finished strip the anneal is designed to produce a specified tensile strength and chosen uniform grain size. There are two methods of annealing operations : coil annealing and strand annealing, both having advantages and disadvantages of their own.

Coil annealing may be carried out in a roller hearth furnace in which the coils are continuously conveyed slowly through the furnace as they are gradually heated to the annealing temperature. This type of furnace usually does not have a prepared atmosphere, but the products of combustion fill the furnace and reduce the metal oxidation rate. More commonly, coil annealing is done in bell furnaces of the type in which a controlled atmosphere can be maintained. The annealing unit consists of a base on which the coils are stacked. Under the base is a fan for circulating the hot gases through the load, to provide more uniform and rapid heating.

After the metal is stacked on the base, the inner hood or retort is placed over the load and sealed. The controlled atmosphere begins to flow through the hood purging the air. The furnace is placed over the hood and heating begun.

The heat input is constantly adjusted to maintain temperature uniformity in the load. This controlled temperature rise also allows roll lubricants to vaporize and be carried off before the metal gets so hot that surfaces can be harmed. After the metal has reached the annealing temperature, it is held there for a short period or soaked to provide maximum uniformity .

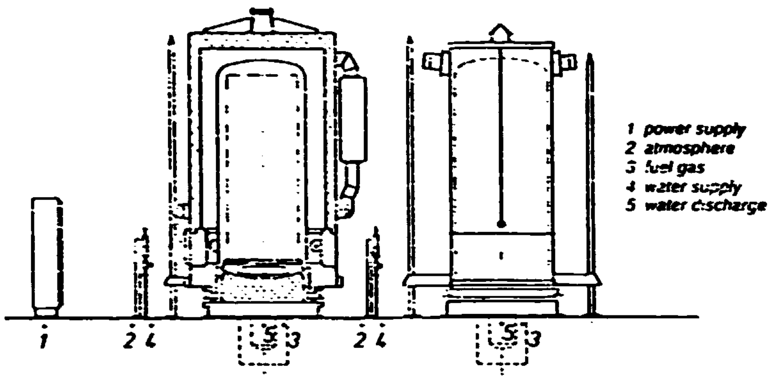


Figure 15/a. High-convection bell annealer : the basic setup

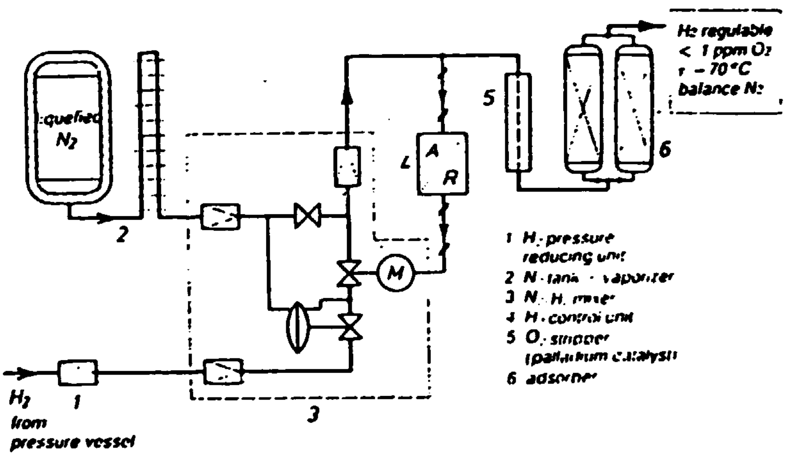


Figure 15/b. Nitrogen/Hydrogen mixer plus absorber

Then the furnace is turned off and removed, and the metal cools in the controlled atmosphere under the inner hood. Cooling may be aided by a cooling ccver containing a water spray system. The inner hood is not removed until the metal temperature is low enough that no discolouring or oxidation of the metal takes place.

The controlled atmosphere is produced in gas cracking units, or in N_2/H_2 mixer and absorber.

For oxygen containing copper, the atmosphere must be nearly free from hydrcgen and the annealing temperature low enough to avoid hydrogen embrittlement.

In traditional coil annealers, a thin oxide film forms on the surface of zinc containing alloys (brasses). The natural colour of the metal has to be restored by dilute sulphuric acid pickling and brushing following the anneal.

In the early 1970s, the Austrian Ebner Company developed a process for bright annealing of brasses in high-convection bell annealers. Since then about 100 annealers of this type are operating all around the world. In these furnaces a charge temperature of 750 ^{o}C can be achieved. They all use a vacuum purge in the first stage of the annealing process. Beside safety reasons, this is a very important feature from technological point of view as well, because during this period traces of properly selected rolling lubricants evaporate easily and without discoloration. After the vacuum purge 25% H_2 - 75% N_2 protective atmosphere is introduced under the tightly sealed hood. The high convection system using powerful fan allows to bring the charge in a very short period to the annealing temperature selected somewhat lower than usual and this prevents diffusion of zinc to the surface layer.

Bell annealing is a very productive method requiring relatively small investment expenditures.

A disadvantage of coil annealing is that large coils of some alloys in thinner gauges can be easily damaged : one wrap can become welded to the next because of the high temperature and pressure encountered, usually making the coil unsuitable for further processing. Another disadvantage of coil annealing is that it is time consuming.

In the late 1940s continuous strand or strip annealing lines began to be used in brass mills. From these early beginnings, the high speed vertical strip annealers were developed in the 1960s. Annealing lines of this type are now in use for annealing copper and copper alloy strips. When several such lines are available, a variety of thickness ranges can be rapidly annealed, providing great flexibility in production scheduling and enabling fast delivery of finished strip.

The continuous strip anneal lines include payoff reels, a stitcher for joining the front end of a coil to the trailing end of the one preceding it, a degreaser for removing roll lubricants, looping towers for metal storage, a seven-story high vertical furnace which includes a heating zone, a controlled atmosphere cooling zone, and a water quench tank. This is then followed by acid cleaning tanks, a water rinse, a drying oven, and a reel for recoiling the metal.
Degreasing unit removes roll lubricants from the metal surfaces before the metal enters the furnace, so a clean, uniform surface is presented for annealing. The metal passes over a large roller outside the furnace at the top and does not touch anything inside while it is being heated.

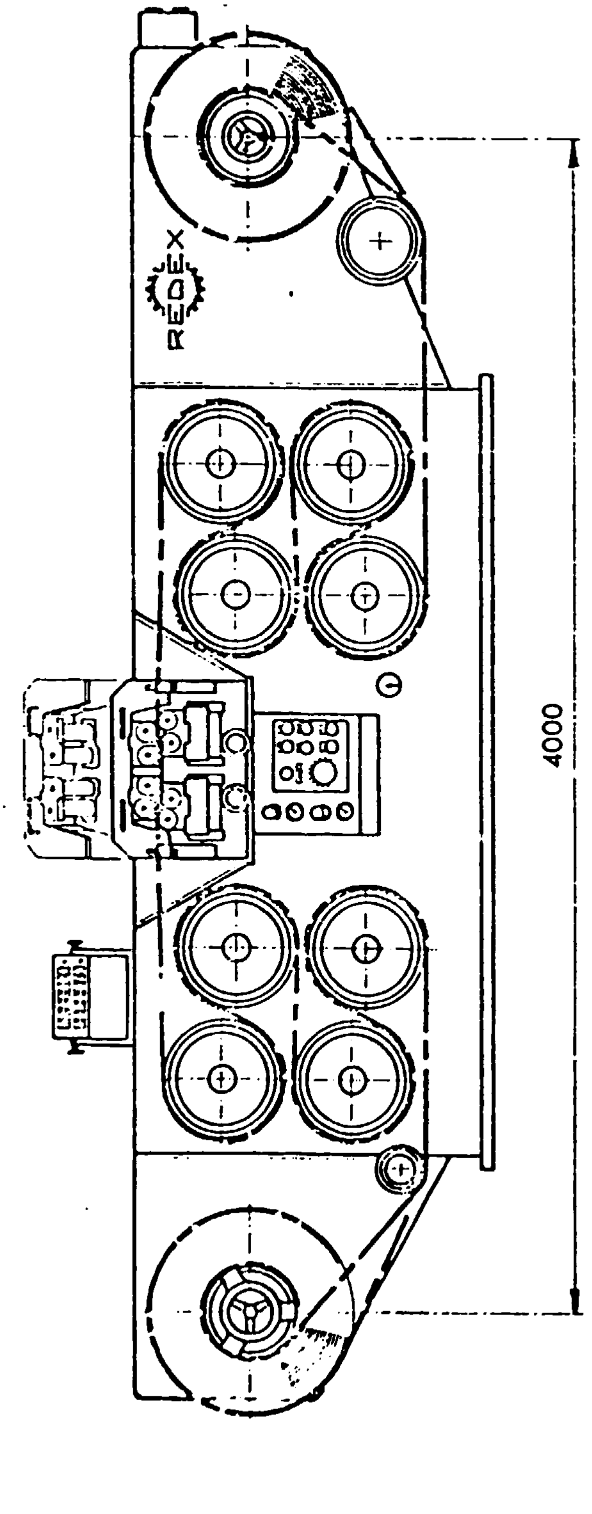


Figure 16. Strip stretch leveller machine

It then passes under another large roller at the bottom in the cooling water tank. This arrangement avoids any possibility of surface damage to the hot metal, which was common in the earlier horizontal strip anneal furnaces.
After acid cleaning, rinsing and drying, the surface is usually coated with a detergent solution or a light sulfur-free oil to protect it during handling in transit.

Since every foot of a coil is exposed to the same temperature for the same period of time as it passes through the strand annealing furnaces, grain size from end to end is uniform.

D) SLITTING, CUTTING, AND LEVELING

Following the final rolling, the metal is slit to final width.
Slitting is accomplished by opposing rotary discs mounted on rotating arbors. These knife sets mesh together as the metal passes between them and shear it into a multiplicity of width.

Processing operations which follow final slitting are occasionally required. Blanking is one such operation. Blanking of squares or rectangles is generally done by cutting to length. The metal is first flattened and then cut to length on a flying shear. When circular blanks are required, they are die cut on a press.
The circles are used for the manufacture of deep-drawn articles, e.g. kitchenware. Coin blanks, cartridge and bullet-blanks and cups are also produced on similar presses from strips.

Edge rolling is another process which may follow final slitting. Edge rolling can produce rolled square edges, rounded edges, rounded corners, or rolled full rounded edges.

For some applications, extremely strict tolerances on flatness are stipulated. To achieve this aim, continuous stretch-levellers are used. These lines are also very well prepared to eliminate some rolling defects, such as waves and buckles, and even slight camber. For width of max. 300mm a compact machine shown on Fig. 16. can be used. It consists of bridle rolls building up tension and multi-roll leveller. A strip elongation of max. 3% can be maintained which is sufficient for a major improvement of flatness.

III. MANUFACTURING OF COPPER AND COPPER ALLOY TUBES, RODS AND WIRES

A) CASTING

The technological route of these products in almost every case starts with casting of billets.
Usually vertical DC semicontinuous or continuous casters are applied. The operation of these machines is described in details in the previous chapter.

Casting of wire rods and large size bars and tubes can be carried out on horizontal continuous casting machines, equipped with graphite mold of desired shape. Features of these lines are the same as those with the strip casters.
After horizontal casting, the surface of the products has to be milled. Cast wire rods are cold rolled, annealed and then drawn to different sizes. For the casting of wire rods - as it was mentioned in Chapter 2 - Outokumpu upcasters are widely used.

The horizontal cast rods and tubes are used for manufacturing of bearings, bushings, washers, etc. at the as-cast and milled sizes, and for hot stamping purposes.

The continuous casting of wire rods, bars and tubes drastically simplifies the technological route. It is a very flexible process regarding the alloys and sizes. Machines are relatively cheap, easiliy maintained, require small building. The production can be easily diversified by adding new lines.
It has to be emphasized that any properly equipped machine can cast wire rod, bar or tube using the desired molds and tools. The process is characterized by low material and energy costs. It is very much suitable for small-scale producers and newly established manufacturing facilities.

The only disadvantages are that the cast tubes or bars have a very coarse grain structure and therefore they don't lend themselves to drawing operations; moreover the size range of the products is limited by the casting parameters.

B) EXTRUSION

The end of the mold or semicontinuously cast billets has to be sawed off to achieve good quality end product. The billets are cut to lengths appropriate for the extrusion press.

Before extrusion the metal has to be preheated to the desired temperature depending on the type of alloy. This is a very delicate operation having great influence not only on the pressing force, speed and other parameters of the presses, but even on the quality of the product.
Gas or induction preheaters are both widely applied, but in many cases a combination of the induction and gas systems provides the best results : homogenous temperature of the billets along their whole cross-section, high heating speed allowing large capacity, and formation of only a very thin oxide layer on the surface.

For the extrusion of bars and tube shells usually horizontal oil or water hydraulic presses are applied with a power typically from 1000 up to 3500 tons.
Tubes can be pierced on the extrusion press itself, if it is fitted with an internal piercer. The piercers usually have a force up to 600 tons. Regarding concentricity of the tubes, very good results can be obtained by using predrilled billets. This method is optimal only if the press is not suitable for good quality piercing, having in mind the increased metal consumption and the extra operations of drilling.

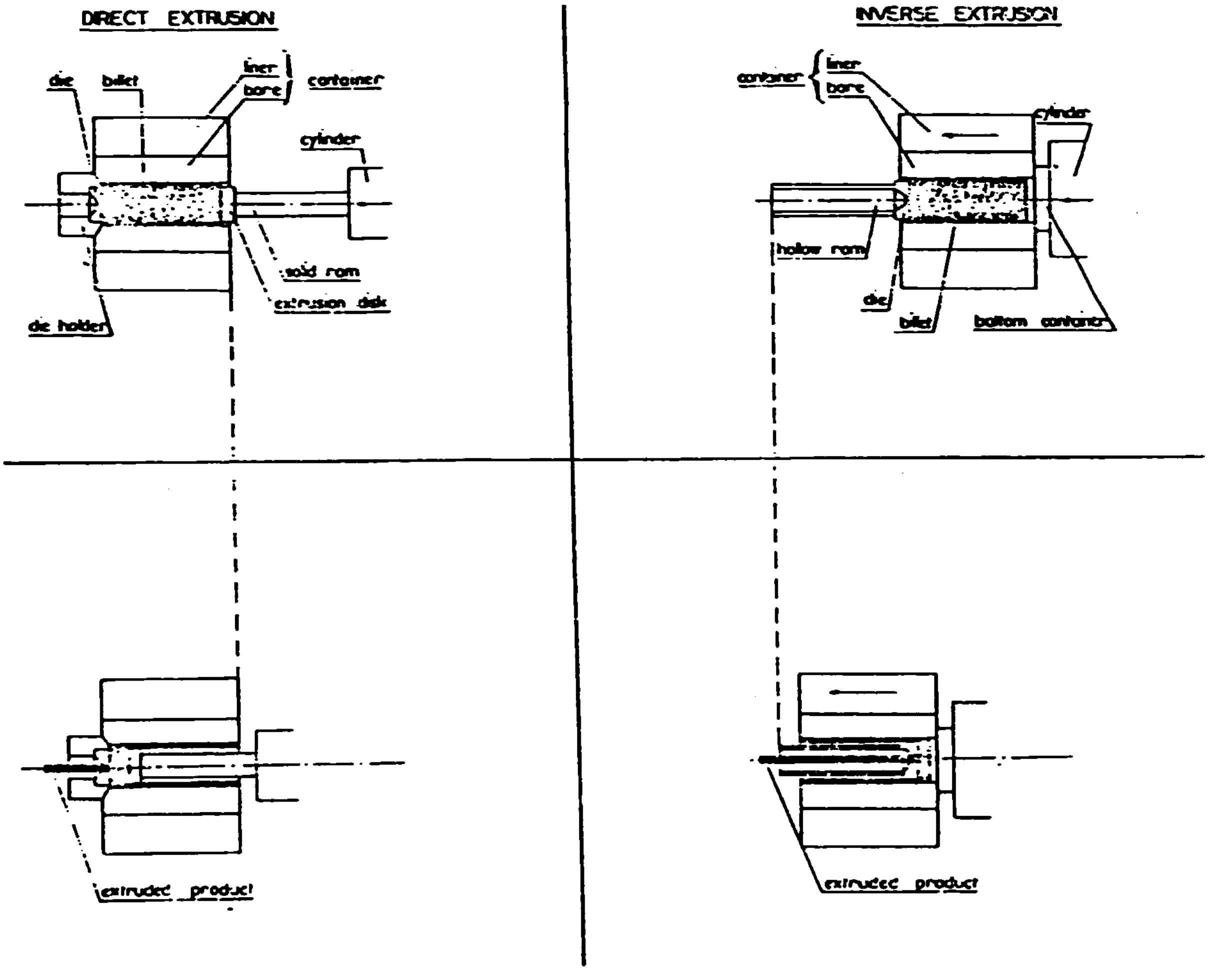


Figure 17. Direct and indirect extrusion methods

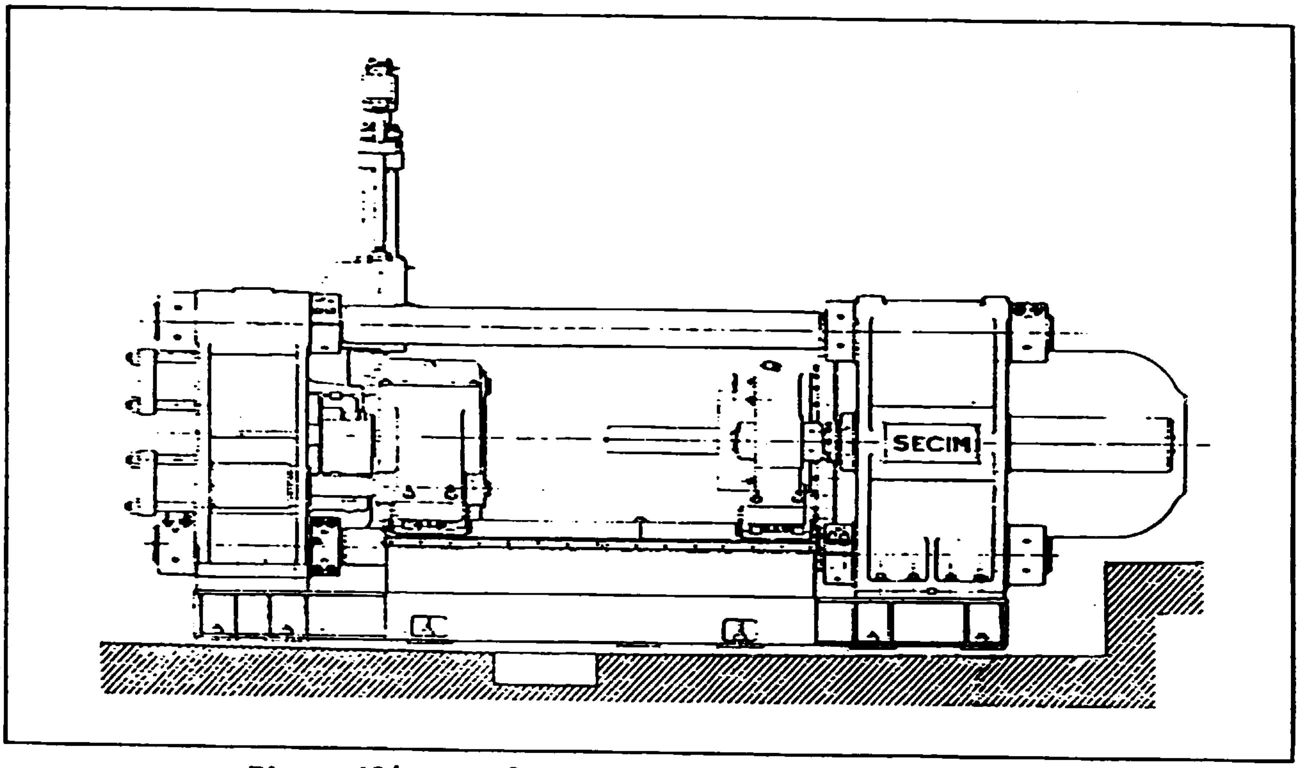


Figure 18/a. Schematic drawing of a direct

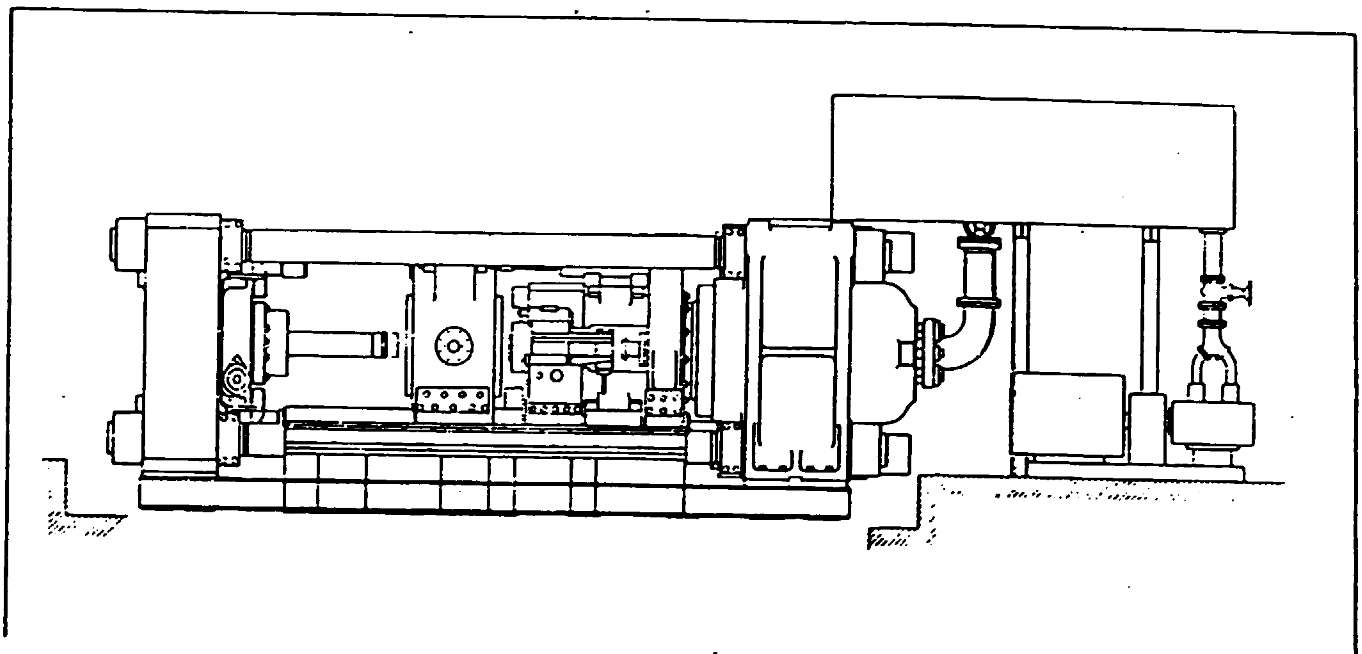


Figure 18/b. Schematic drawing of an indirect extrusion press

In the recent years a new, so-called "indirect" pressing method has started to spread over intensively. This new method, shown on Fig. 21.b, has several outstanding features. Due to the decreased friction the pressing force (energy) is reduced by 15-20 %, speeds are increased by 7-10 %, metal savings can be achieved and wear of the container and its parts is smaller compared to the traditional direct presses.

Existing direct presses are converted into indirect presses in several plants.
The indirect presses at the present stage of art are less suitable for extrusion of tubes and complicated profiles. To achieve good quality of the product, billets are usually extruded with shell. It is a well known fact that the surface of the billets are rich in impurities. To avoid penetration of these impurities into the extrusions, a thin shell collecting most of these particles is formed during pressing. The shell is discarded after finishing the extrusion.

In some cases for the production of smaller tubes vertical presses are used applying force in the range of 1000 tons and using pierced or solid sections.

After pressing extrusions are pickled in dilute sulphuric acid, end-cut, and in case of ready products they are cut to length, straightened, packed and delivered.

C) DRAWING

Other part of the extrusions are pointed and drawn on different drawing benches. Some alloys, e.g. brasses, require intermediate annealing after each drawing operations.

Annealings are carried out in roller hearth furnaces, or in case of wires and coiled tubes in bell annealers. These equipment are described in the previous chapter in details.

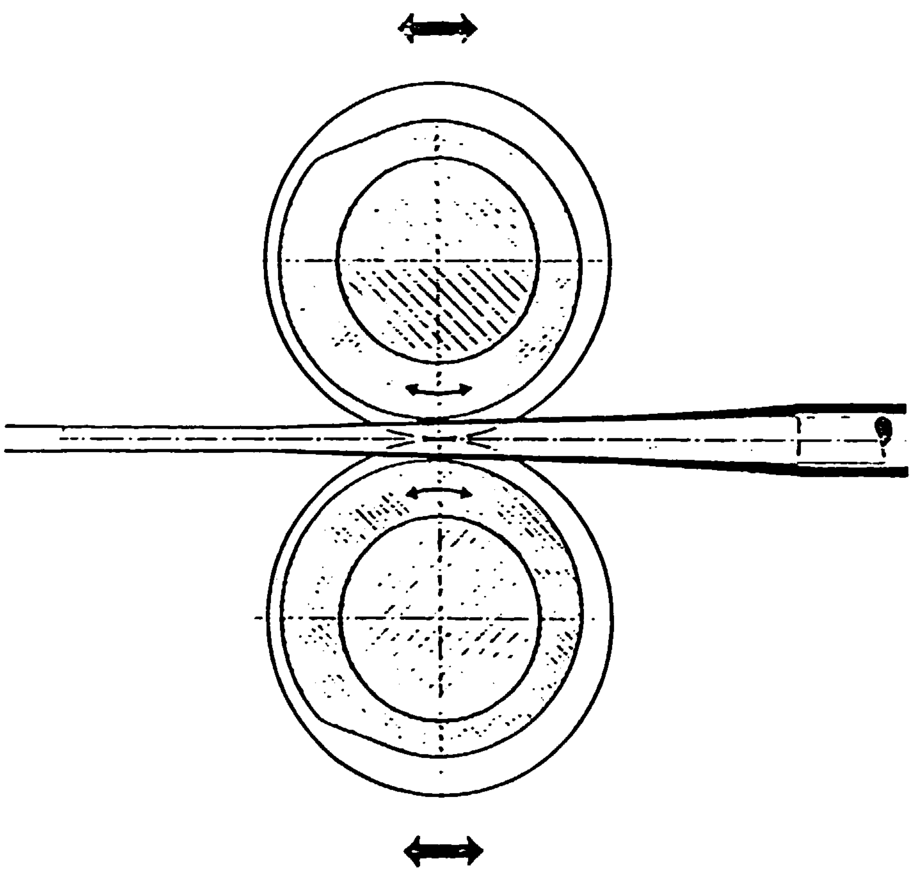


Figure 19. Schematic drawing of a pilger mill

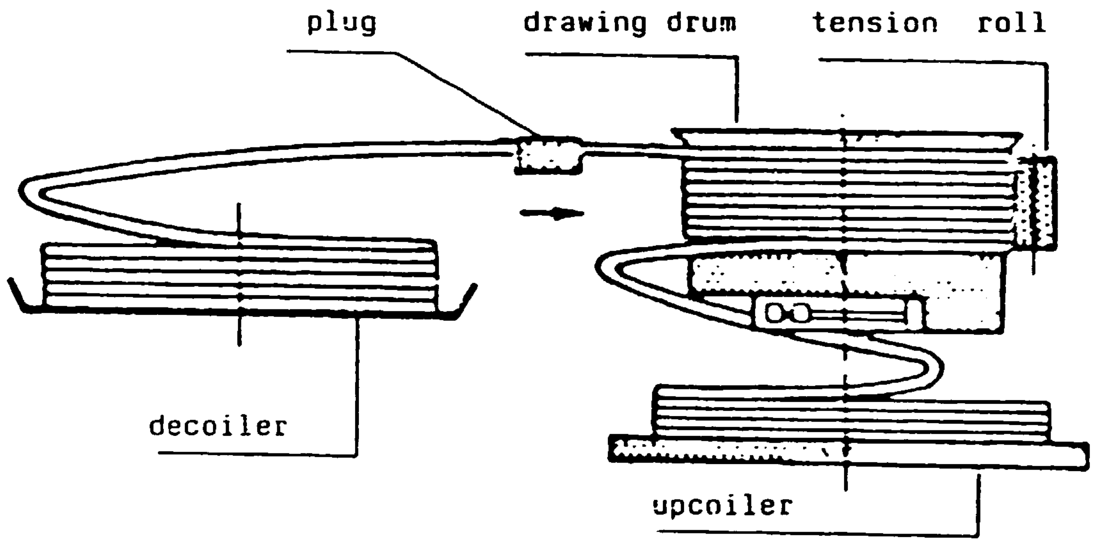


Figure 20. Schematic drawing of a spinner block

a)

b)

c)

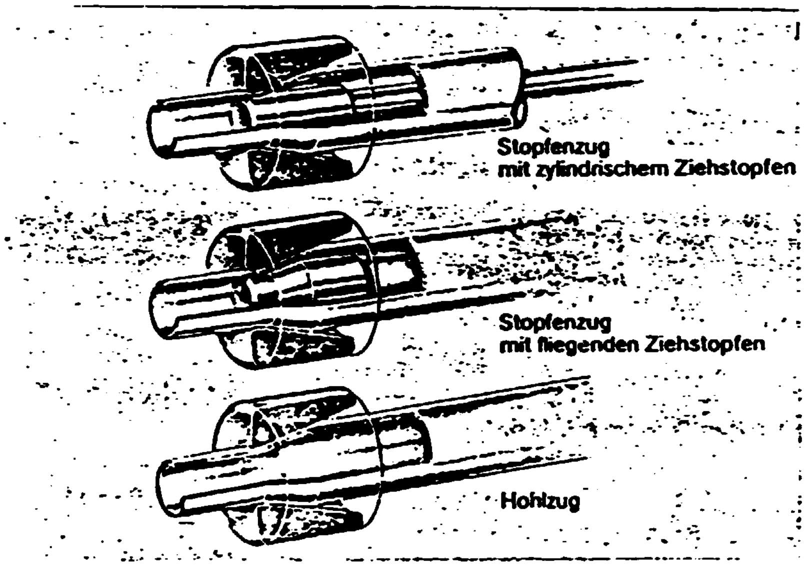


Figure 21. Tube drawing methods

a) tube drawing with cylindric plug
b) tube drawing with floating plug
c) tube drawing without plug

After intermediate or final annealing, the product has to be pickled in dilute sulphuric acid and then washed.

Brass and other copper alloy tubes are usually drawn on traditional drawing benches not in a coil form. Some benches allow drawing of up to three tubes parallel at the same time.

Rods are pressed into larger lengths and coiled after leaving the run-out table of the press. There are several different type of equipment for drawing this type of product, but the most popular among them are the combined Schumag lines allowing continuous drawing, straightening, polishing, sawing, and packing rods of different sizes and cross-sections.

D) COPPER TUBE MANUFACTURING

Copper has much higher ductility than its alloys, therefore the production of copper tubes is different from the previously mentioned technological route.

On the extrusion press heavy gauge tube shell is extruded into water eliminating the oxidation of the surface.
The tube shells are end-cut and directly transported to the pilger mill. The pilger mill is a special rolling mill with alternating rotation of rolls.
It enables to roll long, high-accuracy and good quality copper tubes applying large reductions. The tubes are coiled after pilgering.

Copper tubes weighing up to 250 kgs are drawn in several passes without intermediate annealing on spinner blocks applying "floating plugs". Lubricant and drawing plug are placed inside the tube and a point is made on a push-pointer. The pointed tube is pushed through the die and drawn.
Tubes down to 5 mm OD and 0.5 mm wall thickness are drawn.

35 % reduction is used in each pass. The total reduction is 99 % without intermediate annealing.
The last drawing operation is made on a combined continuous drawing bench which improves the shape of the tube, cut it to lengths, and in case of refrigerating tubes, forms coils of ordered size. The annealing of the tubes is performed in bright annealing roller hearth or batch furnaces.

In the past years the production of copper tube by strip welding has started. By using high-frequency induction welding cf cold rolled and slit strips, combined with spinner-block drawing and finishing lines with incorporated intermediate induction annealing, the production can be made in a more continuous way, and tubes up to 5000 kg per coil can be produced.

WIRE DRAWING

The drawing of wires in principal doesn't differ too much from the production of rods. The only difference is that continuous wire drawing machines with incorporated induction bright annealers are usually applied.
Wires for special applications are flattened on small rolling mills.

IV. OTHER TECHNOLOGIES

Castings and hot stamping of different copper alloys - brasses, bronzes - are widely used in the transport, sanitary, electrical and mechanical engineering.
These items are mostly produced by small specialized companies and rarely in the frame of factories producing non-ferrous semis. The volume of present study doesn't allow to elaborate the various casting and hot stamping technologies in details, so only a few important trends will be mentioned here. At the same time it has to be emphasized that no dramatic changes could be seen in this field during the last decade.

Raw material for these product is mostly scrap mainly collected and purchased outside the factory and partially recycled within the production process. Although the prescriptions for the chemical compositions of the casting alloys are not as strict as the same for the wrought product, it is very important to pay special care and attention for the selection and separate collection, handling and storage of different alloys avoiding their mix-up.

There is a world-wide tendency showing intensification of scrap recycling.
In 1980, the USA, the biggest copper supplier all over the world, gained as much as 46,6 % of his copper production from scraps. In the majority of industrial countries the ratio of production from scraps has also increased between 1970 and 1980; and from the total of 34.8 % in 1970 it grown into 38.8 % in the year of 1980. It is obvious, that the energy consumption has also made an influence on this tendency, which is 13,500 kWh/t for production from ores, while it is as low as 1,700 kWh for production from scraps.

The utilization of scraps and other copper containing waste materials helps to economize with copper ores; at the same time it results in sufficient decrease of environment pollution, for example sulphur emission into the air and diminishes the problem of pollutant storage.

Different casting processes are applied, such as :

- sand casting,
- die casting,
- shell mould casting,
- pressure casting,
- centrifugal casting.

For all these processes copper alloy scrap is melted in oil, gas or electric furnaces of 100-2000 kg capacity. The composition of the melt is controlled by quick laboratory methods, and adjusted if required. After that the metal is cast by the technological route selected depending on the size, shape, quality requirements, size of the series, etc.

In the past decade, more and more producers use horizontal casting instead of centrifugal casting for the manufacture of bearing tubes. The horizontal casting has size and alloy limitations, but it is much more suitable for mass production offering better tolerances, homogenity, lower metal and energy consumption and other advantages.

Hot stamping is a very productive method for mass production of different machine parts and sanitary appliances. As it was previously mentioned, in up-to-date factories these products are made from cut-to-length continuous cast rods of different sizes.

Hot stamping, similarly to die or pressure casting methods, are economically feasible only for parts ordered in large quantities due to the time-consuming and expensive process of die-making. As a result of the relatively high capital investments required - in comparison with sand casting equipment - these production facilities are viable mostly in case of demand over 1000 tpy.

One must note that die-making shop should be an integral part of any well equipped and competitive die casting, pressure casting or hot stamping plant.