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

15C?

TECHNOLOGICAL ALTERNATIVES FOR NON-FERROUS BASE METALS IN DEVELOPING COUNTRIES

by Alexander Sutulov

Santiago de Chile
June 1986

TECHNOLOGICAL ALTERNATIVES FOR NON-FERROUS BASE METALS IN DEVELOPING COUNTRIES

0. EXECUTIVE SUMMARY

01. The post-WWII economic order, as agreed at the Bretton Woods Conference in 1944, was essentially based on acceptance of the US dollar as an international monetary reserve currency, along with gold, at a fixed exchange rate of US$ 35.00 per ounce of gold. Part of the agreement was also the establishment of a fixed rate of exchange among major world currencies, by which governments had to support their currencies - by buying or selling dollars if the pre-established exchange rates fluctuated by more than 1 percent.

02. This was a tough economic order with a lot of monetary discipline, but it assured stability of the post-WWII economy in terms of exchange rates, trade and investment and led the world economy through an unprecedented period of growth and prosperity in the 1950's and 1960's.

03. However, since the beginning of the 1970's this economic order started to suffer deterioration. First came the Monetary Crisis, the most visible sign of which was the suspension of US dollar convertibility into gold (1971). This immediately led to devaluation of US currency in terms of gold. Although this fact was ignored by the US Government, which simply demonetized gold, the fixed rate of exchange among the major international currencies started to deteriorate rapidly, which led since 1972 to replacement of fixed rates of exchange by fluctuating rates. We only now understand the immense impact of this action which destabilized our economic system in all aspects, including trade and investment.

04. By the end of 1973, Monetary Crisis and falling value of the US dollar in terms of gold led to the Oil Crisis, when unit oil prices were increased by roughly four times. The rise of oil prices gave origin to a generalized increase of the energy cost, which increased from 2% of GNP in 1970 to 5.5% of GNP in 1975.

05. Such a drastic increase in the energy cost led to a structural adjustment in the world economy, which first reacted with the 1975 recession (the strongest after WWII) and then with a prolonged period of so called STAGFLATION, when economic stagnation was accompanied by strong inflation between 1976 and 1982. This led to a new collapse in the world economy in 1982.

06. As a result of the energy crisis and the consequences of the stagflation period in the world economy, international indebtedness started to grow dramatically. This happened because the energy crisis led to a colossal redistribution of wealth worldwide, when oil and energy rich countries accumulated immense fortunes while energy-poor countries gradually fell into indebtedness. A contributing factor to the large international indebtedness was also the great liquidity in the world financial system (petrodollars, eurodollars which were freely lended to borrowers at very favorable rates and thus were indiscriminately used to finance development plans, balancing of budgets or simply for consumption).

07. The combined effect of the monetary, energy and financial crises put the end to the post-WWII prosperity cycle and led us to a slowdown of growth in the 1970's and stagnation in growth in the early 1980's.

08. The above described economic scenario introduced drastic changes in consumption and prices of all base metals since the 1970's. Encouraged by post-WWII growth, all base metals, including copper, lead, zinc and tin, experienced exceptionally high growth rates in demand. With the exception of tin, demand for base metals between 1950 and 1970 grew at annual rates that fluctuated between 4 and 5.5 percent per year, which immediately put the pressure on supplies and metal prices because of shortages in production capacity. Higher metal prices in the 1960's encouraged new investment in base metal production and led to a drastic increment in production capacity, precisely when drastic changes in the long-term economic cycle started to take place.

09. Unfortunately, the base metals industry, misled by inaccurate forecasts about the future potential shortage in base metals and natural resources in general, continued its expansion in the 1970's, mistakenly taking 1974 and 1979 as encouraging signs of a new prosperity cycle, when in fact these years, like 1929, were climaxes before the anticlimax. Indeed, in these years, the industry demonstrated its poor judgement and interpretation of economic events, which led it practically to a collapse in the early 1980's.

10. When since the beginning of the 1980's base metals demand started to fall off sharply, the base metals industry was confronted with the following three serious problems: (1) installed production overcapacity, generally about 20% above the required; (2) low metals prices, which fell almost to the level of prices during the Depression, when calculated in constant dollars; (3) tremendous indebtedness of the sector, when in many cases capital/indebtedness ratio approached 50% or less.

11. In the above circumstances and almost without notable exceptions, the base metals industry started to work with tremendous losses since 1981, in which debt servicing absorbed almost all industry income and even required new loans for such servicing. Capital formation in this industrial sector practically ceased since the early 1980's, reason for which these companies became an easy target for takeovers.

12. These demolishing events took place when we were seeing a basic restructuring shift in the world economy from tangibles to non-tangible financial assets. One of the principal characteristics of these changes is a generalized slowdown in demand for raw materials, steel, oil, base metals and others included. Such contraction in demand growth is principally related with lesser consumption of materials and energy per unit of GNP (lesser intensity of consumption), which is in many cases related with a more rational use of materials through miniaturization, substitution and also due to a greater growth of the services sector as compared with production of capital goods, now stagnating due to low investment.

13. In these circumstances, base metals demand stagnated at approximately levels which existed by the end of the 1970's and its growth in the 1980's and early 1990's is expected to be only one half of what it was in the 1970's, and from one quarter to one third of what it was in the prosperous 1950's and 1960's. Specifically, we do not expect a base metals demand growth higher than 1.3% to 1.7% per year for a world GNP growing at 3 to 3.5 percent per year. In the past, base metals demand growth was equal and at times higher than GNP growth and fluctuated between 4 and 5 percent per year.

14. The above conclusions about limited growth for base metals demand indicate that with all the presently installed excessive production capacity, competition for world metals markets will be sufficiently aggressive as to preclude significant growth in prices. In other words, in the medium-term future, base metals industries will have to survive on account of their lower costs and higher efficiency rather than on account of higher metals prices. All econometric studies today point to a slow increase in base metals demand and flat increment of their prices.

15. This situation urgently requires such measures as technological improvements, cuts in labor and energy costs, and overall simplification of production flowsheets because the expected margins should come not from higher prices but from lower costs.

16. In these last months, the world economy experienced some substantial and rather unexpected changes: since November of last year (1985) oil prices started to fall dramatically, which was immediately reflected in lower production costs and higher demand for products. It is still too early to fully assess the magnitude and importance of this event, but one thing is quite clear: after a temporary bouncing of oil (and energy) prices, they will establish themselves at considerably lower levels than before. For example, we can see that after falling from $ 29 per bbl in 1985 to less than $ 10 per bbl in 1986, oil prices will eventually stabilize at, say, between $ 18 and $ 22. This is compared with $ 35 and $ 38 per bbl prices in 1981. Such a sharp decrease in energy cost will add to our economic growth, investment and consumption of all kinds of raw materials, base metals included. After all, the present crisis started with disproportional increases in our

energy costs. Now that this cost has started to normalize, our economy should also acquire higher dynamics.

17. To be sure, these changes are not expected to be immediate because our economic system was already adjusted to life with high energy costs. Now that these costs have fallen sharply, new structural changes are expected because a number of industries will have to adjust to new conditions, among them the oil industry, some speciality steel sectors, all oil country goods products, and above all our governments, which in a good part live on oil taxes, and banks which are involved in important credits to the energy sector. Since such changes are immediately visible in terms of the problems they create rather than the benefits they produce, a lead time of between 12 and 18 months should be given before the potential benefits of this change affect base metals demand.

18. The other important measure, now in the process of consolidation, is the Baker plan to stabilize somehow fluctuating exchange rates among all major currencies. The real results are still invisible, but the fact that major nations talk about this subject is very encouraging, because it will lead to a greater stability in trade and investment, which coupled with lower inflation and interest rates may bring about a significant revival in world economy.

19. The third vital component of our present crisis (the first one being the high energy cost; the second, fluctuating exchange and interest rates) - the huge international indebtedness, which slows our growth, particularly in developing nations - is also receiving a greater, if still not effective consideration. Lower energy costs and lower interest rates will definitely contribute to a better balance of

payments situation in the developing world. But this will not be sufficient to reactivate its economies. With such huge international indebtedness, many developing nations lack sufficient savings and income to service debts, less to embark on economic growth of any significance. Thus, greater flexibility in the treatment of this subject and particularly the availability of new funds for development will be essential for developing nations to make a better use of the present economic conjecture.

20. The combination of all favorable events in our economic life, such as lower energy cost, lower inflation and interest rates, greater stability in fluctuating exchange and interest rates, and improvements in the indebtedness situation, are a good sign for possible improvements in the long-term economic cycle. However, we should be conscious that the changes experienced so far are insufficient to produce a real turnaround in the economic cycle and that much more of the same must be seen before real reactivation in the world economy can be expected.

21. What is, then, the outlook in this context for base metals in the developing countries?

First, we can see that in future expected tough metal markets, developing countries should take full advantage of their cheap labor and, where possible, low cost energy. They also should emphasize their production activities with radically improved technologies based on better use of their polymetallic resources and their complex treatment.

There is a great field in porphyry coppers, which along with introduction of more effective mining methods, permit use of very effective semi-autogenous grinding technology and giant flotation cells. A particularly promising technology exists in the recovery of byproduct molybdenum from porphyry coppers and which consists of recently introduced column flotation. In fact, column flotation technology is of such promise that it possibly can be used for concentration of copper ores themselves. Preliminary plant testing has given encouraging results, but this still must be confirmed by further experimentation.

Further improvements in copper technology are possible at the smelting stage by introduction of oxygen either in flash smelting of concentrates or through its application in the Noranda Process and the modified El Teniente Process.

A particularly convenient technology has been developed for treatment of sulphide and oxide ores by hydrometallurgical means, which avoids expensive traditional concentrating and smelting steps with all their impact on the environment. While sulphide ores can be cheaply leached by ferric solutions helped by bacterial leaching technology with oxides treated by more conventional acid leaching, the new solvent extraction technology gives a possibility to effectively clean such solutions for their final electrowinning step to produce high-purity (99.9% Cu) cathode. This SX/EW technology is now highly popular in developed countries, such as the USA and Canada, because it produces copper at one half of the cost necessary for full treatment of ores by conventional methods. This technology is now increasingly used in developing countries with even a higher economic effect.

22. In the lead and zinc industries, which handle higher grade and more complex ores, but on a smaller scale than copper, significant improvements in technology are visible not so much in the concentrating stage, where classical technologies and flotation concentration are practically and conve-

niently adjusted to every specific ore and its metallurgical problems, but rather in the pyrometallurgical stage. Here, apart from specific processes for individual treatment of each metal, there exist also alternatives for a collective treatment of bulk concentrates to improve considerably over-all metal recoveries and costs. The high complexity of these ores and significant difficulties in separation of individual metals by differential flotation result generally in high metal losses. Thus, processes such as the Imperial Smelting and Sulphate Roasting Process attract increasing attention.

New pyrometallurgical technologies include also the Sherritt Pressure Sulphuric Acid (PSA) leach Process, the emerging KIvcet Process (from USSR), the Outokumpu Process (OKP) from Finland, the QSL Process from Germany and some others.

All these processes should be given due attention with the idea that if not all individual developing countries are in a position to install their own plants, they can conveniently agree on some regional plants which will handle their combined production with a higher efficiency.

23. Tin is now living with very difficult times in view of the collapse of its International Tin Council and the sharp drop in tin prices. The only way to cope with these problems is to concentrate on exploitation of higher grade ores and economically more effective methods. In this respect, a relatively cheap and effective method of flotation for tin fines is brought to attention. Also, the fuming process has received increasing consideration, because of its more effective possibilities to eliminate impurities and increase recoveries into final product.

24. In general terms, the developing countries have already built sufficient scientific and technological capacity for

their own research in all pertinent areas of base metal production. Some of these research centers 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 centers should enjoy all the support and understanding of their national governments and of international institutions, and no effort should be spared for intensive training and formation of national technological elites to carry forward national and regional development plans.

25. More particularly, developing countries should be conscious that raw materials supply difficulties are affected by a serious structural problem because the labor cost in industrially developed nations has become excessive and really prohibitive under present economic conditions. This leads to important shutdowns all across the base metals industries in the developed world and orients industrial nations increasingly towards supplies from developing countries. This is particularly felt in copper, although shutdowns in the lead, zinc and tin industries have been also important.

26. Such tendencies require that developing nations prepare themselves for future challenges with adequate technologies in order to foster such a favorable trend which can lead to a new international distribution of labor and speed up their industrial development and integration.

27. In view of all this, the time has arrived when developing nations should make a decisive effort to strengthen their

economies and trade balances by expanding their base metals output aggressively through technological innovation, economic integration and international cooperation by which their competitiveness in the world metals markets will be greatly fostered. This is incidentally one of the only few options which developing countries have to face a better economic future.

28. In anticipation of these changes, 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 improve efficiency, decrease costs and increase the competitiveness of developing countries in international markets. This initiative should also help developing countries to form their technological elites 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.

29. As we show in this report, the greatest potential benefits can be obtained today in the area of copper related technologies, where taking into consideration the overall copper mining production in developing world of some 3.8 million tpy and that of smelted and refined copper of some 2.9 million tpy, the following potential benefits can be obtained in the different technological areas:

Mining	US$ 100 to $ 120 million/year
Comminution	US$ 100 to $ 150 million/year
Flotation	US$ 30 to $ 100 million/year
Smelting	US$ 100 to $ 200 million/year
Hydrometallurgy	US$ 100 to $ 200 million/year
Total	US$ 430 to $ 770 million/year

At the present copper prices (65 c/lb) and production rates (3.8 m tpy), these potential benefits represent from 8% to 14% of the present copper revenue of developing countries, which is today about US$ 5.4 billion. In terms of savings per pound of copper, this would amount to from 5.2 to 9.1 c/lb, which is sufficient to provide safe margins for copper production activities in the developing world.

30. Since such a project would require the ample cooperation of governments of developing copper producing countries, international institutions, industrial organizations, and engineering firms, it would be convenient at the next experts meeting to raise the question of the feasibility of creating a broadly based project with the aim to cover the copper area in terms of new and emerging technologies for general improvement of production efficiency and costs and greater competitiveness of developing nations in international markets. Such a project carried out at research institutions in developing world will also contribute to the creation of new technological elites in these countries and to their greater industrial development.

1.0 INTRODUCTION

This report has been prepared in fulfillment of agreement CLT 86/036 between the undersigned and the United Nations Industrial Development Organization (UNIDO). Its purpose is to give a broad based survey of existing technologies which are used in processing and recovery of four non-ferrous base metals, copper, lead, zinc and tin, and new innovative patterns of development which they follow. This is in order to design a development strategy for these resources in developing countries, with a possible optimization of results in medium and long terms, as referred to national mining activities, international markets and possible cooperation between major sectors of the world economy, such as for example North/South or South/South, as reflected in documents of the United Nations.

This analysis, obviously, had to be started with a general, if necessarily brief, survey of the principal base metals resources of developing countries in order to obtain a clear idea where and for which areas the importance of the present technological analysis can be assessed. Then, the general situation of the present supply-demand equation was analyzed for these four base metals, with its implication on present and future base metals prices. This was particularly in view of the changing world supply-demand situation, which has both cyclical and structural characteristics that affect differently future demand.

From this preliminary analysis of these base metals was derived the scenario in which they will have to operate in the future and to which technological changes and optimization have to be applied.

After this introductory part of the report (Chapters 2 and 3), we turn to the basic problem treated, which is to assess the present state of the art, its present technological trends and its likely future evolution. We analyzed these problems first in general terms, and then specifically for each metal to cover more extensively each possible alternative and its economic significance.

In the final part of the report, we make conclusions about the possible implications of new technologies on the development of these four base metals and their production in developing countries, and also comment on possible steps which can be taken through research and international cooperation to foster such progress.

In general, as we will see, we are living today in an exceptionally complex period in the world economy, which is caused by combined cyclical (i.e., temporary) and structural (permanent) changes. Both types of changes, which have coincided since the mid 1970's and which were strongly affected by consecutive monetary, energy and financial crises, have brought metals prices down to their minimum levels because of a considerable buildup in supplies and rather restricted demand, caused in these last years by deceleration of our economic growth and, particularly, the creation of new wealth through investment in capital goods, which has been substituted by a massive shift from tangibles to financial speculation.

In these circumstances, all four base non-ferrous metals, as well as almost all other raw materials and commodities, suffer from excessive supplies and low market prices, which have dramatic repercussions in corporate balances and require an extraordinary effort of streamlining and readjustment to new realities. In the medium term, metals prices are quite unlikely to improve, probably until the 1990's, which will require from all base metals producers an effort to cut down on costs rather than to expect improved prices. This will require introduction of new and more efficient technologies, substantial cuts in the labor force and its salaries, as well as other structural adjustments.

In a longer term, the perspective, probably from 1995 on, is more favorable, but competitiveness in technological efficiency and costs will still be primordial.

The developing world has some natural advantages for production of base metals. These are their generally higher grade of ores, as reflected in higher content of metals, their higher byproduct contents and substantially lower labor cost. However, to make full use of these advantages, and particularly in treatment of complex polymetallic ores, new and more efficient technologies should be applied so that the metals are effectively recovered in high grade products to the fullest extent. It is in this area where the most significant advances can be made and where technological research will be the most beneficial.

As we show, the Developing World already has a significant scientific and technological research base, whose effectiveness can be substantially expanded by international cooperation and multinational sponsorship of key research projects. This cooperation obviously includes developed and developing nations from the south and the north of the globe and will make beneficiaries of all those who actively participate in such programs. It should be clearly understood that what economically underdeveloped nations cannot really afford is to remain also scientifically and technologically underdeveloped, because then there will be no hope for the future.

In this context, we consider that a pragmatically designed and carried out research program, placed now in the area of four non-ferrous base metals, will doubtlessly contribute very significantly to the improved, competitive position of developing countries in future metal markets. It should be carried out now, in times of low metal prices and general demoralization of all major competitors, in order to capitalize on all expected advantages when metals demand and prices improve.

Simultaneously, such a program will contribute to the industrialization of developing countries, which is the first step to their economic independence and progress. However, at this stage, cooperative rather than individualistic national effort should be favored. This is because of the nature of the problems, which in many cases will require a common cooperative effort with an international division of labor and multinational investment effort. In some cases, effective use of new technology is dictated by the economies of scale, which is often too big for one nation but may work better when served from one location to various participants.

In this report, the author has mostly relied on his own professional experience and the fundamental part of the presentation of the economic moment, and importance of the different alternatives of technological character are of his own appreciation. However, in the description of different processes he has used the available and updated bibliography and freely reproduced both illustrations and descriptions from available texts in order to maintain their maximum eloquence and professional value.

2.0 BASE METALS RESOURCES OF THE DEVELOPING WORLD

In order to demonstrate the possible practical application of the findings of this report to Developing World countries, we prepared a summary of the base metals resources of some 20 of the most potentially important producers in this technological area. Obviously, the information presented in Table 1 is not complete, and several other countries could be added to this list. But what we intended to do is to give a brief summary and classification of the most important deposits of copper, lead, zinc and tin, of those which still await their development, and to classify in general the countries mentioned according to the importance of their resources, their metal content, complexity of mineralization and economic viability of projects.

In this context, we have used the following abbreviations to characterize the different types of orebodies in the specified countries:

In respect to:

RESOURCES	WC-world class, over 5% of world resources	IM-important	SG-significant
METAL CONTENT	HG-high grade	MG-medium grade	LG-low grade
MINERALIZATION	MM-monometallic	BM-bimetallic	PM-polymetallic
ECONOMIC VIABILITY	PV-presently viable	VF-viable in future	VD-viable in distant future

TABLE 1

BASE METAL RESOURCES IN DEVELOPING COUNTRIES

COUNTRY	COPPER	LEAD ZINC	TIN
ARGENTINA	IM LG PM VF El Pachon, San Juan Alumbrera, Catamarca	SG MG PM PV Huemules,Chubut (Pb,Zn,Cu,Au,Ag)	SG MG MM VF Palpala, Jujuy
BOLIVIA		SG HG PM PV Colivar, Oruro (Pb,Zn,Ag,Sn)	SG MG MM PV Kenko, Catavi Machacamarca,Oruro
BRAZIL	IM LG BM VF Salobo, Carajas state of Parana	SG MG PM PV Paracatu. Minas Gerais	IM MG MM PV Rezende, Rio deJ Minacu, Goias
CHILE	WC HG BM PV Escondida, Antofagasta Cerro Colorado,El Abra Pelambres, Quebrada Blanca	SG MG PM PV Aisen	
COLOMBIA	SG LG BM VD Mocoa, Putumayo Cu-Mo		
ECUADOR	SG LG BM VD Chaucha - Cu-Mo	SG MG PM PV San Bartolome,Azuay (Ag,Zn,Pb,Cd,Au)	
MEXICO	IM LG BM PV Cananea, Sonora IMM Zacatecas	WC MG PM PV Tepehuanes,Durango Charcas, Zacatecas Parral, Chihuahua	
PANAMA	SG LG BM VD Cerro Colorado,Chiriqui Pataquilla, Espinar		
PERU	WC LG PM VF Cerro Verde II,Arequipa La Granja, Cajamarca Tambo Grande,Piura (Cu, Zn, Ag)	WC HG PM PV Antamina, Ancash San Cristobal, Andaychagua Iscaycruz, Cajamarca-Zn Huanzala,Huanuco	
PAPUA NEW GUINEA	IM LG PM PV Ok Tedi Cu-Au-Mo		
INDIA	SG LG MM PV Chitradurga,Karnataka Malanjkhand. Madhuya Pradesh	SG MG PM PV Banaskantha, Gujarat (Cu, Pb, Zn) Baroi, Rajastan Rampura Agusha, Rajastan	
INDONESIA	SG MG BM VF Tambulilato,N.Sulawesi Cu-Au		IM MG MM PV Bangka Island
MALASIA	SG MG MM VF Pekan Sri Jaya		WC MG MM PV Kuala Langat,Sel.
PAKISTAN	SG LG BM VD Saindak,Beluchistan Cu-Mo		
PHILIPPINES	IM LG BM VF Mankayan, Benguet Cu-Au Hinobaan,Negros Occidental	SG MG PM VD	
THAILAND	SG LG BM VD Khao Loan, Nong Khai	SG MG BM PV Mae Sot -Zn	WC MG MM PV Offshore -3[illegible] Phangnga,P[illegible]et
TURKEY	SG MG PM PV Asikoy, Cure Cu-Fe Cayeli- Cu-Zn-Ag-Au ~~Sirt Madenkoy Cu-Zn-Fe~~	SG MG PM PV Balaya Pb-Zn	
SAUDI ARABIA	SG LG BM VF Jabal Sauid - Cu-Zn ~~Al Masane -Cu-Zn-Au-Ag~~		
ZAIRE	WC HG BM VD Tenke Fungurume	SG MG PM VF	SG MG MM PV Sominki
ZAMBIA	WC HG BM PV	SG MG PM PV Karenda, Central Prov	SG MG MM PV Choma , Kalomo

The examples cited in each area are principally the pending projects having a certain degree of certainty for development. They do not necessarily reflect characteristics given for this category as applied to the whole country. For example, Chile, which is characterized by its large high-grade bimetallic (Cu-Mo) deposits, has a perspective for development of another new such deposit at Escondida, but the other projects such as Cerro Colorado, El Abra, Pelambres and Quebrada Blanca are considered to be low-grade.

We also tried, where possible, to specify with the name of deposit its geographical location by indicating the name of the state or province, as for example El Pachon, province of San Juan or Tepehuanes, state of Durango. We omitted words "state" or "province" for reasons of space.

Since most of lead and zinc ores are complex and generally found together, we put them all in the same category, indicating however when lead or zinc appear separately or in company with copper and other precious metals.

From Table 1 it clearly appears that in copper, world category in these deposits is achieved by Chile, Peru, Zambia and Zaire. Mexico is slightly less important, but Brazil, Colombia, Ecuador, Indonesia, Panama and Pakistan acquire their importance generally because of one very large low-grade orebody. The absence of new potential developments in Zambia is generally due to a very difficult economic situation of this country, which has problems to maintain even its present production capacity. In Zaire, the situation is about the same, but the Tenke Fungurme project was well studied and postponed indefinitely because of the financial arrangements. In Chile, the Escondida project is very promising and will be one of the first projects to materialize. Besides there are a large number of expansions in existing mines.

In general, copper production in the developing world is the most promising area because of both the size of deposits and their quality. It turns out thus to be of great importance for industrially developed nations in terms of supplies.

In lead and zinc, world category is achieved only by two countries, Peru and Mexico, which have very large high-grade resources accompanied by important byproducts, particularly silver. Of lesser importance are resources and projects in Bolivia, Argentina, Brazil and India. However, the present oversupply situation requires especially favorable conditions in terms of grades, technology and energy to favor their development.

Finally, tin, now in shambles, is the least promising resource today among all base metals because of its structurally shrinking demand and recent catastrophic fall in prices, which leaves much of its installed production capacity unused. However, Bolivia, Brazil, Malasia, Indonesia and Thailand - all of them in the world category of resources - may have some high-grade deposits which can compete with existing mines if properly developed or if treated in an innovative way. This is, for example, the case with an offshore deposit in Thailand, 35 kms outside of Phangnga Phuket, which will require a completely new technology to be exploited. Such technology will be required to be developed in combination with Billiton, a Royal Dutch subsidiary which is interested in it.

A further analysis of the base metals industry and resources situation in developing countries indicates that while some of them can produce a considerable surplus for exports, others need to expand their existing production in order to satisfy national needs and reduce imports of these metals. The typical cases for the first type of countries are those with relatively small population and large resources, such as Zambia, Chile, Peru, Papua New Guinea, Bolivia and some others. The second group is obviously composed of India, China, Argentina, Brazil, Mexico, Pakistan, Turkey and some others. Some of developing countries, although large in population, produce a substantial surplus of base metals production, which then allows significant exports. This is the case of Zaire, Mexico (lead and zinc but not much so for copper), the Philippines, Malasia, etc. In any case, what is important is that in each case greater base metals production improves the balance of payments situation in each country, either by increasing exports or decreasing imports, or both, and significantly contributes to the industrialization and general development of the country, because the expansion of mining and metallurgical industries has a multiplying effect on other productive sectors, for example, steel and construction materials industry, the energy sector, transportation and a number of servicing industries.

Geopolitically, new mining and metallurgical activities bring new life to desertic or abandoned areas, promote colonization and bring about creation of new industrial complexes. This is particularly important in new or desertic countries, such as in Latin America or Saudi Arabia.

All this only stresses the importance of base metals in the general economic development of developing countries and transforms it into a focal point for future progress. The industrialization process here goes hand in hand with electrification, develops new communications, creates new transportation lines, gives place to numerous other industrial and craft activities, creates new towns and cities and generally improves the standard of living of the population.

3.0 BASE METALS PRODUCTION, DEMAND AND PRICES

In attached Tables 2, 3 and 4 we give base metals consumption and production figures along with the evolution of their prices in function of the world supply-demand situation.

Broadly speaking, prices are the function not so much of production costs as of market conditions as reflected in supply (production + stocks) and demand. Excess of supplies over demand always is reflected in falling prices, while shortages in supplies result in escalation of prices.

From Tables 2 and 3, we can observe how copper prices from an average level of 28 - 31 c/lb before 1965 started to experience drastic increases since 1966 and reached levels 50% and 100% higher in years between 1966 and 1970 than in the previous period. This was due to a drastic increase in demand, which grew from 6 million tpy to 7.3 m tpy in the same period and which required higher copper prices for stimulation of investment. Prices temporarily fell in 1971 and 1972 due to recessive conditions, but demand started to grow vigorously in 1973 and 1974, which resulted in new record price levels of 79 and 90 c/lb. In part, this was the consequence of the oil price shock, which between 1972 and 1974 increased its level by 4 times, and when all commodities expected that comparable increases in raw materials prices would happen. Similar increases happened in zinc, tin and on a lesser scale in lead.

But the 1975 recession, the most severe recession since WWII, dissipated all doubts about the problem: price hiking affected only the energy sector and particularly oil and hydrocarbons, while it had depressive effects on the base metals sector. The only base metal able to continue with its price growth was tin because of the formation of an international tin cartel which

TABLE 2

BASE METALS PRODUCTION : 1960-1985

(in thousand metric tons)

YEAR	COPPER	LEAD	ZINC	TIN
1960	4,998	2,717	3,151	195
1961	5,127	2,822	3,334	188
1962	5,296	2,776	3,472	187
1963	5,400	2,949	3,583	185
1964	5,739	3,100	3,846	185
1965	6,059	3,208	4,068	190
1966	6,324	3,318	4,280	195
1967	6,004	3,373	4,346	214
1968	6,653	3,545	4,819	224
1969	7,212	3,860	5,252	217
1970	7,592	3,988	5,218	221
1971	7,404	3,937	5,104	225
1972	8,100	4,086	5,523	233
1973	8,545	4,224	5,821	229
1974	8,909	4,281	5,993	225
1975	8,356	4,741	5,462	223
1976	8,792	4,989	5,792	225
1977	9,079	5,285	5,974	223
1978	9,205	5,382	6,036	235
1979	9,354	5,548	6,437	245
1980	9,483	5,401	6,145	240
1981	9,732	5,379	6,186	241
1982	9,424	5,273	5,951	225
1983	9,734	5,298	6,299	204
1984	9,549	5,375	6,578	206
1985	9,583	5,468	6,596	211

TABLE 3

BASE METALS CONSUMPTION : 1960-1985

(in thousand metric tons)

YEAR	COPPER	LEAD	ZINC	TIN
1960	4,756	2,617	3,082	201
1961	5,039	2,695	3,241	202
1962	5,105	2,788	3,384	212
1963	5,406	2,922	3,620	218
1964	5,995	3,149	3,952	211
1965	6,193	3,182	4,096	210
1966	6,445	3,333	4,280	214
1967	6,195	3,323	4,346	209
1968	6,523	3,667	4,729	216
1969	7,148	3,836	5.127	224
1970	7,283	3,907	5,063	225
1971	7,310	4,010	5,200	226
1972	7,945	4,158	5,728	231
1973	8,792	4,419	6,262	251
1974	8,325	4,350	5,963	241
1975	7,444	4,759	4,980	216
1976	8,539	5,179	5,776	239
1977	9,057	5,492	5,789	229
1978	9,527	5,521	6,304	234
1979	9,842	5,636	6,369	232
1980	9,390	5,392	6,176	222
1981	9,509	5,289	6,087	212
1982	9,054	5,264	5,966	200
1983	9,115	5,285	6,355	205
1984	9,791	5,390	6,464	232
1985	9,414	5,330	6,510	220

TABLE 4

EVOLUTION OF BASE METALS PRICES : 1960-1985

(in cents per pound of metal)

YEAR	COPPER	LEAD	ZINC	TIN
1960	29.9	11.9	12.9	101.4
1961	27.9	10.9	11.5	113.3
1962	28.5	9.6	11.6	114.6
1963	28.4	11.1	12.0	116.6
1964	31.0	13.6	13.6	157.6
1965	35.6	16.0	14.5	178.2
1966	49.5	15.1	14.5	164.1
1967	47.2	14.0	13.8	153.4
1968	50.3	13.2	13.5	148.2
1969	62.0	14.9	14.6	164.5
1970	62.7	15.6	15.3	174.2
1971	47.9	13.8	16.1	167.3
1972	46.5	15.0	17.7	177.5
1973	78.8	16.3	20.7	227.6
1974	90.4	22.5	35.9	396.3
1975	53.2	21.5	38.9	339.8
1976	60.9	23.1	37.0	379.8
1977	56.7	30.7	34.4	534.6
1978	59.2	33.6	31.0	629.6
1979	92.3	52.6	37.3	753.9
1980	101.4	42.6	37.4	846.0
1981	83.7	35.5	44.5	733.0
1982	72.7	25.5	38.5	653.9
1983	77.8	21.7	41.4	654.8
1984	66.7	25.5	48.6	567.8
1985	65.5	19.1	40.4	525.9

coordinated production with demand, accumulated buffer stocks and dictated prices in correlation with fuel price increases.

During the 1976-1982 period, commonly known as the stagflation period, which was characterized by an unusual combination of economic stagnation with inflation, prices continued to grow more in function of existing inflation conditions rather than of a vigorously growing demand, which in fact remained rather static. Normally, during stagnation periods, metal and other commodity prices fall, while their growth is propelled only by the growth of the economy. This time just the opposite happened. It was in this period that the world economy started to enter into growing indebtedness under a false presumption that it was convenient to be indebted during inflationary periods because afterwards debts would be paid in devaluated currency.

This policy eventually led to an even sharper decline in the economy than in 1975, decline which is known as the 1982 recession. All base metals demand, culminating in 1979, started to fall off in the early 1980's and ended up with huge oversupplies, as reflected both in production and stocks. This led to a catastrophic fall in metals prices.

In real terms, base metals prices, except for tin, are the lowest in years, which leads to numerous shutdowns in all production sectors and particularly in copper, lead and zinc.

But the above described changes, which can be termed as cyclical, i.e., temporary changes, only partially explain the stagnation in demand for base metals. Apart from them we also observe structural changes, i.e., permanent changes in the demand for metals, which come from such phenomena as substitution, miniaturization or the drastic switch in the economy from production of goods to services, which decrease metal demand per unit of Gross National Product. These changes are particularly well observed in the so called intensity of consumption figures, which are the quantities of metal necessary to produce a unit (say, $ 1,000) of GNP, when this is expressed in constant currency.

For example, if we express world GNP in constant US dollars of 1984, then the following fall in intensity of consumption for the four base metals can be observed (see Table 5).

For most metals, the maximum metal consumption per unit of GNP happened in the 1967 period, but the falloff in others started even earlier. This is particularly the case of tin and lead, uses for which replacement or reduction of consumption had already started in the late 1950's and early 1960's.

From 1966 on, the other part of the problem was a fast growth of the services sector in our GNP. As known, this sector practically does not consume metals. The apparent contradiction of economic growth in the last 3 years and static consumption of metals is due precisely to this factor: we grow in services but do not grow in production of goods. Construction, mining, agriculture, manufacturing, etc., all are in a stagnation period because our growth is exclusively or predominantly in non-tangibles, such as bonds, securities and financial speculation. In other words, we are creating paper wealth rather than material wealth. This obscures the panorama for raw materials, including energy and materials resources, part of which are base metals and steel.

To distinguish between structural and cyclical changes, we have prepared Diagram No. 1, which shows the growth of four base metals demand in a longer-term perspective, since 1860 when the decisive period of the Industrial Revolution started. In the 19th century, lead was a more important metal than

TABLE 5

WORLD GNP AND INTENSITY OF BASE METALS CONSUMPTION

(all figures in kgs/US $ 1,000 of 1983)

YEAR	WORLD GNP in $billion	COPPER tons 10^3	COPPER $\frac{kgs}{\$1000}$	LEAD tons 10^3	LEAD $\frac{kgs}{\$1000}$	ZINC tons 10^3	ZINC $\frac{kgs}{\$1000}$	TIN tons 10^3	TIN $\frac{kgs}{\$1000}$
		c o n s u m p t i o n							
1960	5,405	4,756	0.88	2,617	0.48	3,082	0.57	201	0.037
1965	6,980	6,193	0.89	3,182	0.46	4,096	0.70	210	0.030
1970	8,904	7,283	0.82	3,907	0.44	5,063	0.57	225	0.025
1975	10,832	7,444	0.69	4,759	0.44	4,980	0.46	216	0.020
1980	12,811	9,390	0.73	5,392	0.42	6,176	0.48	222	0.017
1985	14,195	9,414	0.66	5,330	0.38	6,510	0.46	220	0.015

both copper and zinc. It was an old and widely used metal, so the Industrial Revolution greatly promoted its uses. Lead kept its leadership over copper until the beginning of the 20th century. Its priority over copper started to fall off roughly since 1910, when the electrification period entered into a massive and decisive phase.

Curiously enough, copper demand growth was closely followed by zinc, another newer base metal which found extensive uses in steel coatings, paints and other protective purposes prior to the development of stainless steels and ferro-alloys. From 1880 to 1914, zinc consumption was practically the same as that of copper. It was during WWI when copper uses exceeded those of zinc, but it was after the Great Depression when copper recovered more vigorously than zinc and maintained its advantages over zinc in the postwar period. By 1960 copper consumption was 1.7 million tons per year higher than that of zinc, and by 1980 the difference surpassed 3 million tpy.

Zinc demand trajectory in comparison with lead was, however, considerably better. During WWII, zinc consumption surpassed lead demand and by 1960 the difference was about 460,000 tpy. By 1969, the difference rose to 1.3 million tpy; by 1974 it was 1.6 m tpy, but then narrowed to 600,000 - 700,000 tpy, when zinc demand growth started to fall off, while lead consumption growth increased.

Nevertheless, it can be generally observed that structural changes in lead demand had already started in the 1920's and only important uses in the dynamically growing automotive industry (batteries

The most notorious structural change in base metals consumption can be observed with tin. Throughout the 19th century and first half of the 20th century tin demand grew impressively, particularly for tin cans used for all sorts of commodities. However, WWII dealt a mortal blow to this growth by substituting all sorts of alternate materials for tin. Since those days, tin demand has narrowly fluctuated between 200,000 and 220,000 tpy and has almost no chance for significant demand growth in the future.

It will be observed that the stagflation period, roughly since the oil crisis, brought metals consumption down to stagnating levels. Similar curves can be constructed even for substituting materials, such as aluminum and plastics, which only confirms the cyclical rather than structural falloff of all basic materials and commodities. This implies that once the present "turbulence period" is over, a new long-term growth period, similar to those of the second half of 19th century and of the prosperous 50's and 60's, may start again. Since the present depression was brought on by the monetary crisis, the most visible manifestation of which was the suspension of convertibility of the US dollar into gold, followed by the oil and energy crisis leading to the colossal international redistribution of wealth and indebtedness and ending up in the present financial crisis, the primary conditions for long-term cyclical recovery are: (1) stabilization of our monetary system through elimination of excessively fluctuating rates; (2) reduction of energy costs to more reasonable levels; and (3) a realistic solution to international indebtedness through lower interest rates and possibly through an important devaluation of the US dollar.

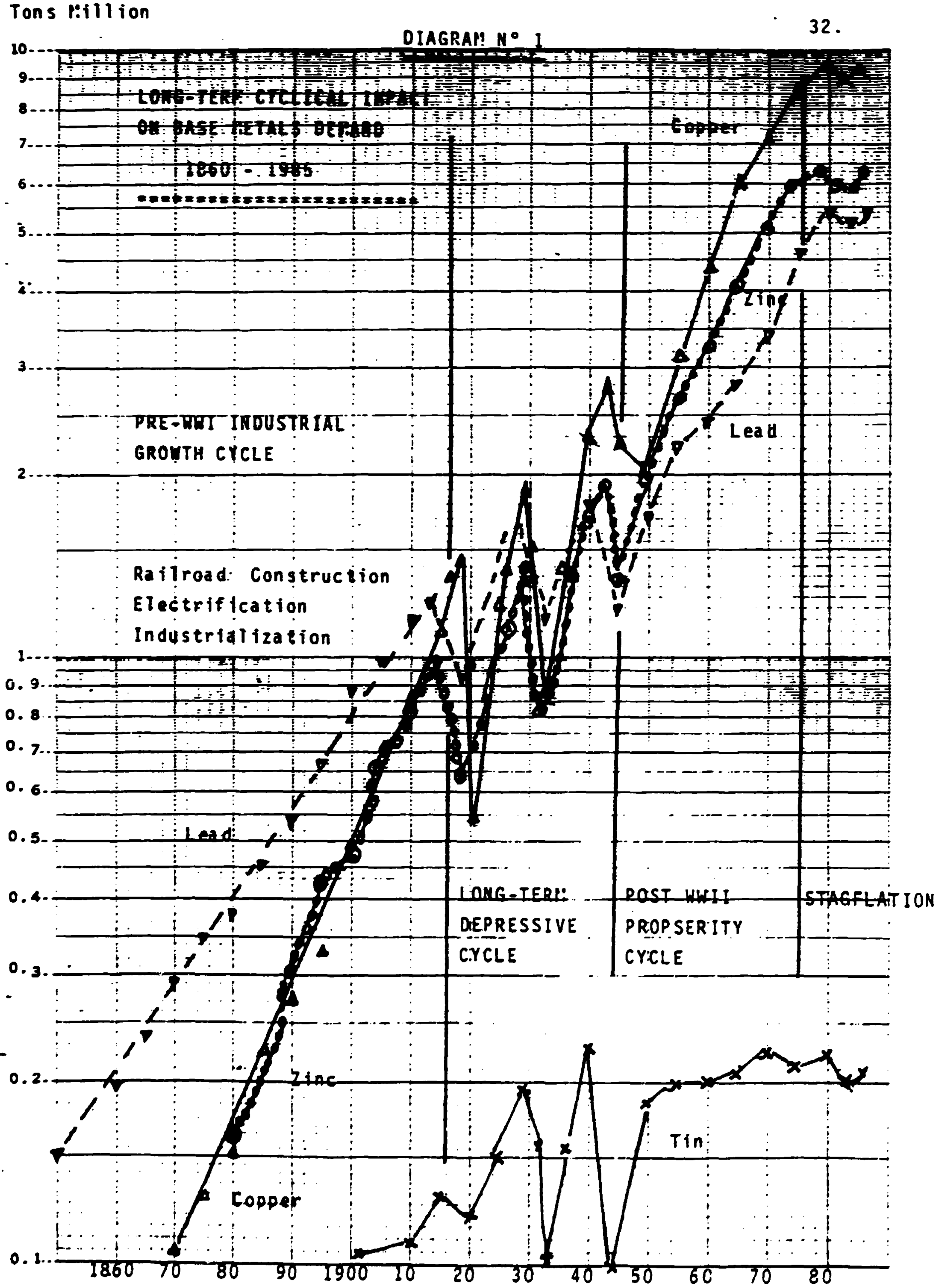
It seems that some of these processes are already in development. Oil prices have already experienced a drastic fall. The US dollar lost about 26% of its peak value in February of 1985, and the Baker Plan calls for more reasonable treatment of indebted nations. If all savings obtained through all these measures are properly oriented towards growth (but not paper wealth growth), demand for base metals may eventually increase, with which the supply-demand equation will improve and so will base metals prices. But this may require a 12 to 18 months lead time to see if all these factors work in the right direction.

Diagram 1 shows us the evolution of base metals demand during long-term cyclical changes. It indicates that cyclical depressive demand situations are characterized by drastic fluctuations, while long-term prosperity cycles are characterized by steady exponential demand growth. Since coordinates in this diagram are semi-logarithmical, straight lines indicate exponential growth. This refers to cyclical or temporary changes in demand.

On the other hand, structural changes in metals demand can be observed from changes in growth rates. For example, lead demand between 1850 and 1910 was growing at a 3.5 percent cumulative annual rate (from 150,000 tpy in 1850 to 1,000,000 tpy in 1905). However, lead demand between 1955 and 1970, another prosperity cycle, grew only at a 2.9% annual rate (from 2.16 to 3.41 m tons), which means that structural changes took place in demand.

In the case of copper and zinc, pre-WWI growth rates were similar at about 4.5% per year. But while between 1950 and 1970 copper demand continued to grow at an average cumulative rate of 5%, the zinc growth rate was reduced to only 4.3% due to structural changes in demand.

To illustrate an extreme case of structural changes in the demand for a metal, we take copper consumption in the United States between 1958 and 1984, as shown in Diagram 2. Here we correlated copper consumption on a per capita basis (when no structural changes in demand appear if per capita consumption is unchanged) with the intensity of consumption, which reflects structural changes in demand.


Tons Million
DIAGRAM N° 1
LONG-TERM CYCLICAL IMPACT
ON BASE METALS DEMAND
1860 - 1985
PRE-WWI INDUSTRIAL
GROWTH CYCLE
Railroad Construction
Electrification
Industrialization
LONG-TERM
DEPRESSIVE
CYCLE
POST WWII
PROPSERITY
CYCLE
STAGFLATION
Copper
Zinc
Lead
Tin
10
9
8
7
6
5
4
3
2
1
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
1860
70
80
90
1900
10
20
30
40
50
60
70
80

If we take the average per capita copper consumption in the USA between 1958 and 1962 as a standard, then at this fixed rate of consumption, copper demand in the USA due to the increase of population should have increased from 4.47 billion lbs in 1958 to 5.91 b lbs in 1984. On the other hand, if we take as constant the intensity of consumption which existed between 1958 and 1962 and project it to 1984 under the assumption that no structural changes took place in copper uses in the US economy, then the theoretical consumption should have increased from 4.17 billion lbs in 1958 to 10.05 b lbs in 1984.

However, as we can see from the attached Diagram 2, the actual consumption of copper in the USA followed completely different patterns: between 1958 and 1966 real consumption grew faster than expected on a fixed intensity rate, which means that intensity of demand and new copper uses were growing. Then came a transitional period, when demand started to diminish, but basically followed GNP growth until 1973. However, since the energy crisis, the economic changed so much that consumption fell off in intensity drastically. All told, in 1984 it was only 7.01 billion pounds, which is still 18.6% higher than consumption on a fixed per capita basis and 30.3% below the fixed intensity of consumption. In other words, out of a potential demand growth of 4.14 b lbs (10.05 - 5.91 b lbs) the US achieved only 1.10 b lbs of demand increase (7.01 - 5.91), which means that structural changes in consumption replaced 73.4% of the potential demand growth.

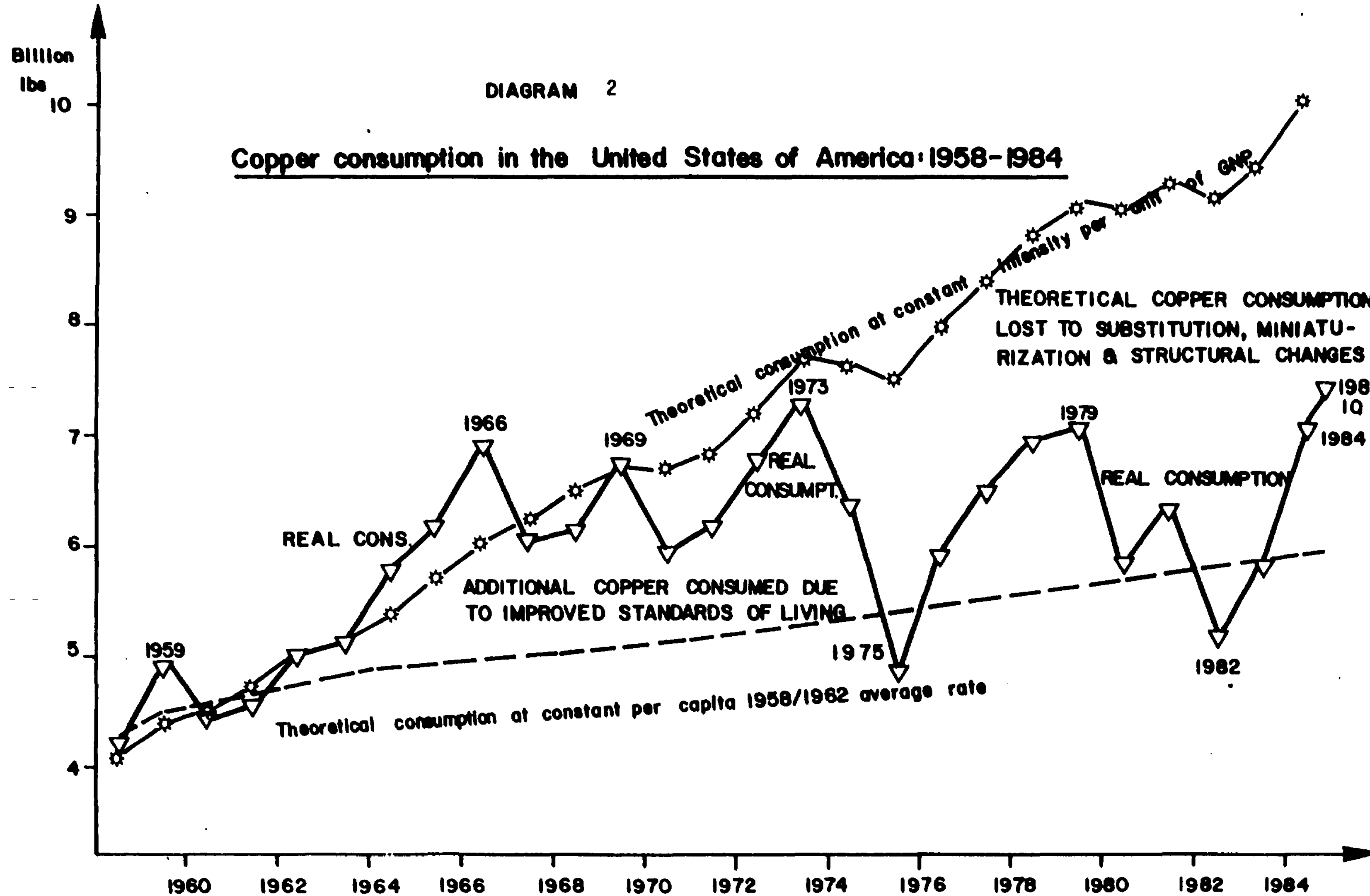


DIAGRAM 2
Copper consumption in the United States of America: 1958-1984
Billion lbs
10
9
8
7
6
5
4
1960
1962
1964
1966
1968
1970
1972
1974
1976
1978
1980
1982
1984
Theoretical consumption at constant intensity per unit of GNP
THEORETICAL COPPER CONSUMPTION LOST TO SUBSTITUTION, MINIATU-RIZATION & STRUCTURAL CHANGES
1959
1966
1969
1973
1975
1979
1982
1984
1985 1Q
REAL CONS.
REAL CONSUMPT.
REAL CONSUMPTION
ADDITIONAL COPPER CONSUMED DUE TO IMPROVED STANDARDS OF LIVING
Theoretical consumption at constant per capita 1958/1962 average rate

4.0 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.

4.1 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 roadbed 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

4.2 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 3. 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 'escends 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.

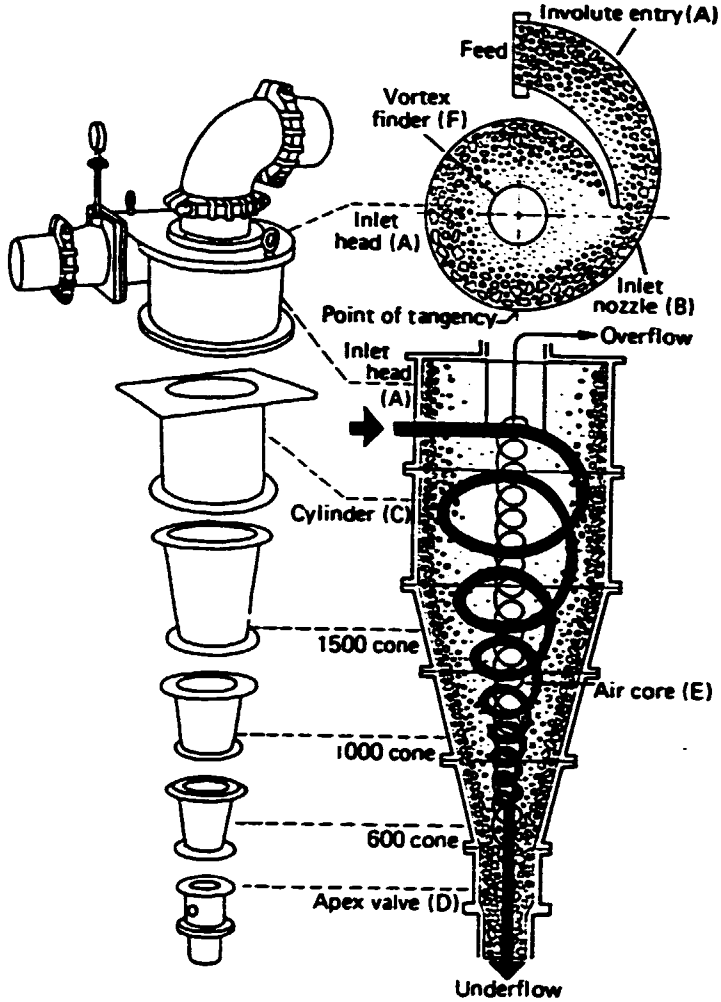


Fig. 1 : Schematic view of Webco Hydrocyclone

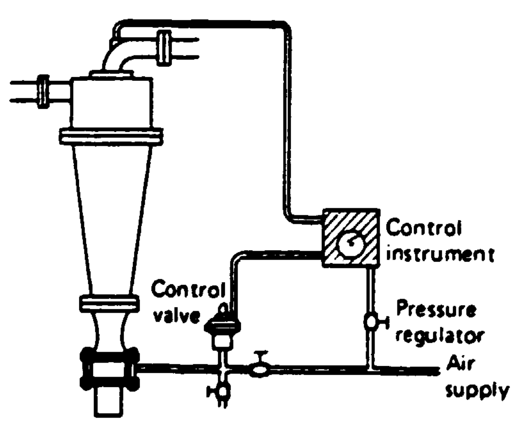


Diagram 3 : Automatic control of apex orifice for constant underflow density control.

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 4.

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 6.

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 6

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 4

CONVENTIONAL VS SEMIAUTOGENOUS GRINDING

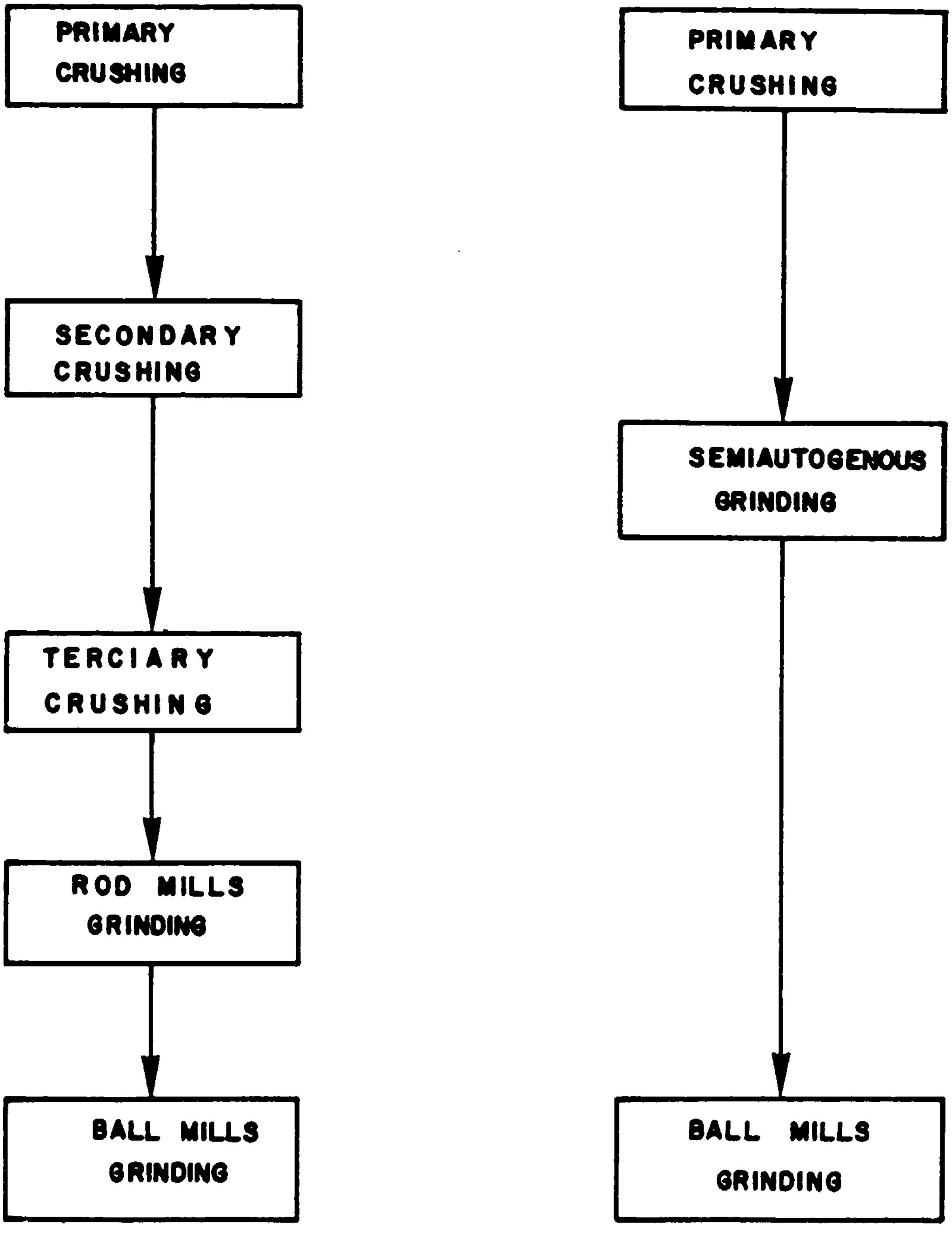


Fig. 2 and Diagram 5 give us, respectively, a cross-section of the Aerofall mill and how this system operates in a dry circuit. The feed is directly fed to the mill, which autogenously grinds by a fast rotation. A draft fan, located above the system, drives the finely ground material through a vertical 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 6.

In Diagram 7, a typical setup for a semiautogenous grinding flowsheet is given, as is normally tested in research laboratories on different ores, while in Diagram 8 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

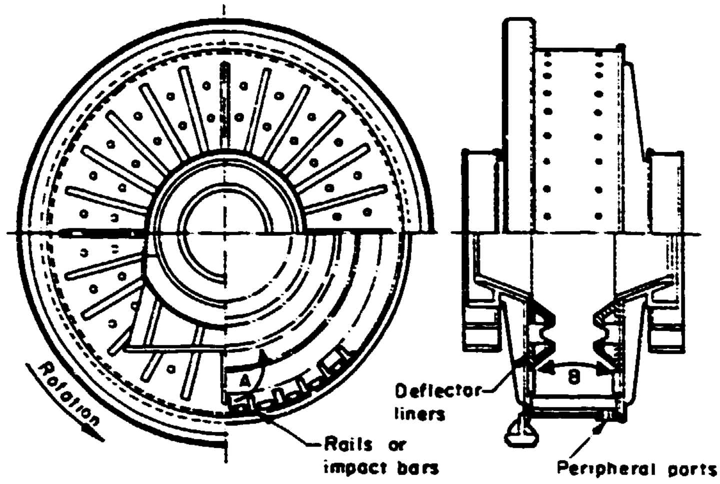


Fig. 2 : Cross-section of an Aerofall mill

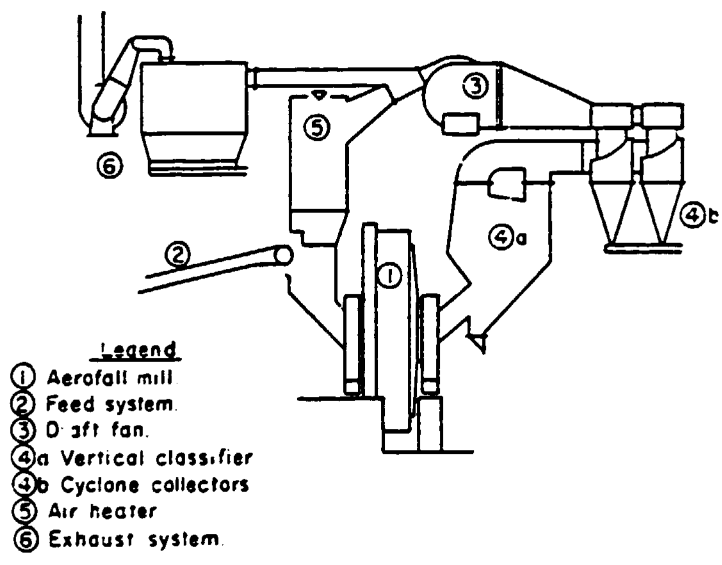


Diagram 5 : Operation of an Aerofall system

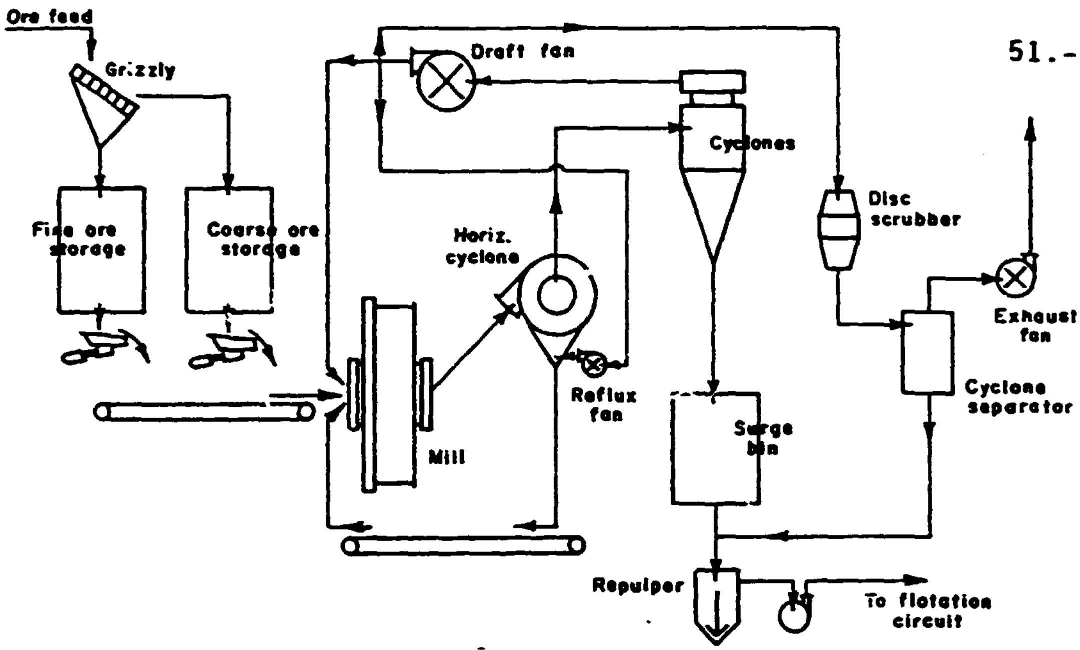


Diagram 6 : Flowsheet of a closed circuit grinding system

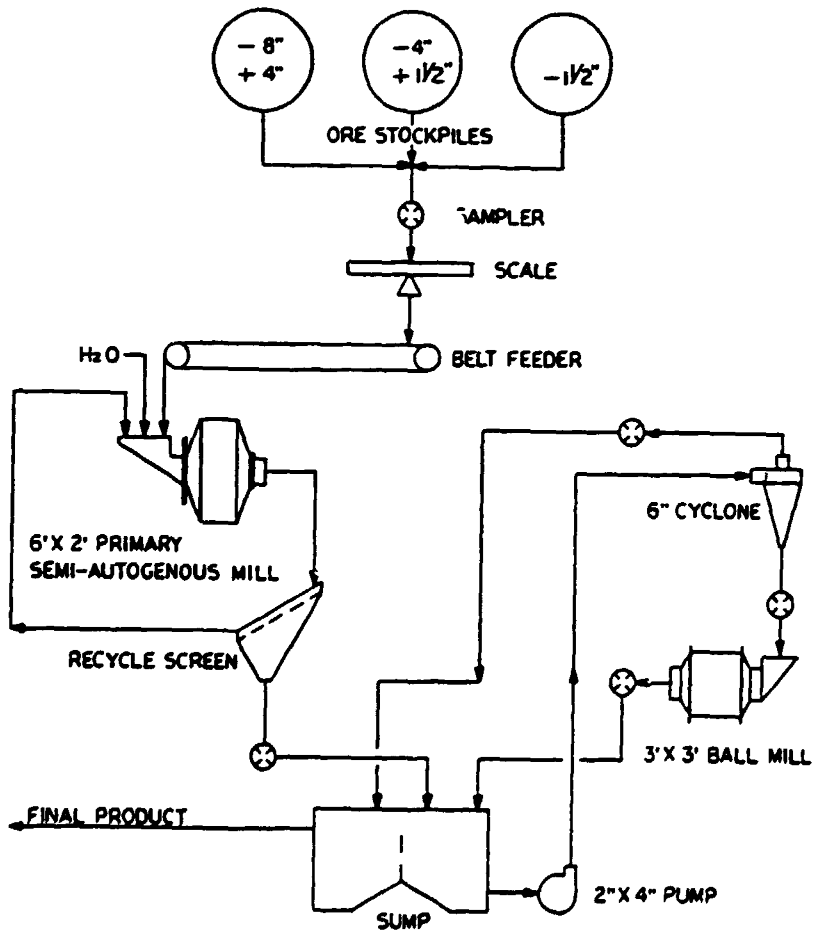


Diagram 7 : Flowsheet of a Semi-Autogenous Mill Operation

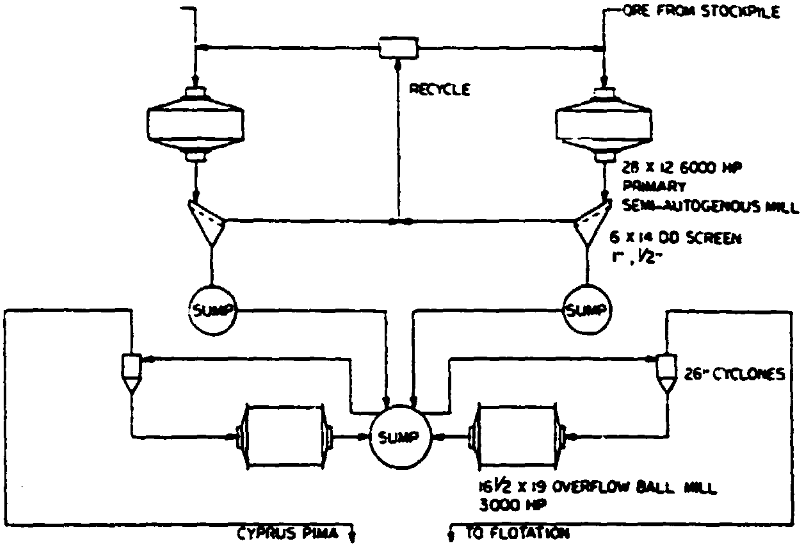


Fig. 8 : Semi-Autogenous Grinding flowsheet at Pima Mine

4.3 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.

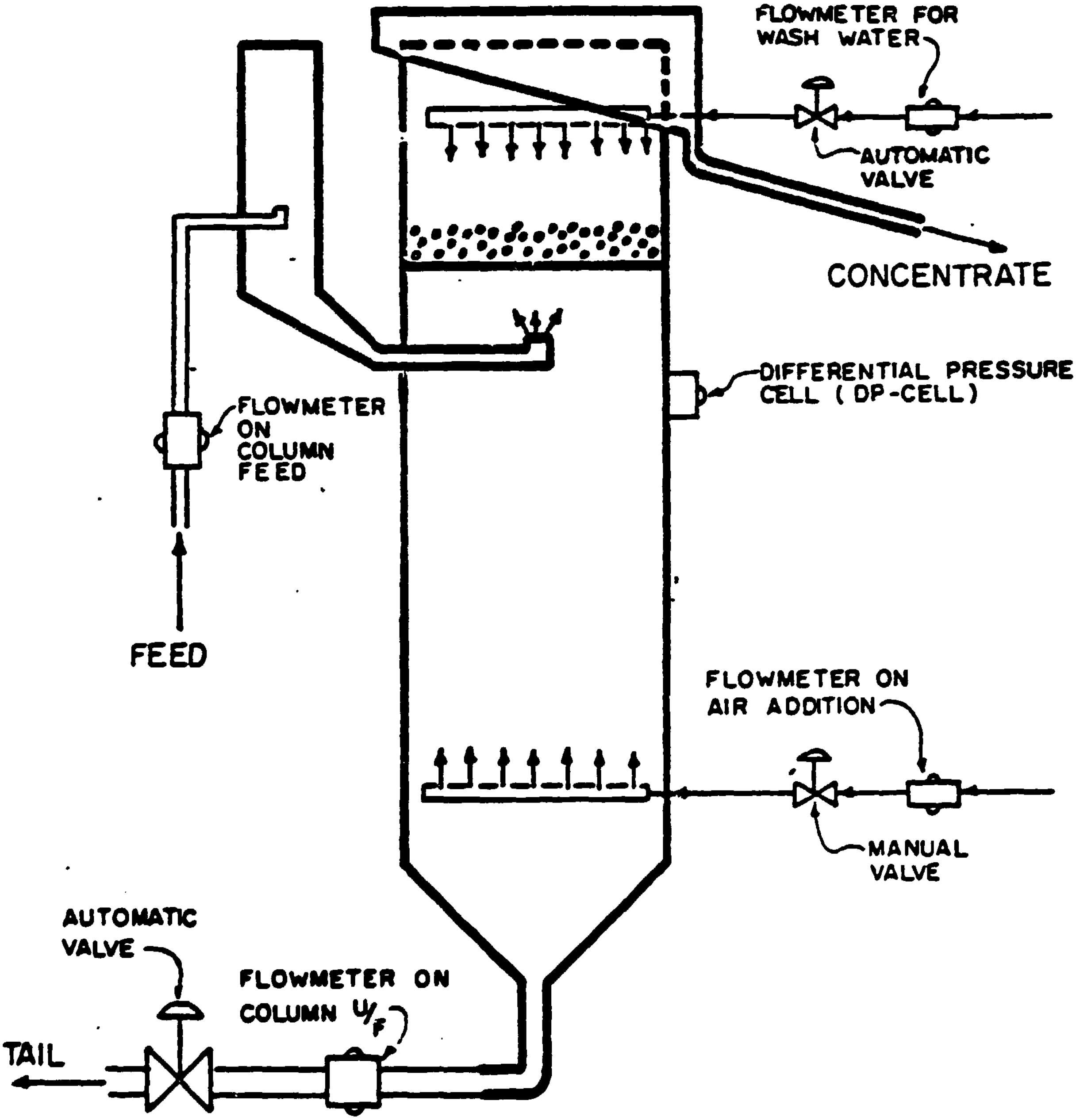
However, a technologically new system for replacement of traditional flotation cells was developed in these last years in Canada. This is the so called column flotation. The idea offers a number of advantages in the separation of different minerals, particularly in the copper industry and in the area 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



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.

4.4 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.

4.5 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.

5.0 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.

5.1 COLUMN FLOTATION

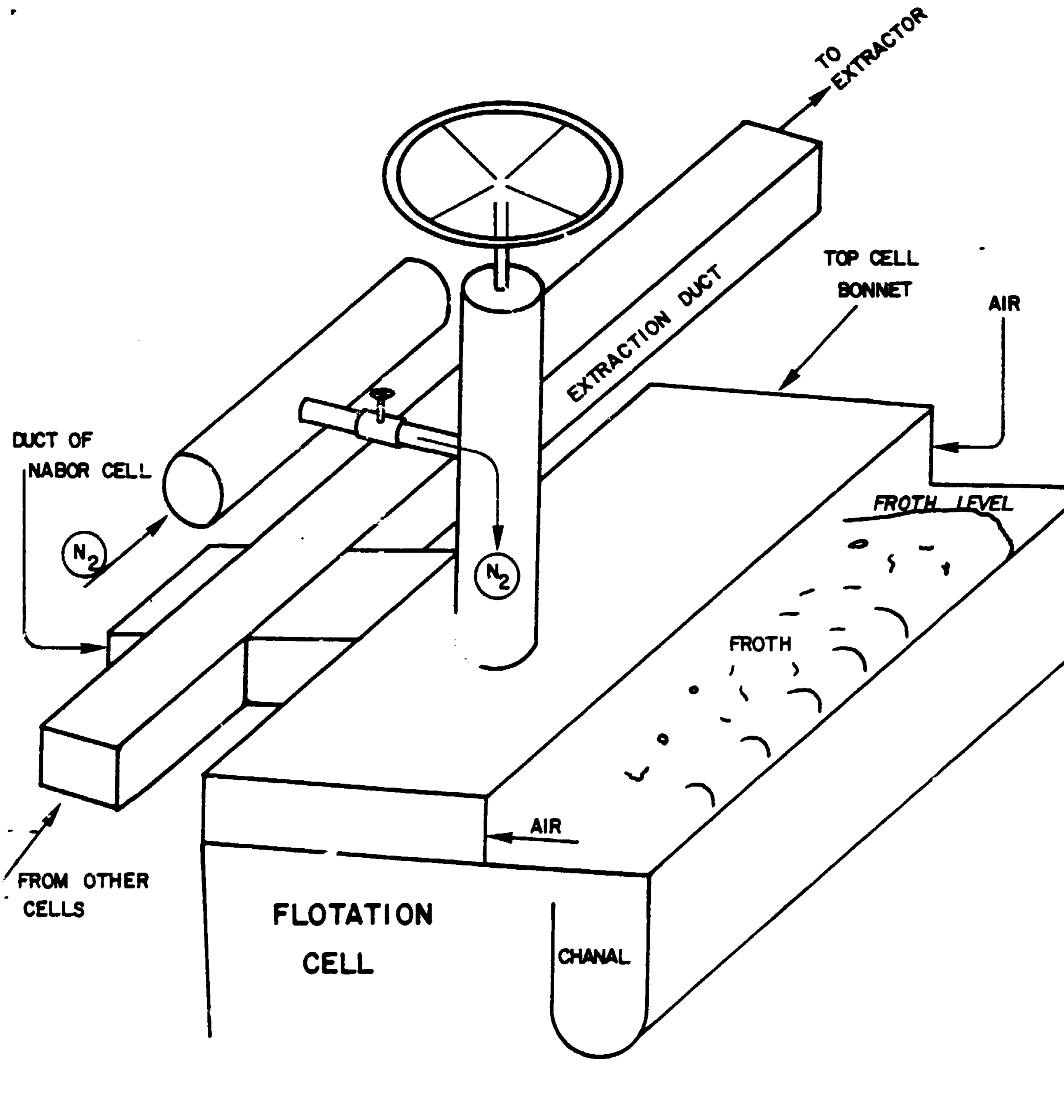
World molybdenum production today amounts to about 200 - 210 million pounds per year, of which roughly 40% is produced by primary molybdenum mines while the other 60% is recovered as a byproduct of copper. These 120 million pounds of molybdenum per year are worth roughly $ 350 million at present molybdenum 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 9, 10 and 11, 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 9

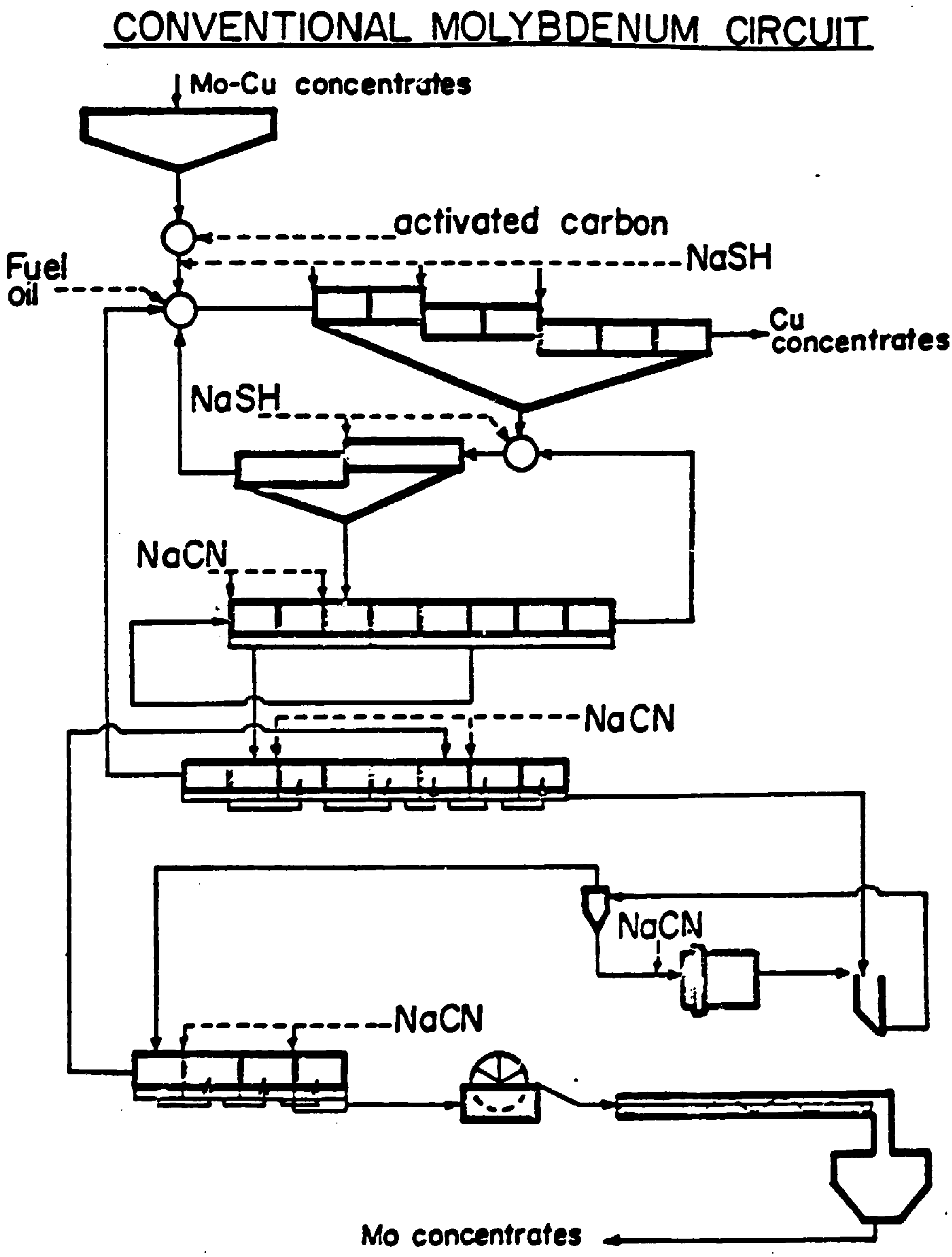

CONVENTIONAL MOLYBDENUM CIRCUIT
Mo-Cu concentrates
activated carbon
NaSH
Fuel oil
Cu concentrates
NaSH
NaCN
NaCN
NaCN
NaCN
Mo concentrates

DIAGRAM 10

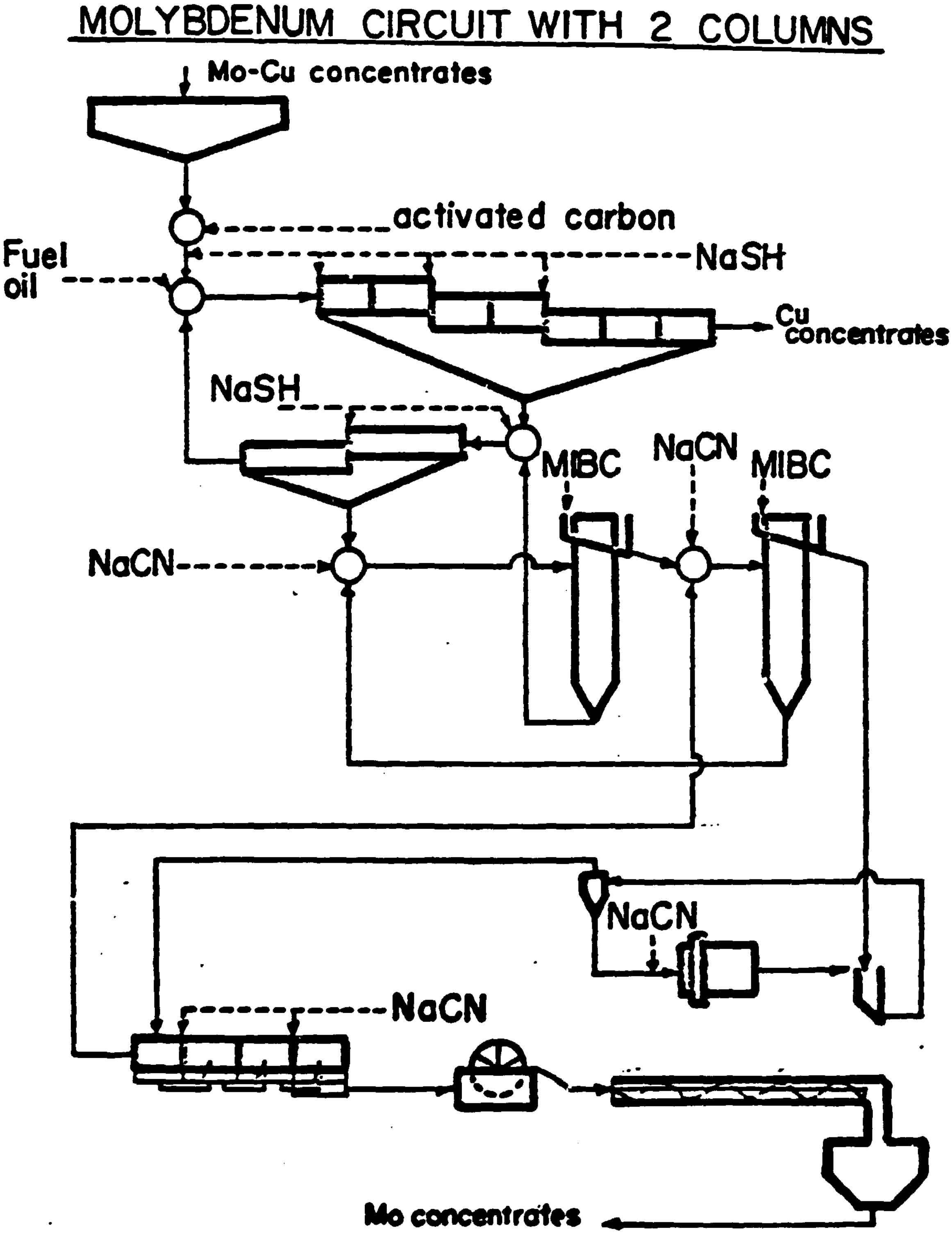
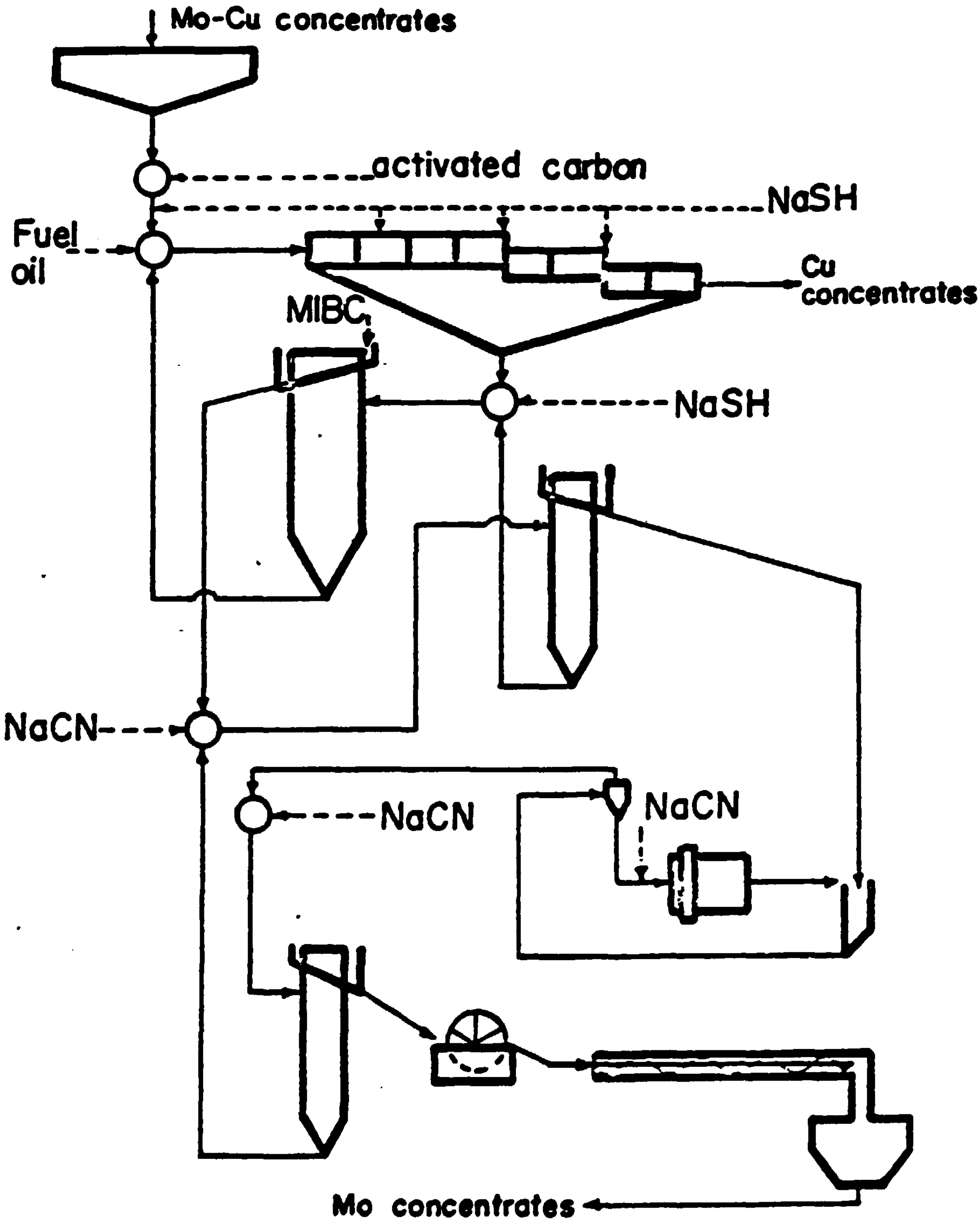

MOLYBDENUM CIRCUIT WITH 2 COLUMNS
Mo-Cu concentrates
activated carbon
Fuel oil
NaSH
Cu concentrates
NaSH
MIBC
NaCN
MIBC
NaCN
NaCN
NaCN
Mo concentrates

DIAGRAM 11

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.

5.2 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 <u>Thiobacillus ferrooxidans</u>, 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:

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

<u>Thiobacillus ferrooxidans</u> 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 <u>Thiobacillus ferrooxidans</u> catalyze the second reaction and accelerate the whole leaching process.

Other bacteria such as <u>Thiobacillus thiooxidans</u> 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:

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

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

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.

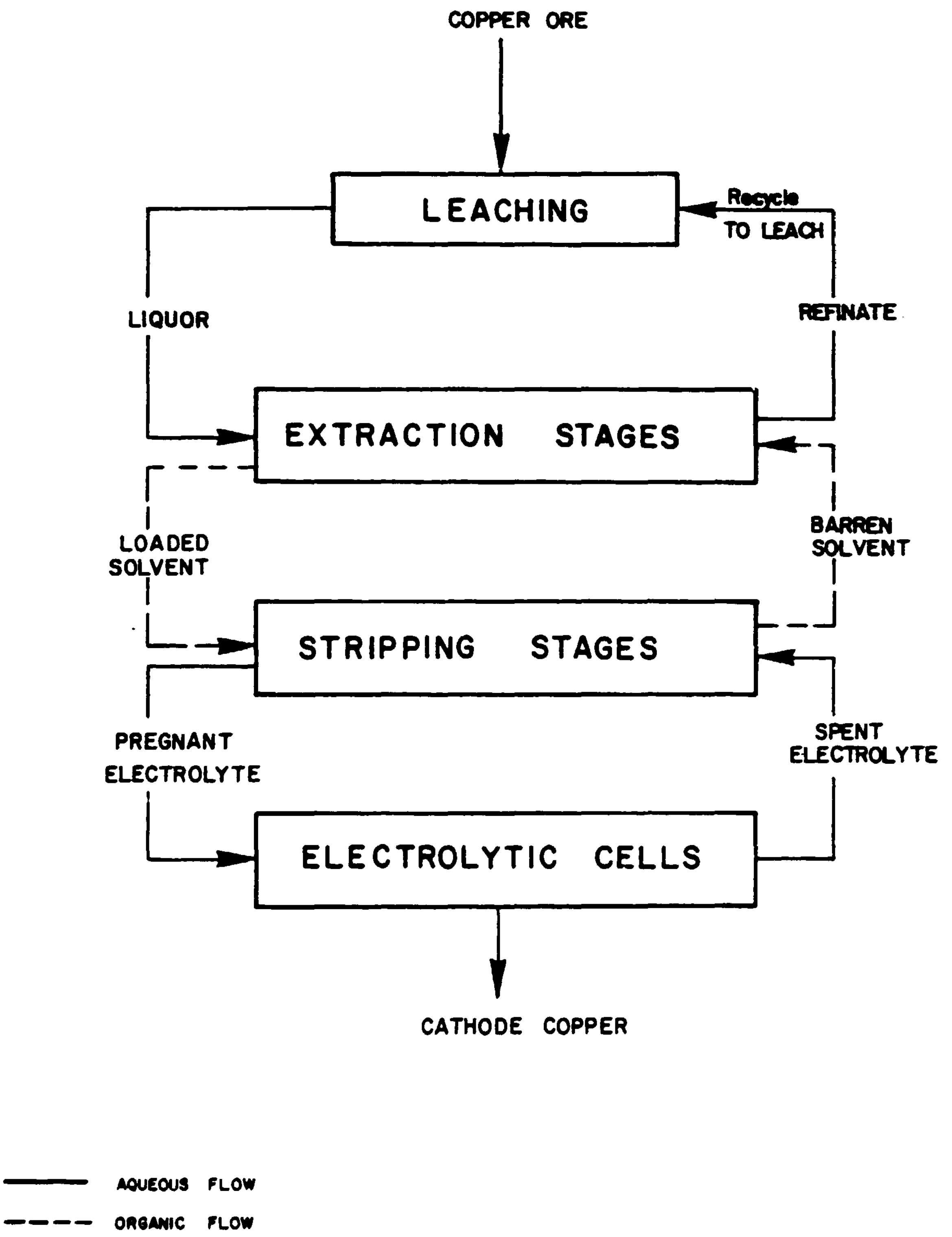
5.3 SX/EW PROCESS

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 12, 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 12: SCHEMATIC FLOWSHEET FOR EXTRACTING COPPER BY SX/EW PROCESS.



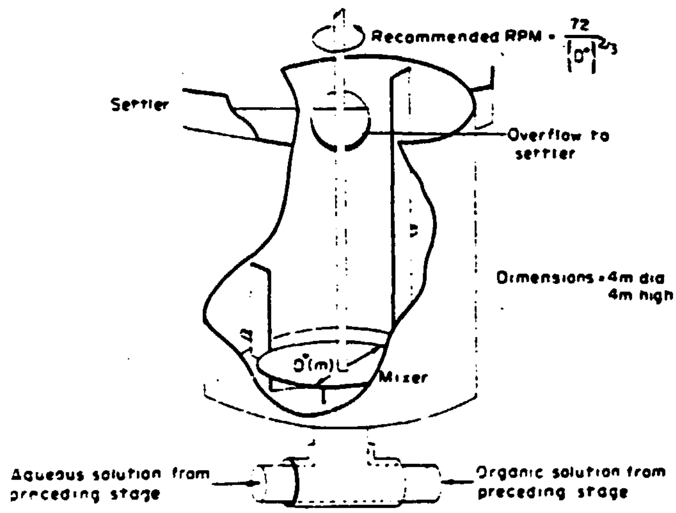
which consist of a mixer and a settler. In the mixers, the pregnant solution is mixed with an organic chemical of the chelating type, which in the case of copper solvent extraction is known under the names of LIX reagents (produced by General Mills Chemicals) and KELEX (produced by Ashland Chemical Company).

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 13.

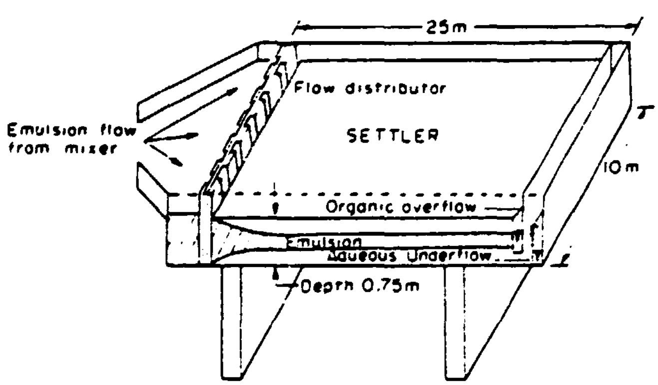
The loaded solvent normally contains about 2 grs/l of copper, while the refinate recycled to leach contains between 0.1 and 0.2 grs/l of copper and 3 to 5 grs/l of sulphuric acid. In this way, copper recovery at this stage is about 80 to 90

Figure 5



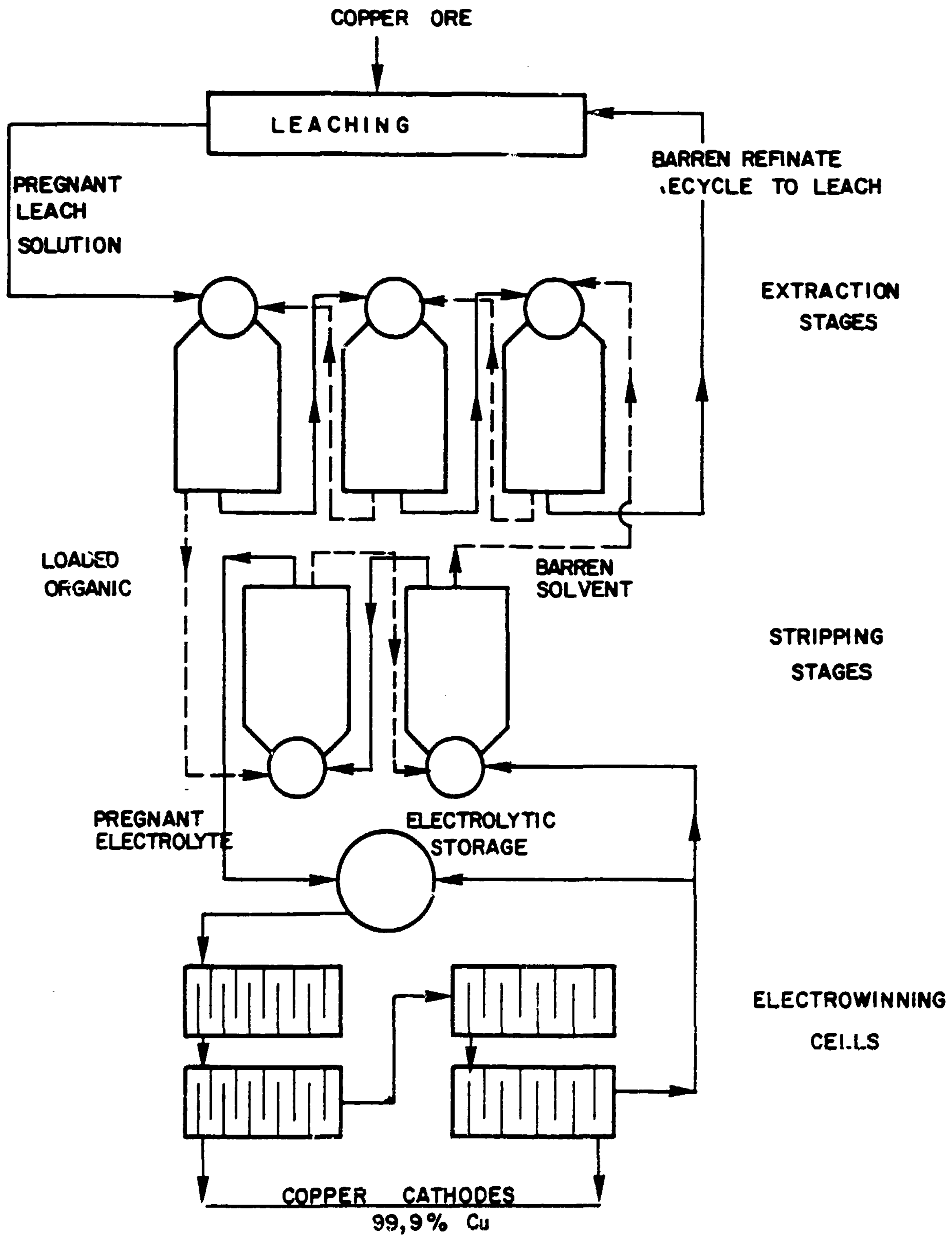
(a) Mixer

Figure 6



(b) Settler

DIAGRAM 13: SX/EW PROCESS PLANT



AQEOUS FLOW

ORGANIC FLOW

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 13, 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 7, 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 7

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

5.4 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 14. 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 15.

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.

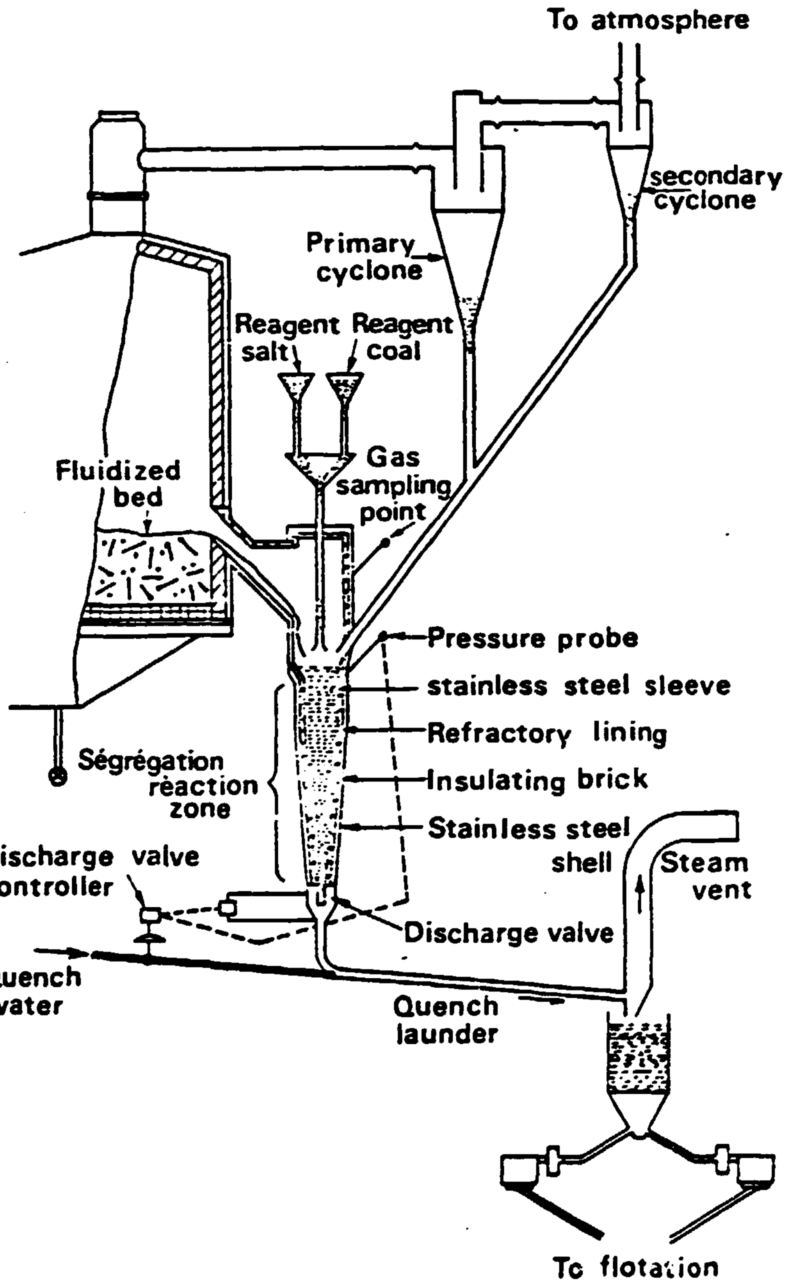
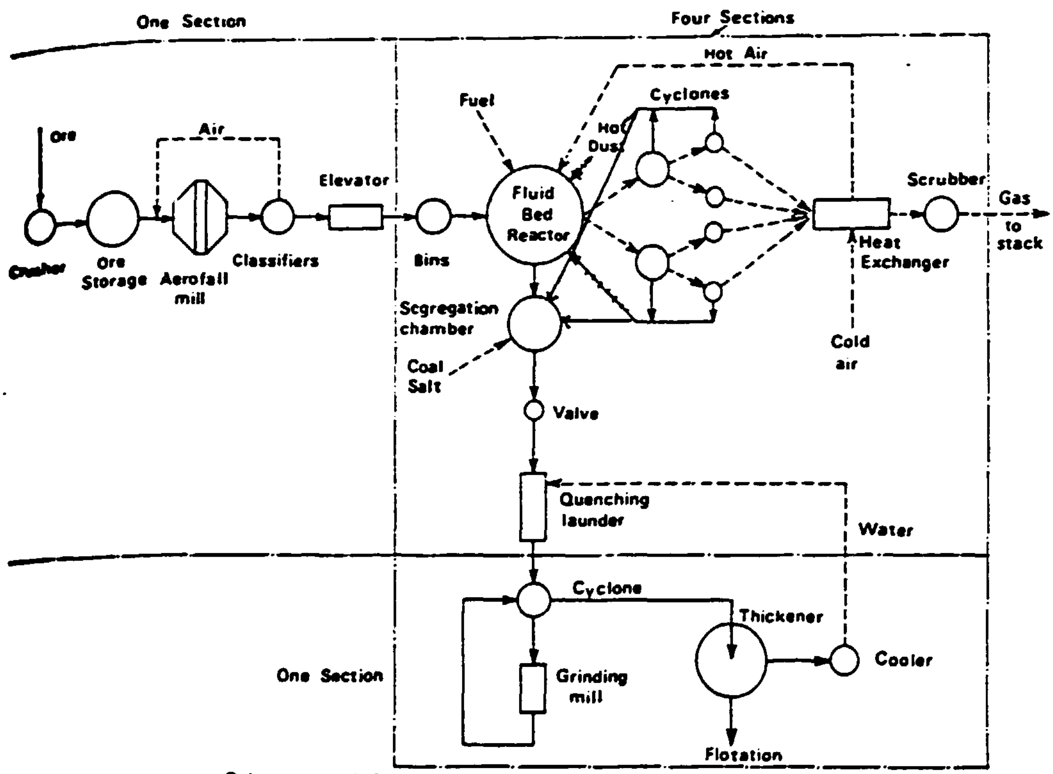


Diagram 14 : TORCO PROCESS UZED IN ZAMBIA



Diagr. 15 Flow sheet of TORCO plant at Akjoujt (Mauritania).

5.5 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 16, we can follow the advances of copper smelting technology in the 20th century along with the increase in copper consumption and production.

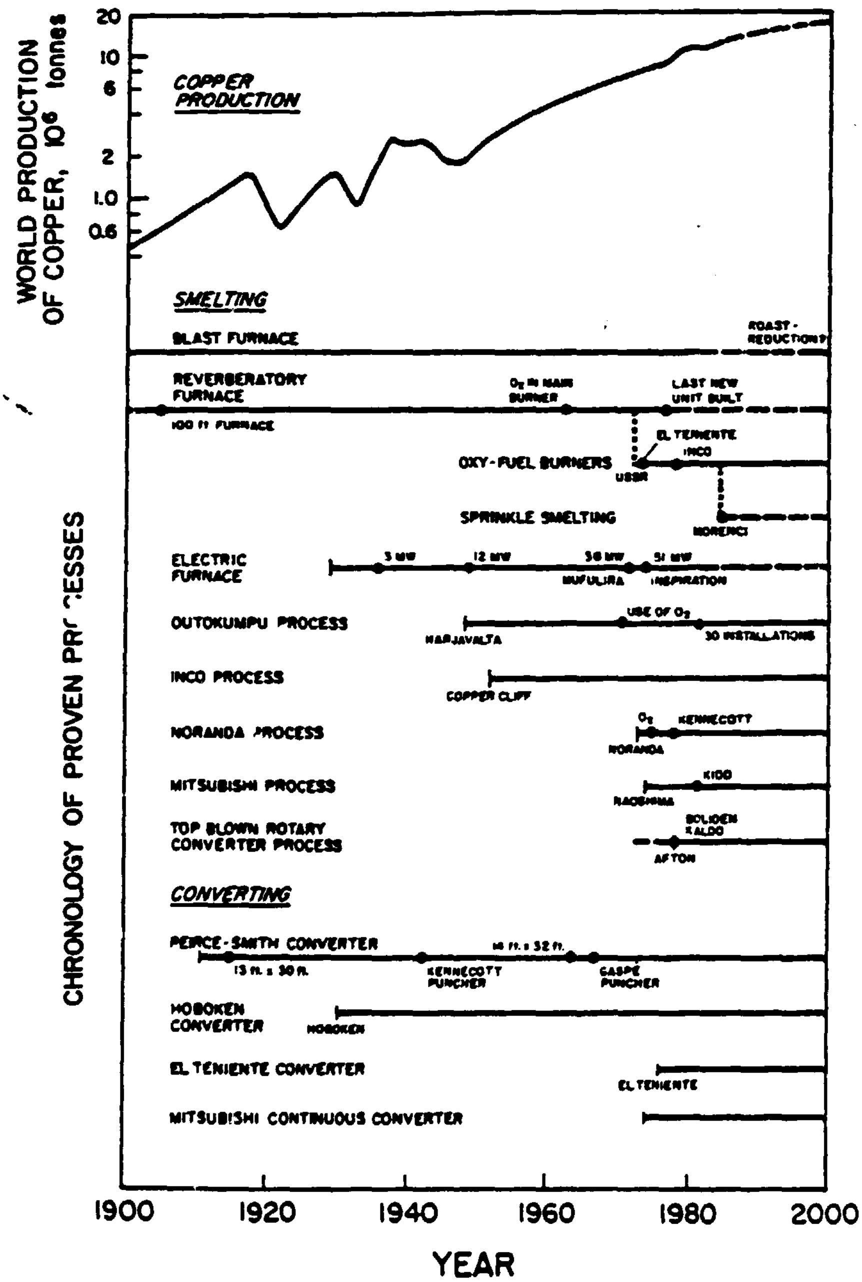
The old blast furnace which served so much for recovery of copper from oxide and then sulphide ores has now given way to roast reduction. Reverberatory furnaces,which permit smelting of sulphide concentrates, received an important innovation in the early 1960's when oxygen enriched air started to be 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 16


WORLD PRODUCTION OF COPPER, 10^6 tonnes
20
10
6
2
1.0
0.6
COPPER PRODUCTION
SMELTING
BLAST FURNACE
ROAST-REDUCTION?
REVERBERATORY FURNACE
100 ft FURNACE
LAST NEW UNIT BUILT
EL TENIENTE
INCO
OXY-FUEL BURNERS
USSR
SPRINKLE SMELTING
MORENCI
ELECTRIC FURNACE
12 MW
36 MW
51 MW
MUFULIRA
INSPIRATION
OUTOKUMPU PROCESS
HARJAVALTA
USE OF O_2
30 INSTALLATIONS
INCO PROCESS
COPPER CLIFF
NORANDA PROCESS
O_2
KENNECOTT
NORANDA
MITSUBISHI PROCESS
KIDO
NAOSHIMA
TOP BLOWN ROTARY CONVERTER PROCESS
BOLIDEN
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
CHRONOLOGY OF PROVEN PROCESSES
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.

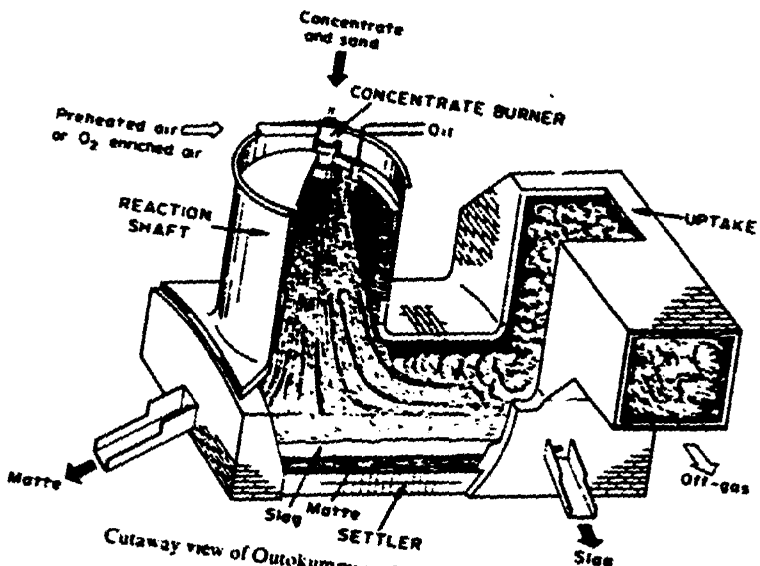
5.51 Outokumpu Process - The attached figures 7 and 8 give us cutaway views of Outokumpu and Inco flash smelting furnaces. Outokumpu is an older process and the first which introduced the flash principle. It originally operated on preheated air but then also introduced oxygen enriched air in order to 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.

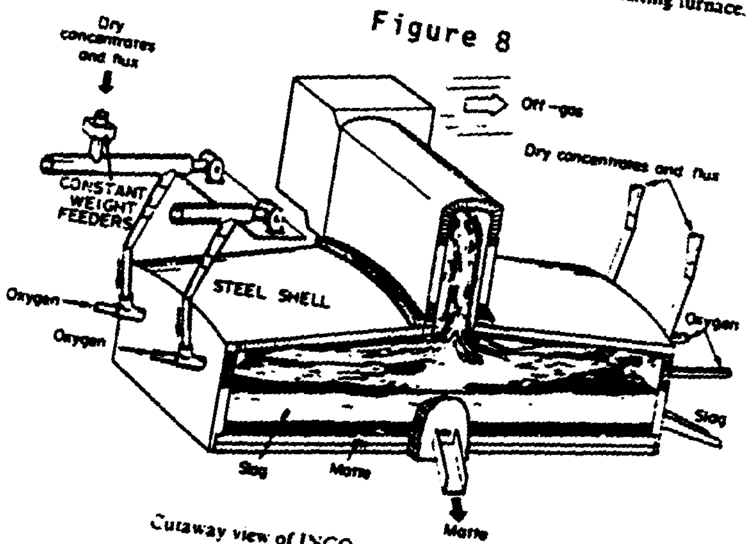
5.52 Inco Process also combines roasting, melting and partial converting in one autogenous operation. The use of technical oxygen (about 96% purity) instead of air to combust labile sulphur and iron sulphide in the feed eliminates the need for heating up nitrogen, which is the major cause of low fuel

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.

5.53 <u>Noranda Process</u> 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 17.

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)

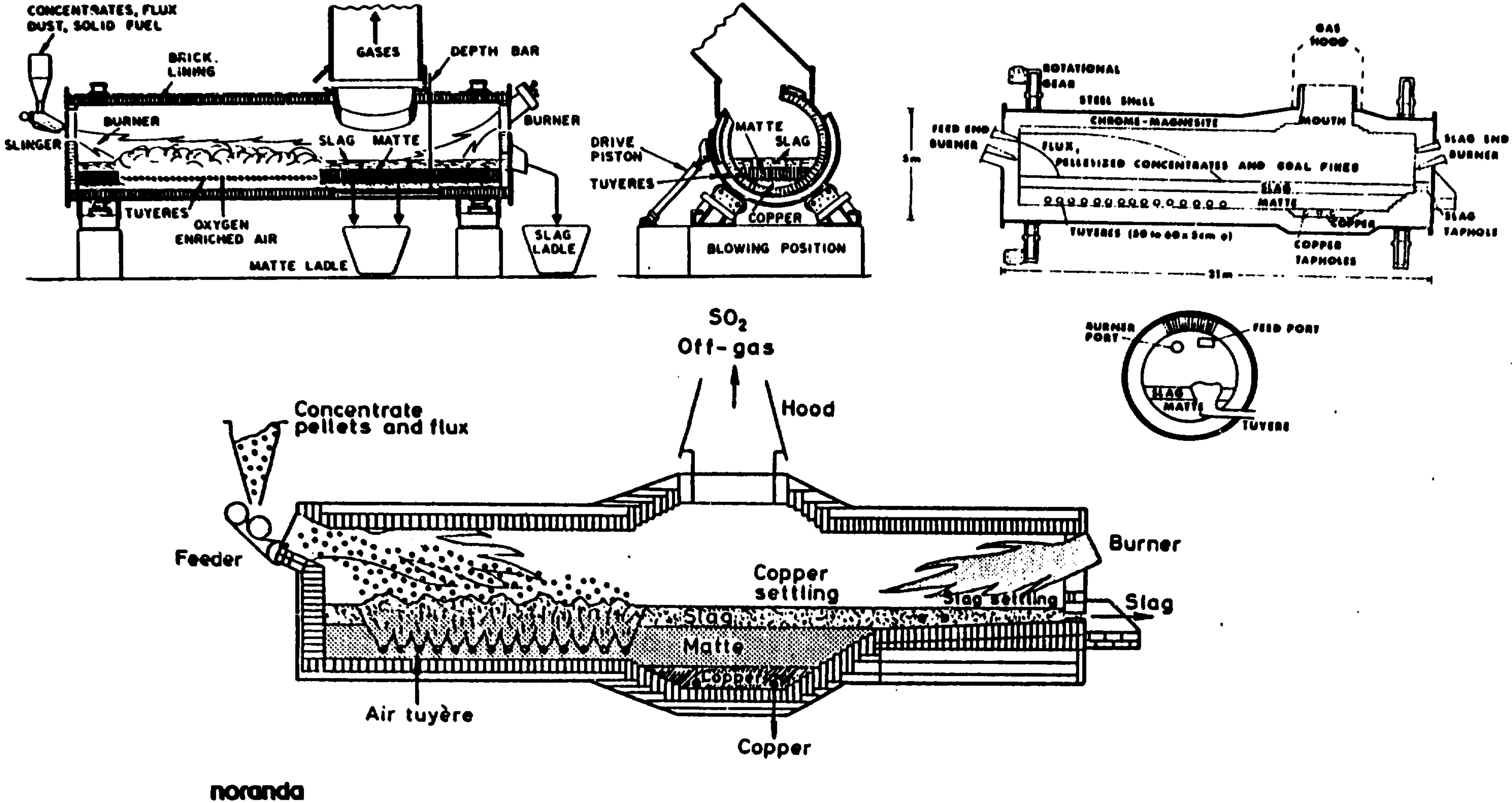
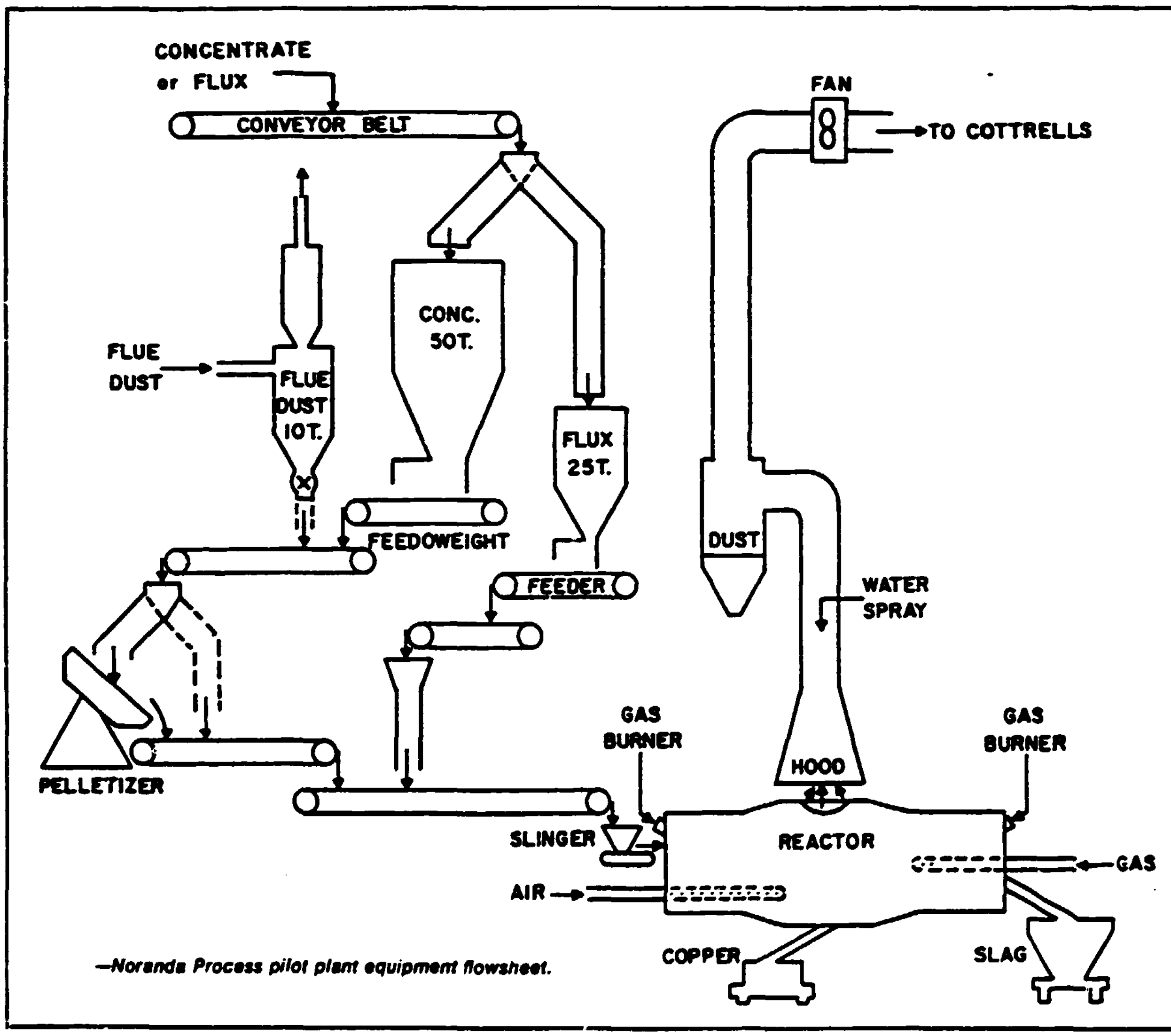


Fig.9 - Schematic drawing of Noranda Process reactor.

DIAGRAM 17



—Noranda Process pilot plant equipment flowsheet.

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

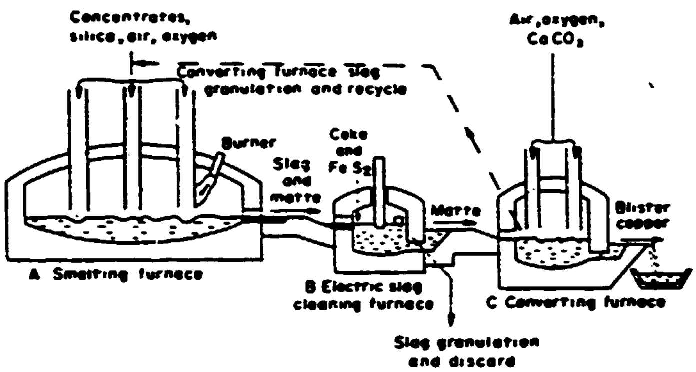
The process is relatively insensitive to changes in grade of feed material, including secondaries, and is very flexible for operation. The basic controls are reduced to oxygen/concentrate ratio and flux/concentrate ratio to maintain the Fe/SiO2 ratio in the slag at 1.8/1.

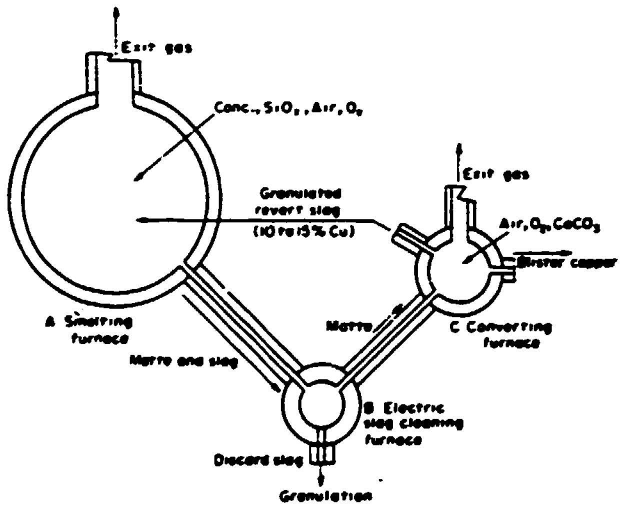
5.54 <u>Mitsubishi Process</u> - The Mitsubishi continuous copper smelting process was developed in Japan at the Naoshima smelter and apart from Naoshima is used today at the Kidd Creek mine. Its principle of operation is shown in Diagram 18.

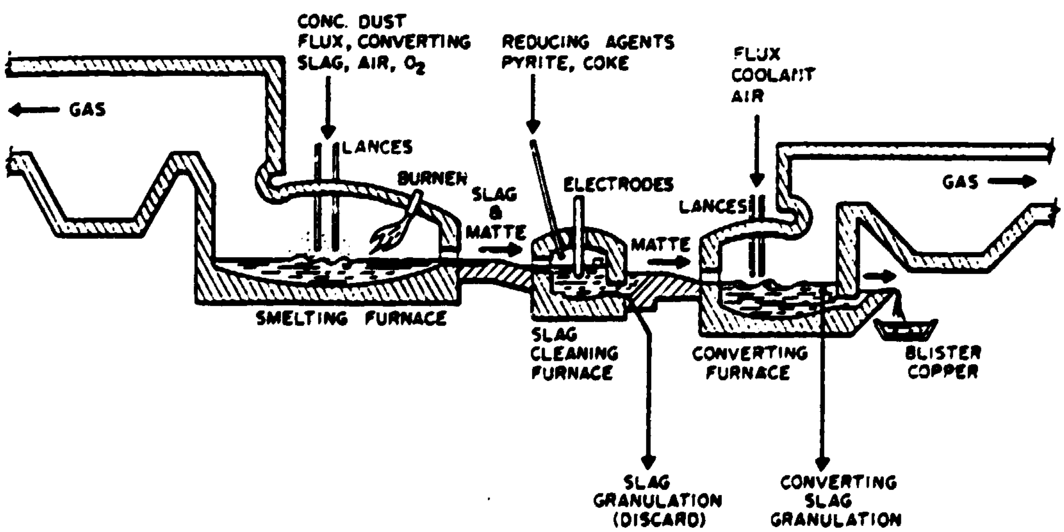
This is a multistep process which produces blister directly from concentrates, using for this three interconnected furnaces and thus dispenses with the conventional converter aisle and transport of materials by crane.

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% SO2, respectively.

DIAGRAM 18







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.

5.55 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 19 and 20, 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.

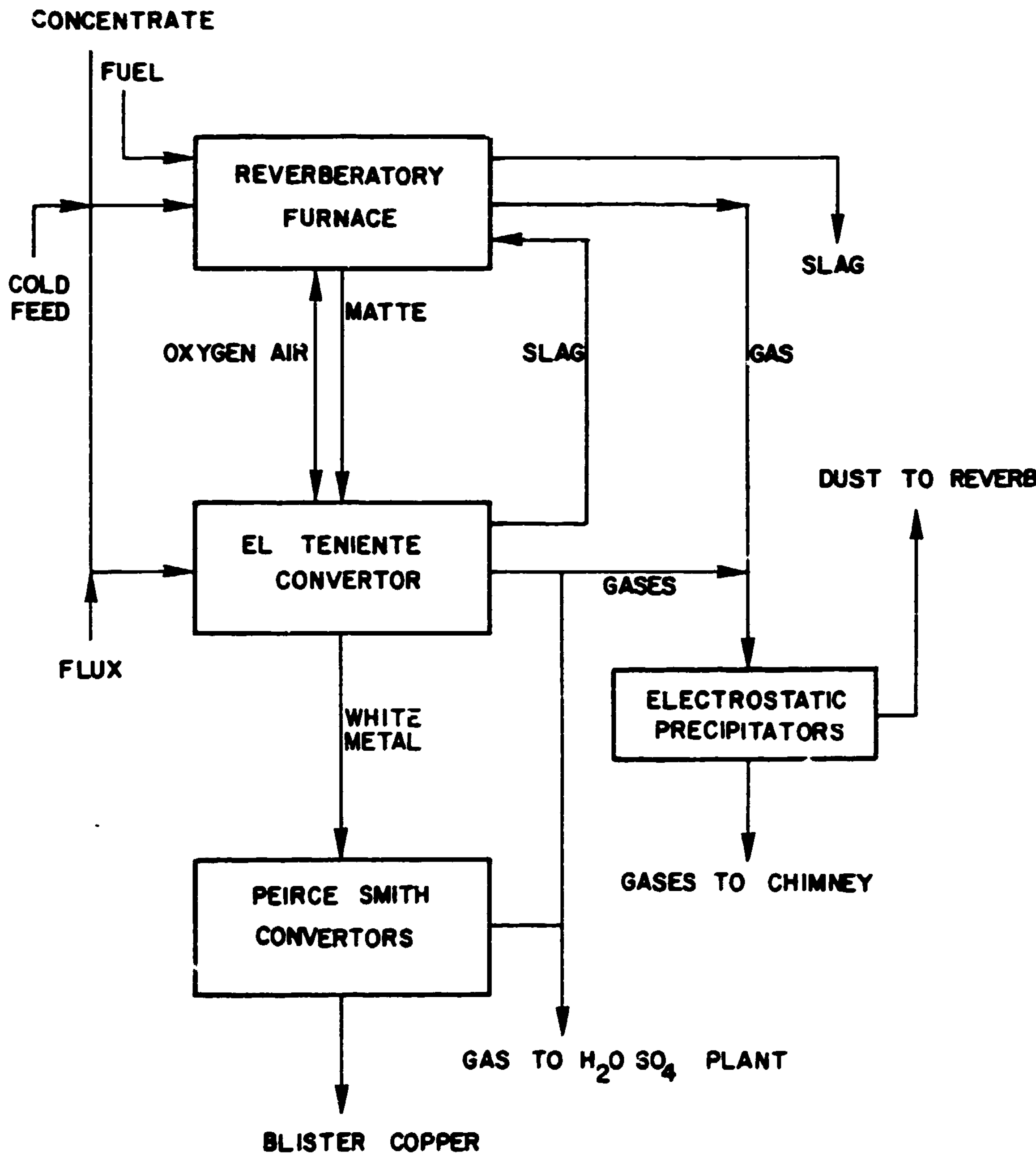
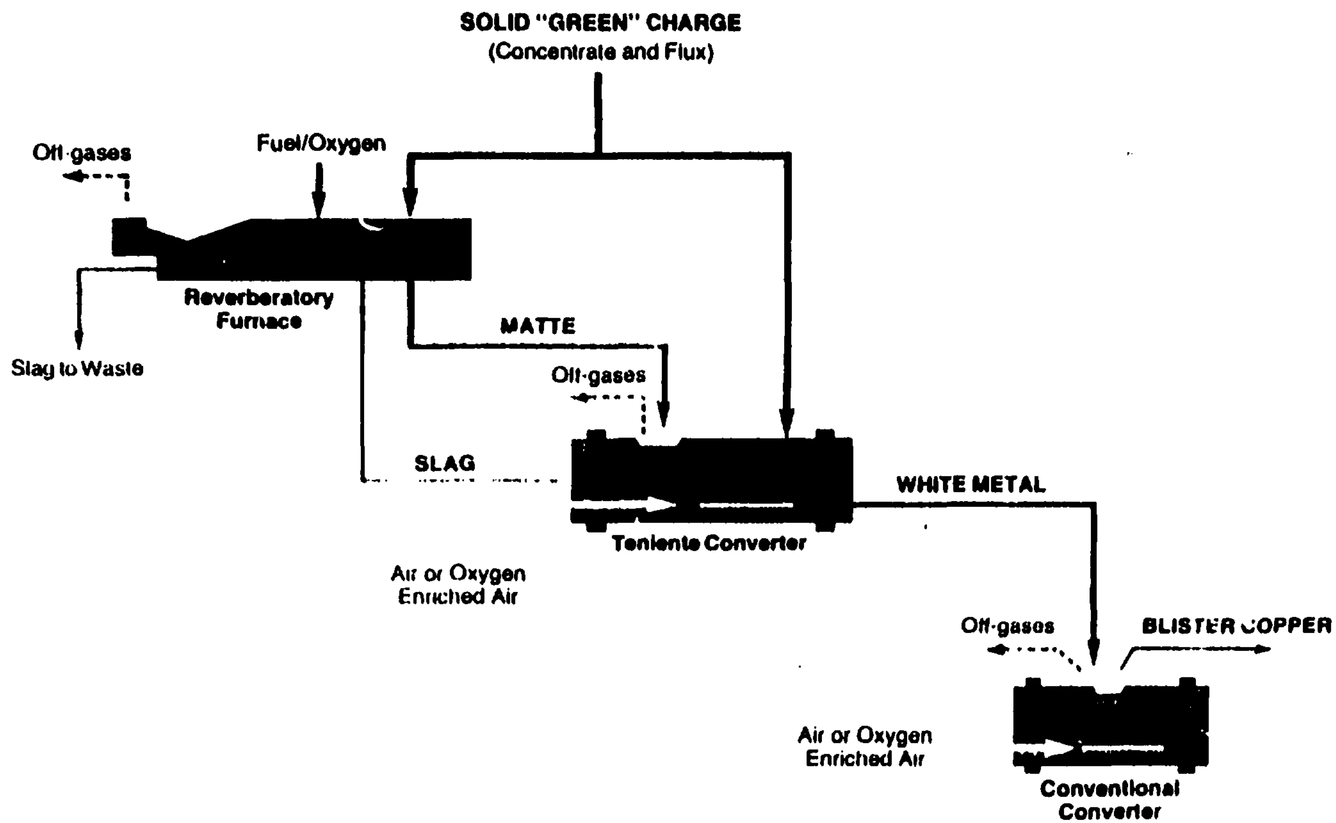


DIAGRAM 19: SCHEMATIC PRESENTATION OF EL TENIENTE PROCESS IN CALETONES

DIAGRAM 20



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

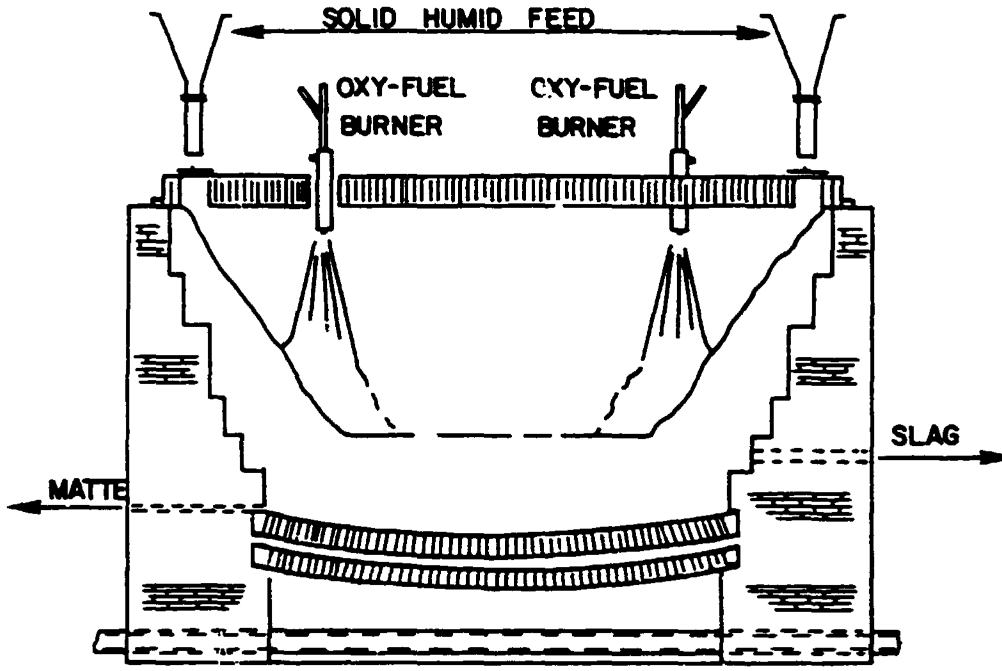


FIG. 10 POSITION OF OXY-FUEL BURNERS IN REVERBERATORY FURNACE.

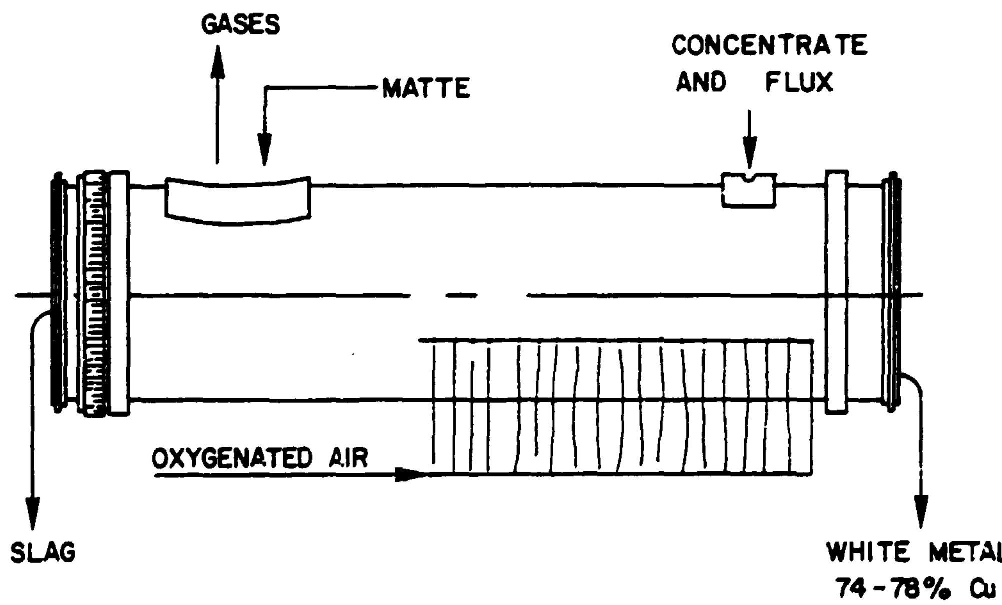


FIG. 11 EL TENIENTE CONVERTOR

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.

5.56 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 95% 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 8).

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 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 8

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 9 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 9

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.

6.0 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.

6.1 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.

6.2 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 fluo-solid 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 21 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.

6.3 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

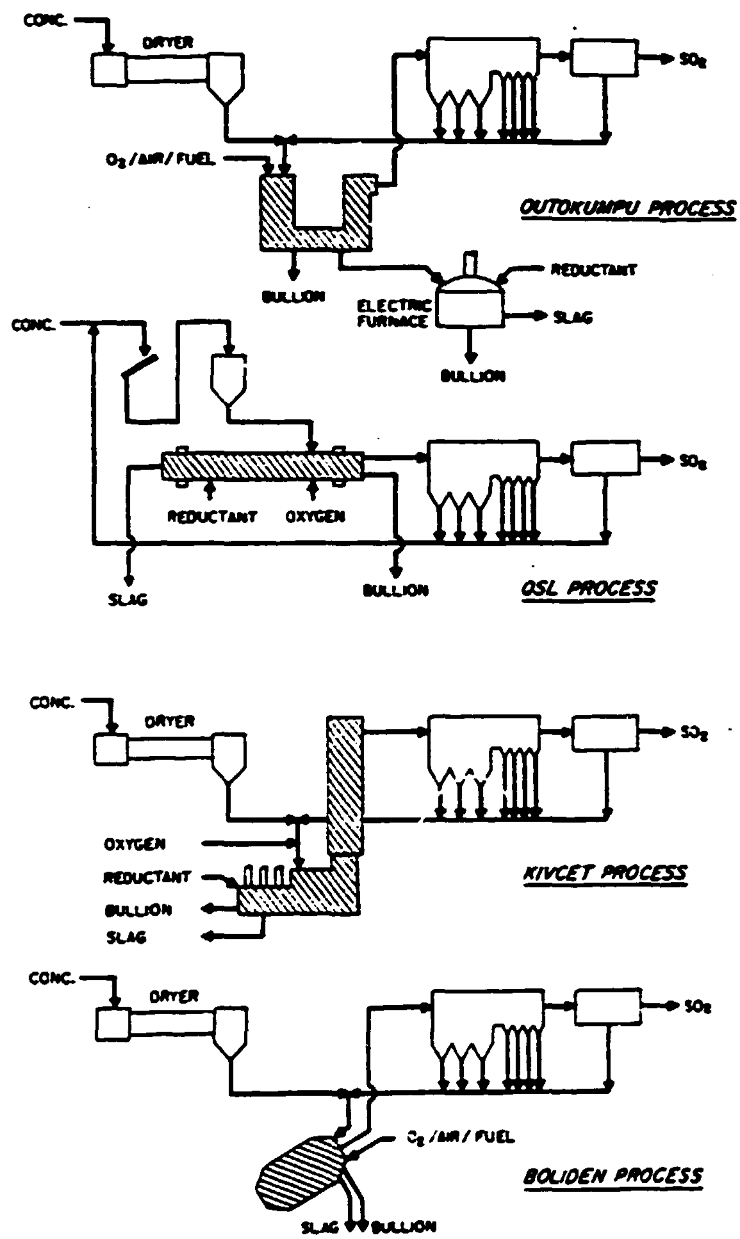


Diagram 21 : Direct Lead Smelting Processes

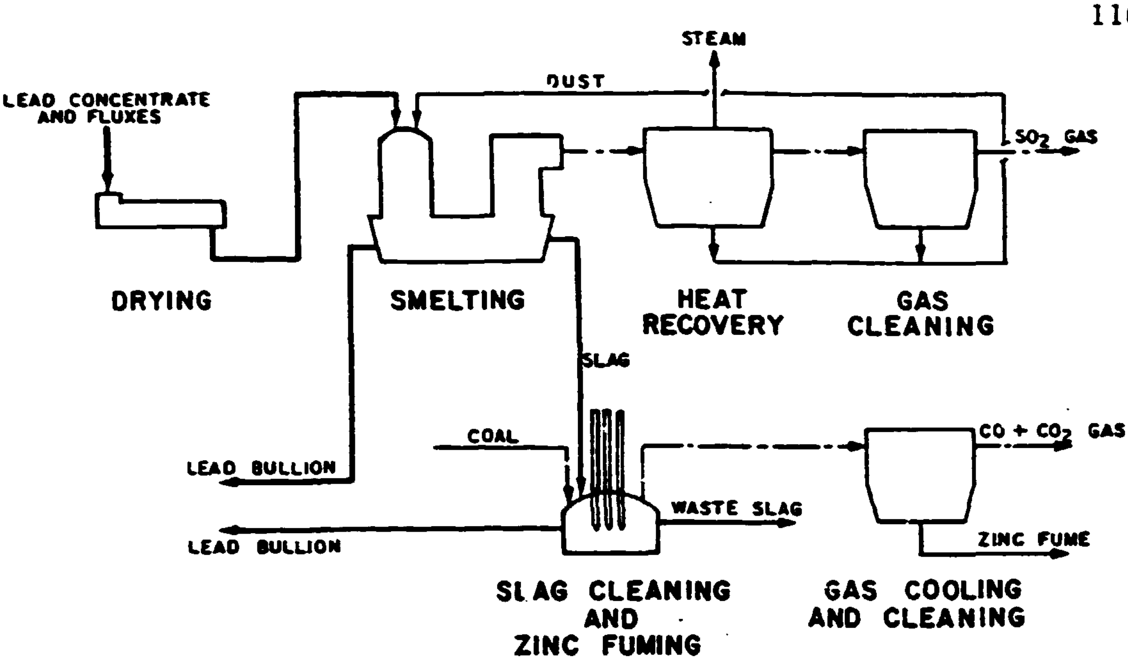


FIGURE 12. OUTOKUMPU LEAD FLASH SMELTING PROCESS

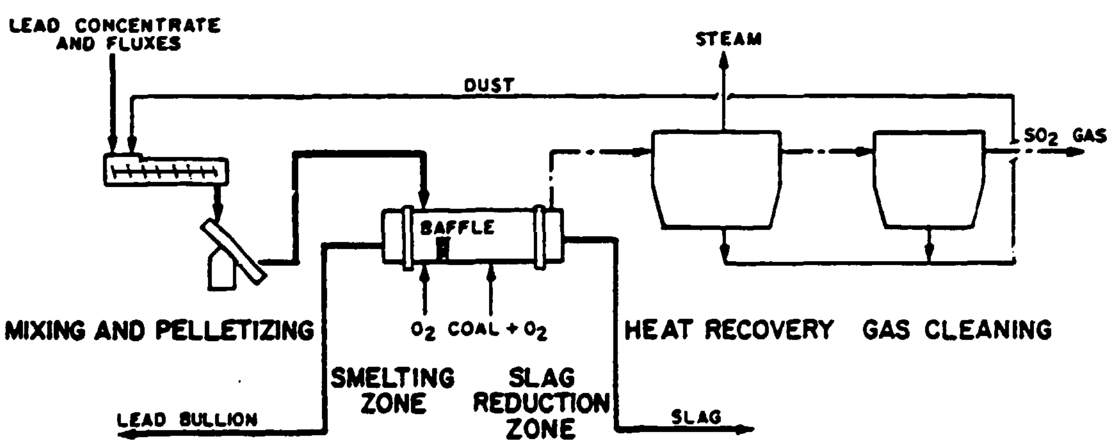


FIGURE 13. QSL LEAD SMELTING PROCESS

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.

6.4 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 22 and 23. 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.

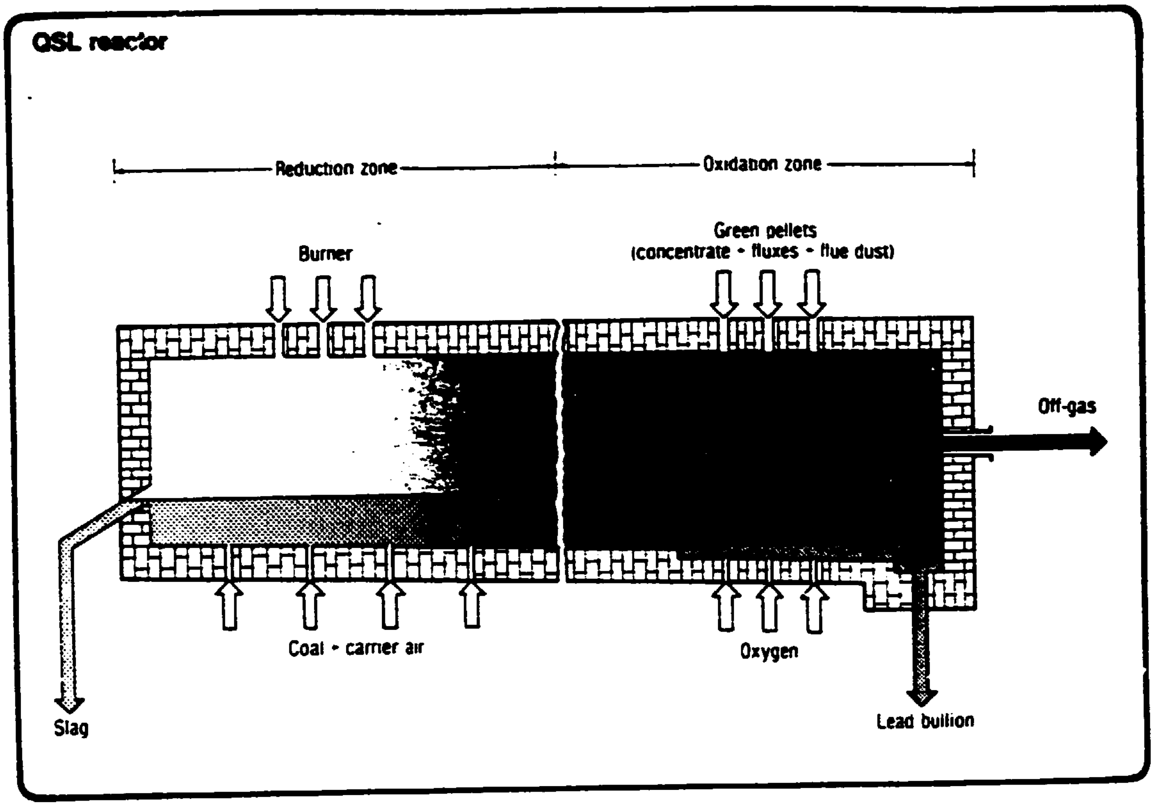


Fig. 14 QSL kiln-reactor for continuous lead smelting

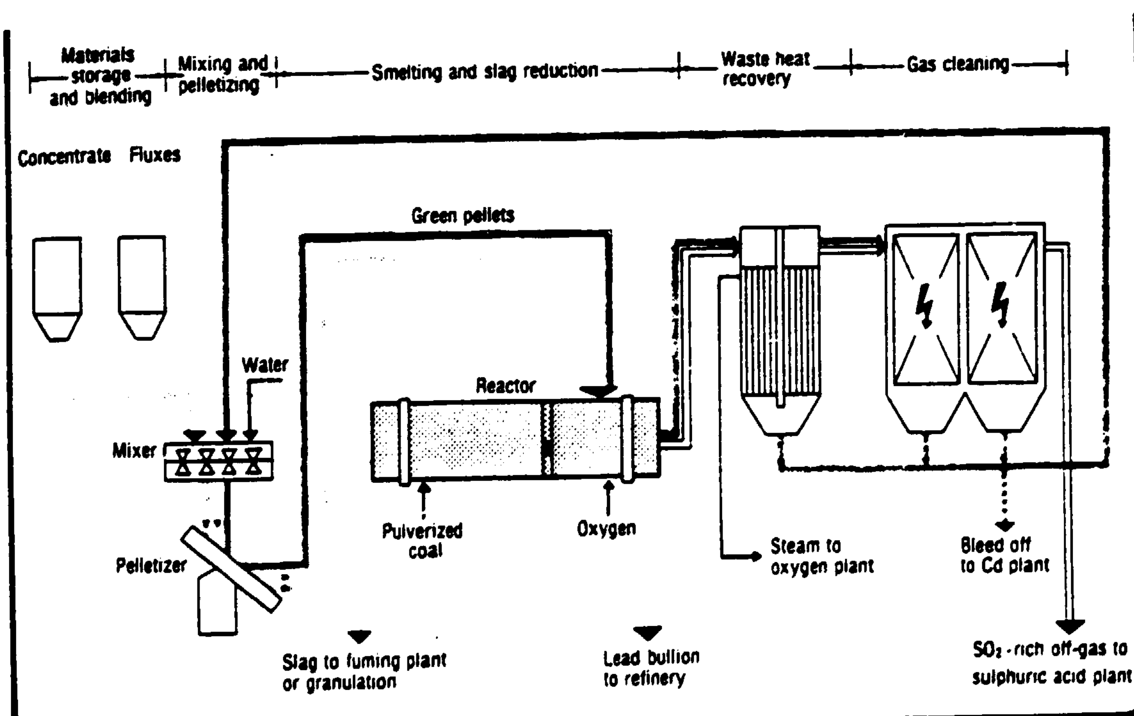
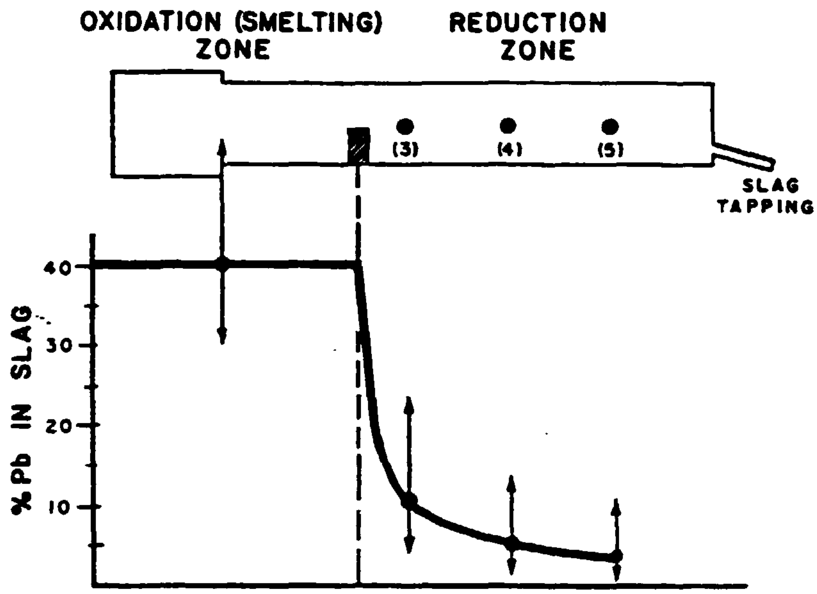


Diagram 22 - QSL Lead Smelting Process



LEAD CONCENTRATION PROFILE IN SLAG
rom LURGI graph C83-1033E presented by P. Fisher at 1983 AIME Annual Meeting)

Diagram 23 : 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 23 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.

6.5 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.

6.6 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.

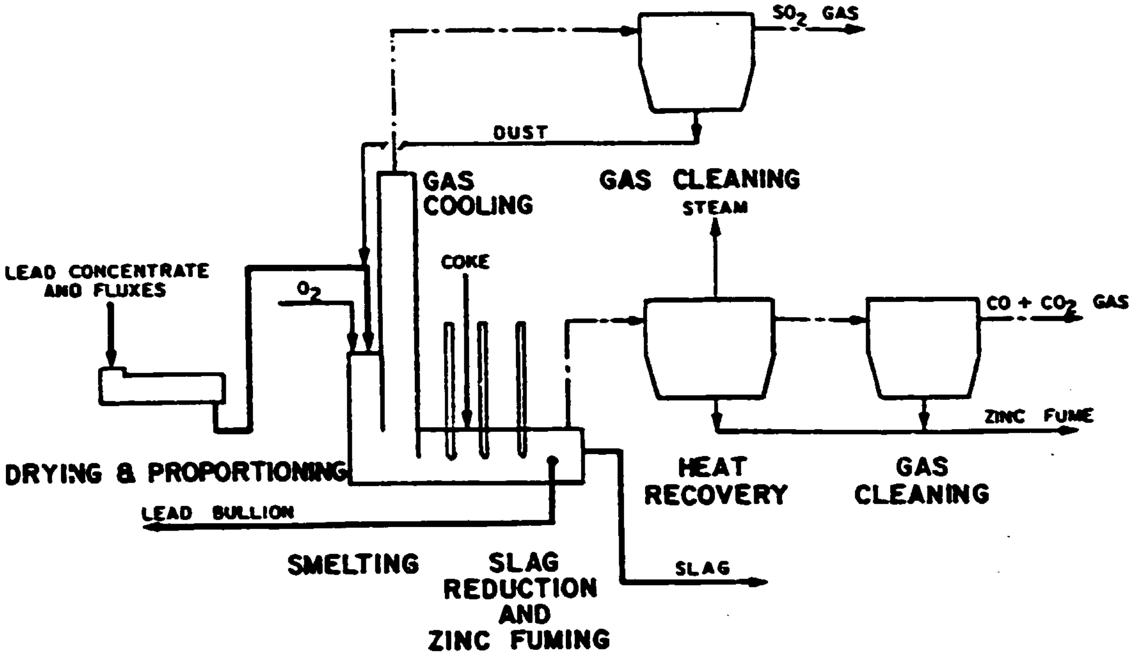


FIGURE 15. KIVCET CS LEAD SMELTING PROCESS

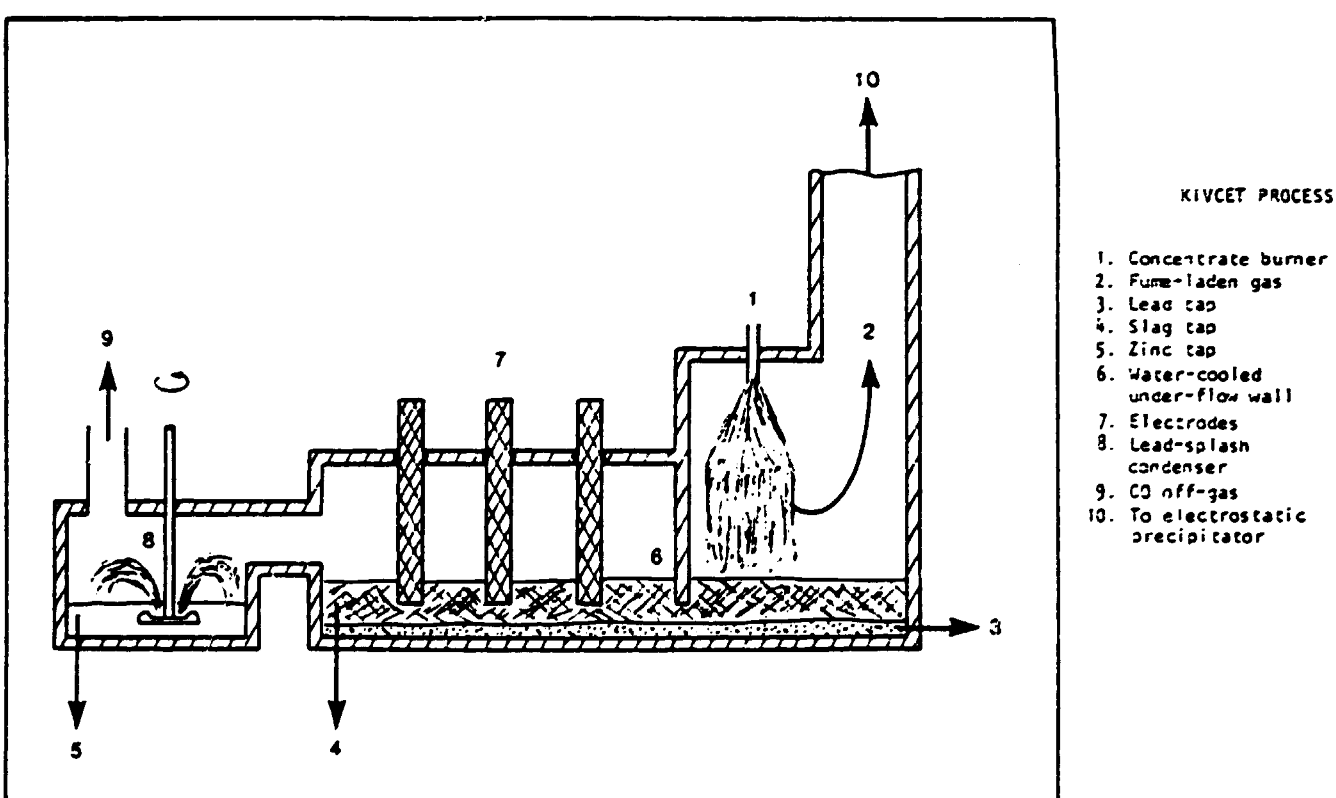


Figure 16 : KIVCET FLASH SMELTING FURNACE

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.

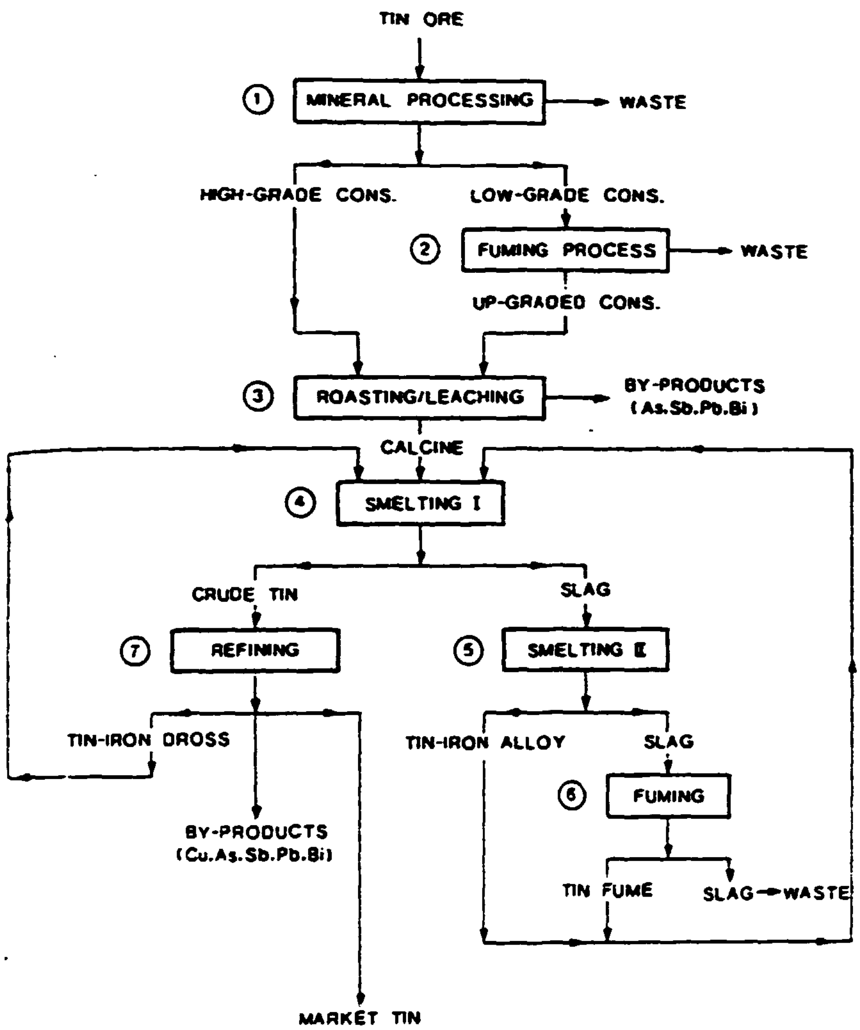
7.0 TIN TECHNOLOGIES

7.1 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 24. 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 24), 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. 24 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 chloridizing) 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 leaching 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.

7.2 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.

7.3 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.

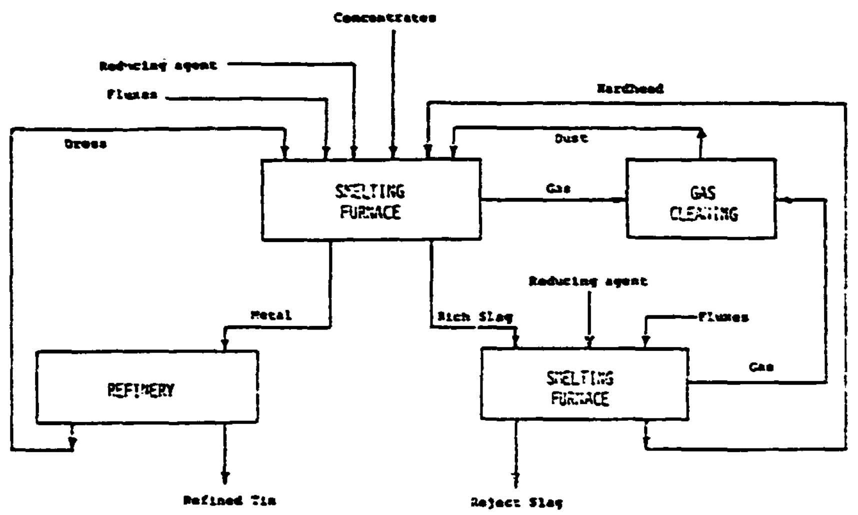


Figure 17 The conventional tin smelting circuit.

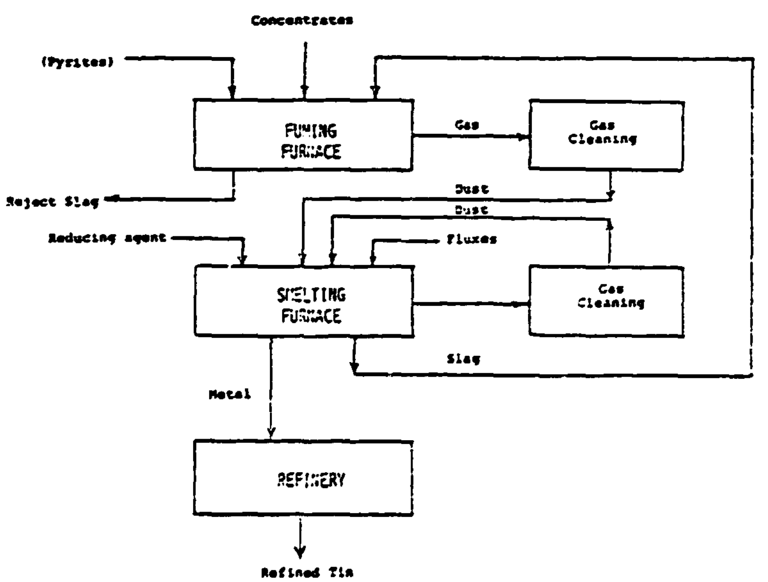


Figure 18 Smelting circuit for low grade concentrates.

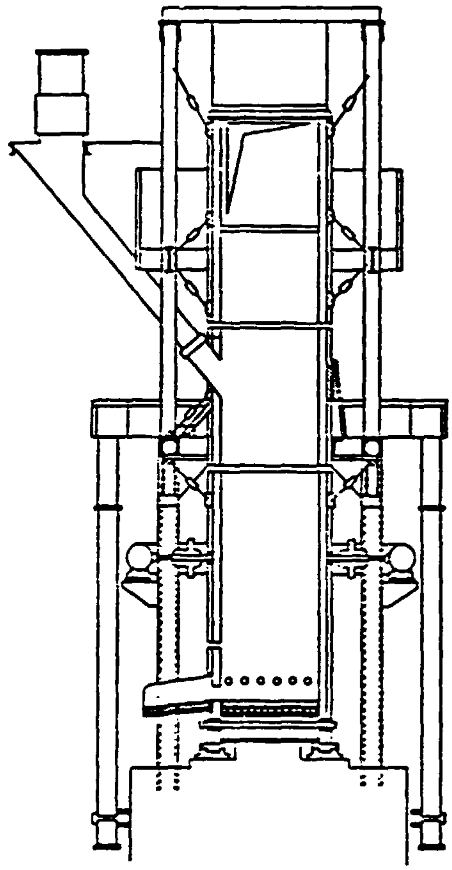


Figure 19 The stationary tin fuming furnace as designed by Kolodin.

8.0 POSSIBLE IMPLEMENTATION STRATEGIES

8.1 VERTICAL AND HORIZONTAL INTEGRATION

There is little doubt that some existing and emerging new technologies in the area of non-ferrous base metal production can be of great benefit for developing nations. We have already mentioned some important implementations made in Chile in its copper expansion programs. Similar uses of technology in other countries can greatly contribute to vertical integration of metallurgical operations and horizontal integration with other productive sectors. Vertical integration in the area of non-ferrous metals not only reduces transportation costs related with shipping metals instead of concentrates but also generates new employment in smelting and refining and leaves in the country the added value of the metals. It also contributes to greater access and consumption of metals in the country through development of semi-manufacturing and manufacturing industries, which can be both export and internal consumption oriented. Such is the case of converting copper cathodes into wire rod, which is then transformed into insulated and bare wire and cable. Zinc and copper are raw materials for brass mills which manufacture strip, rod, bars, mechanical wire and a number of manufactured products, such as plumbing and commercial tube. On the other hand, foundries process lead, zinc, tin and copper into a number of alloys and castings, while powder plants produce supplies of powdered metals.

All these products find practical uses in building, electrical, electronic, transportation and other industries, including industrial machinery and equipment, consumer and general products industries. Indeed, once cheap and ample supplies of metals are domestically available, they spur a number of activities related with their uses and contribute to horizontal integration of the metals sector with other production and manufacturing industries. This is apart from the creation of new activities related with operation of and supplies to metal industries themselves, such as energy supply, transportation, all sorts of materials and food supplies, communications, etc.

This vertical integration of mining and metallurgical base metals industries in developing countries is gaining an even greater importance when we consider the present structural change in industrially developed countries from production of goods to services and from tangible to financial assets, which is accompanied by escalation of the labor cost, loss of competitiveness in metals markets and general deterioration of mining and metallurgical activities. Since metals markets are not expected to improve substantially in years to come, because of both the lesser growth in metal demand and the existing oversupply situation, this is a unique opportunity for developing countries to firmly establish themselves in this sector and to displace competitively the industrial world base metals producers.

It should be clearly understood that this cannot be done by a temporary cut in costs and prices, but rather this must be a long-term policy. This integration should be helped by the now decreasing energy costs, but we must be aware that this is only a temporary phenomenon: oil and energy prices are going to bounce back immediately after supply and demand get into balance or supply becomes short because of the effects of low prices. This is unavoidable for two reasons: first, because our population grows at about a 1.7% annual rate, and with it the demand for energy; secondly, because our standards of living improve, and with it grows the energy demand per capita. So, it is only a question of time before the present oil and energy oversupply is absorbed.

In these circumstances, we should not lose time in enforcing our energy saving measures and introducing new, integrated technologies for most effective and lowest cost production efficiency. Only so can the developed world producers be displaced and substituted on a long term basis by developing world producers. As already mentioned, our strategies must be based not on expectations of higher metals prices but rather on substantially decreased production costs.

8.2 POTENTIAL AREAS OF ADVANCE

In this context, the most promising areas for technological development in developing countries are those related with:

a. Energy savings and materials handling in mining operations.

b. Energy savings and processing efficiency increments in comminution operations, including semi-autogenous grinding and cyclone classification.

c. Substantial improvements in flotation technology through greater volume flotation cells, introduction of new types of flotation cells, such as column cells, and improved reagent formulae.

d. Greater use of hydrometallurgical energy and capital saving technologies as reflected in bacterial leaching, solvent extraction and electrowinning technologies.

e. Introduction of processes which perform conversion of the metal content of minerals, both of sulphide and non-sulphide character, into the metallic state with the lowest consumption of energy, as for example in the SX/EW and Segregation processes.

f. Greater emphasis on byproduct recovery in all phases of base metals production, including flotation separation, hydrometallurgical, pyrometallurgical or electrometallurgical separation. A number of important metals, such as gold, silver, molybdenum, cobalt, bismuth, selenium and rhenium, can serve as valuable byproduct credits to reduce production costs of base metals. Also, non-metallic components such as sulphur and arsenic may be recovered at a profit.

g. Introduction of modern oxygen technology in base metals smelting in all its possible forms, with a clear aim towards optimization of results both economically and capacitywise.

h. Systematic study of potential advantages to expansion of existing facilities through improved and intensified technologies versus construction of new plants, which may result in very significant reduction of capital costs, typically two to three times smaller for expansion projects vs. new installations for the same additional production capacity.

i. Intensify studies related with optimization of scale of operations for a determined process or technology, which can come from economy of scale or through miniaturization of plants more in accordance with national necessities and possibilities.

8.3 ORGANIZATION OF RESEARCH AND DEVELOPMENT

New processes and technologies are created generally through the following consecutive steps:

1. Conceptual development of an idea which appears for the solution of an existing problem or creation of a new product or technology. This work may be carried out in an engineering office, research institution or university.

2. Experimental confirmation of the concept through laboratory testing and experimentation, which is carried out

in a university or in industrial laboratories.

3. Development of a model for equipment and flowsheet for the new concept or technology with the idea of testing it experimentally. This is normally done by a specialized engineering organization in cooperation with the author or authors of the idea or by inventors themselves.

4. Construction of a pilot plant to reproduce the idea or process on a small scale, sufficient to study the parameters of the process and to arrive at the necessary conclusions about the efficiency, viability and practical importance of the process, as well as to obtain all necessary basic data for engineering. Depending on the size and cost of such a pilot plant, it can be erected at a university, research institute or directly at the site of some industry.

5. Pilot plant testing eventually leads to development of a prototype or model of the definitive equipment or flowsheet, which is then implemented on an industrial or semi-industrial scale. This is generally done by an engineering firm or industrial organization.

6. Once such equipment or technological process is fully implemented, it serves for testing and marketing of the product on an industrial scale. It is generally the industrial organization which developed it which markets it, either througn selling patents and licences for its uses or through construction of specific plants for a fee.

In the area of non-ferrous base metals mining and milling, most of the existing equipment and industrial installations have been developed in industrial nations. Large equipment making companies, such as Allis-Chalmers, Dorr-Oliver, Western Machinery, Armco Grinding Systems in the United States;

Lurgi, Humbolt and Wedag in West Germany; Imperial Smelting in the UK; Atlas Copco in Sweden, and many others in a number of industrialized nations, have designed and constructed modern equipment and machinery, which they market through their organizations. On the other hand, large engineering firms, such as Bechtel, Parsons, Davy-McKee, Wright Engineers, Fluor, etc., are instrumental in the introduction of new technologies and equipment through specific projects and studies. There is a very numerous international community, particularly in the Northern Hemisphere, numbering probably dozens of thousands of all kinds of specialists, which can be used for all kind of services, technology transfer and execution of projects. Also, there are a large number of specialized metallurgical and other laboratories, such as Warren Spring Laboratories in the UK; BRGM in France; Hazen Research, Mountain States, USBM Research Laboratories in the USA; Canmet in Canada, and many others, which can do all kinds of testing and pilot plant confirmation of processes. Good contact and collaboration with all these institutions is very important for developing countries because at these institutions they can get all necessary training in modern techniques and associate with them for know-how and technological transfer.

On the other hand, developing countries also have significant research institutes of their own. These are, for example, Comisión de Fomento Minero, at Tacmachalco in Mexico City; Mining and Metallurgical Research Center in Batainica, near Belgrade in Yugoslavia; Mining and Metallurgical Research Center (CIMM) in Santiago, Chile; Mining and Metallurgical Research Center, at Oruro, Bolivia; Compañia de Pesquisa de Recursos Minerais (CPRM), at Rio de Janeiro in Brazil, and so many other research institutions all over the world, in India, China, Peru and other developing countries.

In cases of specialized equipment such as Aerofall or SAG mills, cyclones, flotation cells, etc., development of new ideas has already been done by certain specialized equipment making firms, and the conveniences of such equipment should be determined by the client himself, either by direct testing of such equipment at his plant (normally in cooperation with the equipment producer) or by contracting such service either from the equipment producer or some independent organization, such as a university or research institution.

In cases of new technologies related with testing of new reagents or hydrometallurgical or chemical processes in general where such testing does not require large industrial installations but can be conveniently conducted on small size equipment, this work is generally carried out at university, research or industrial laboratories, all depending who the client is and what his final objective is.

An innovator in flotation reagents or chemical processes may approach a university just to see how his idea works, which is the cheapest way to do it. If such an innovator wants to place his product on the market, he may wish to prove its merits to his potential clients through laboratory or pilot plant testing at some reputable research or industrial installation. Finally, if the product or process already has established itself as reputable technology, then final client testing may be carried out by the client himself at his own industrial installation.

In the area of heavy equipment and processes, such as flash smelting furnaces, fluo-solid reactors, new types of reverberatory and convertor installations, etc., industrial rather than laboratory testing should be conducted. True, there are some small size pilot plant testing units in fluo-solids, rotary furnaces and reactors, and even some small reverberatory furnaces can be built - but all this serves only for very preliminary studies. The real confirmation work for a process on specific ores or concentrates should be done at industrial type installations. And this requires large investment and prolonged experimental work, which nobody but the interested parties can afford.

Development, pilot plant and industrial plant testing may take anywhere from a couple of years to a decade to materialize. Flash Smelting of lead concentrates as developed by Outokumpu took over two decades to develop in spite of all the experience of Outokumpu in copper flash smelting technology. Also, Paul Queneau, the original developer of Inco Flash Smelting, had worked with Lurgi, a West German metallurgical engineering firm, for probably a decade before he developed and put on sale his QSL Process, now being marketed by Lurgi. Lurgi constructed a medium size industrial unit for processing 10 tph of galena concentrates at Metallgesellschaft AG's Metallhuttenwerke Berzelius in Duisberg, which now runs on a constant basis with a 30,000 tpy capacity of metal. Of course, Lurgi is a subsidiary of Metallgesellschaft and has to recover its investment in this process through selling technology to all those interested. Obviously, the only way to test this technology is to send concentrates to Lurgi or try to build, with Lurgi's licence, a similar furnace, which would be very expensive.

The Russian Kivcet Process was developed at a Siberian city named Ust Kamenogorsk, where a 25 tpd pilot plant was erected. To sell this process to Bolivia and Italy, bilateral agreements were made, and combined groups of specialists participated in pilot plant testing and industrial design for large-scale furnaces.

Outokumpu is also selling its technology in copper and lead worldwide. It has its furnaces installed all over the world, including in the Soviet Union. General arrangements call for testing of metal concentrates in Finland for a fee and then development of an engineering project for specific cases, for which Outokumpu receives a royalty. It also can or will not participate in construction and engineering of the furnace. For example, at Chuquicamata it is a Japanese firm which will construct the flash smelter.

In SX/EW technology, the pioneer work belongs to General Mills and Holmes and Narver. The testwork on this process is rather simple and can be carried out at laboratory and small pilot plants. Since the process has been fully implemented at the Lo Aguirre Plant of Pudahuel Mining Company near Santiago, a lot of useful industrial information has been developed and can be used for all those interested in process. CIMM, the Chilean mining and metallurgical research center, also has all necessary equipment to test this process.

On the basis of the above information, it can be concluded that practical implementation of new technologies in the non-ferrous base metals area in developing countries will require a rather complex and imaginative combination of work at existing research facilities at universities and national research centers, along with work at industrial type research organizations and engineering firms. The cooperation on the South/South level will be probably carried out between different national research centers and institutions, with some possibilities for engaging some industrial testing, while the cooperation on the North/South level will be pre-eminently carried out on an industrial testing level with the idea of transferring technology directly. To avoid turnkey-job situations, which have so many negative effects on developing countries, a maximum engagement of national talent of interested developing countries must be secured so that the technology transfer takes place with the fullest and most intimate knowledge of all necessary technical details and ideas.

This will also be instrumental for development of new technological ideas in the future.

When Kennecott bought its smelting process from Noranda, arrangements were made for the chief of Noranda's metallurgical group which developed the Noranda Process to become a member of Kennecott's staff in charge of the project. If such exchanges are required between developed nations, then even more is necessary for technology transfer between developing and developed nations.

This indicates that in developing nations, national research institutions, in practical terms, will be able to serve as first test grounds for existing or emerging new technologies, with the function of sorting out viable from non-viable new processes and technologies. However, for final decision making, engagement of specialized engineering firms, local and international, will be required to fulfill the job satisfactorily.

8.4 SPECIFIC PROJECTS IN NON-FERROUS BASE METALS

8.41 Copper

In Table 10, we give a summary of copper mine and smelter production in developing countries. From the indicated figures we can clearly see that developing countries account for between 45 and 46 percent of the world's copper output and about 55-56% of the Western World total. Moreover, we can see that developing countries' copper mine production has practically stagnated in these last four years of economic crisis. In spite of development of new production in some countries, others have suffered substantial deterioration. This means that the present crisis also affected production growth for this metal in the developing world as well.

TABLE 10

MINE COPPER PRODUCTION IN DEVELOPING COUNTRIES

(in thousand metric tons)

	1980	1981	1982	1983	1984	1985
Chile	1068	1081	1242	1257	1290	1356
Zambia	596	587	530	591	576	520
Zaire	460	505	503	502	501	513
Peru	367	328	356	322	364	387
Philippines	304	302	292	271	234	226
Mexico	175	231	239	206	189	173
P.N.G	149	165	170	183	164	175
Yugoslavia	117	111	119	130	137	131
Zimbabwe	27	25	25	24	23	23
India	28	25	24	44	47	48
Indonesia	59	63	75	79	86	72
Iran	1	2	43	65	60	60
Malaysia	27	29	30	29	29	29
Turkey	21	34	33	25	27	33
Brazil	2	14	24	32	26	36
Total	3401	3502	3705	3760	3753	3782
		Smelter		Production		
Chile	953	954	1047	1059	1098	1088
Zambia	601	572	681	563	532	514
Zaire	426	468	466	465	464	472
Peru	349	302	323	297	334	351
Phillippines	---	---	---	58	109	125
Mexico	86	61	62	70	70	65
Yugoslavia	94	102	106	119	126	127
Zimbabwe	26	30	31	31	24	30
India	29	26	33	35	40	34
Others	16	27	53	98	141	162
	2580	2542	2802	2795	2838	2968

The smelting of copper concentrates produced in developing countries covers about 75 percent of mine production. Actually, this figure fluctuated between 72 and 78 percent in the last six years, with a tendency to increase, which is in line with vertical integration of mines.

It can be observed that formerly colonial countries, where foreign capital was firmly established, all had perfectly integrated operations, which is the logical way to do it. This refers to Zambia, Zaire and Zimbabwe. The lowest integration rate was in free underdeveloped countries, where mines were developed with foreign capital in order to make them be "captive" production for their own smelters. The only countries which escaped this were Chile and Peru, but still have have a small deficit in smelting capacity for their own national small and medium size mines.

It is remarkable that while Chile didn't build any smelters between 1980 and now, its smelting capacity increased 15% since 1981 due to the implementation of the El Teniente Process, which is still only partially introduced into the smelting sector. When it is fully implemented, smelter output is expected to be between 35% and 40% higher than in 1981, i.e., about 1.3 million tpy. With the new Outokumpu Flash Smelter, now under construction, it will reach 1.5 million metric tpy. This means that within two years the Chilean copper industry will be fully integrated.

In Zambia, Zaire and Peru, existing capacity is sufficient, but costs could probably be cut by further improvements of technology and better operation of smelters. Mexico is in the process of building a new smelter at the La Caridad mine to take care of its increased copper output, of which it has to export about two thirds in the form of concentrates. While the concentrate business today offers some advantages because of the shortages of this material on international markets, from 1988 on such shortage will be turned into an oversupply because of the development of new projects which do not provide for their own smelters. In this case, concentrates will suffer some surcharges instead of premiums, which will be to the disadvantage of primary producers.

Brazil and the Philippines have each put on stream a new smelter since 1983. In Brazil, it was put near Salvador, in Bahia, and in the Philippines at Pasar. Both have had initial difficulties in their operations, which only shows how technological transfer is still difficult and imperfect. At any rate, these operations will eventually work well but need numerous adjustments. Probably the most important is to increase their efficiency, which is rather low and is reflected in abusive surcharges on smelting of metal.

Papua New Guinea, Malasia and Indonesia do not have their own smelters because most of their production goes to Japanese smelters. They are typical captive mines and probably will never have their own smelters. Even construction of the relatively small Pasar smelter caused discontent among concentrate buyers because it cut into their supplies.

India, Iran and Turkey are in a considerable better position because of the integration of their production lines. But Iran had to overcome many difficulties to initiate its copper production, which was delayed for several years because of the Islamic Revolution. Operations at Sar Cheshmeh still work at about 45 to 50 percent capacity.

For a 3.8 million metric ton copper metal production, developing countries have to process about 300 million tons of ore and to extract probably as much as 700 to 800 million tons of rock.

In mining costs this represents probably between $ 600 and $ 700 million, and in milling costs a similar figure. Provided that efficient use of explosives is made and that material is removed by already developed new technologies, as reported in Chapter 4.1, introduction of modernized mining and transportation methods could save as much as $ 100 to $ 120 million per year.

Similarly, by introduction of SAG grinding, milling and concentrating of these ores can save 50 cents per metric ton in the comminution process alone. This is roughly $ 150 million per year.

Then comes the possibilities of flotation technology improvements and byproduct recovery, which can cut concentrating costs from 10-15 c per pound of copper, probably by 5 to 10 percent, if proper operation of circuits and reagent formulae are applied and if maximum byproduct recovery is assured. This is also a benefit which may range from $ 30 to $ 100 million.

Finally, by application of oxygen technology in smelting, in which we stand to gain anywhere from 1.5 to 3 cents per pound of copper smelted, there is a potential gain of between $ 100 and $ 200 million.

If we add to these figures the benefits which we can achieve by using the SX/EW process in processing oxide ores and old sulphide dumps, which means an average cost 15 to 30 cents per pound lower than by traditional technology, then with a potential copper production coming from this area alone, and which is estimated at at least 300,000 tpy, the expected benefit should be between $ 100 and $ 200 million.

All told, the indicated improvements in technology through more efficient treatment or cost-cutting could save some \$ 430 to \$ 770 million per year for the copper sector of the developing world, which amounts to cost reductions of approximately 8 to 14 percent, or 5.2 to 9.1 cents per pound. This is just enough to keep developing world countries permanently in a very competitive position with respect to developed world producers throughout this period of mediocre copper prices.

Where and how should such research be carried out?

As we already suggested, porphyry coppers and sedimentary coppers merit separate attention because of their different metallurgical problems. This suggests that one research center of excellence could be located in Latin America and the other in Central Africa. South/South cooperation between these centers could continue in explosives, open pit mining, autogenous or semi-autogenous grinding and copper hydrometallurgy. But obviously foreign expertise from the Northern Hemisphere will be necessary for oxygen smelting technology. Of great potential interest for all copper producers is the El Teniente oxygen smelting process, which also forms part of South/South cooperation.

The general idea would be to select a center of excellence for copper research at an already established institution with a good record of past performance, and which is well equipped and with a competent staff. This institution, with government and international help, could accommodate research engineers and scientists from other countries and accept foreign experts from more developed technological areas. It also could have direct contact with engineering firms and even have resident representatives who would help in orienting research towards practical engineering goals.

8.42 Lead and Zinc

In lead and zinc things are quite different. As shown in Table 11, developing world production in this area is not so significant as in copper. Latin America, where the bulk of lead and zinc production of developing nations is concentrated, represents only 13.5% of world lead and 17% of world zinc output. In lead and zinc, the leading producers are the United States, Canada, the Soviet Union and Western Europe. Of real significance in the developing world in lead and zinc production are only Peru and Mexico, and to a considerably lesser extent Yugoslavia, Morocco, Argentina, Brazil and Bolivia. Peru converts only 40% of its lead production into refined product and less than a quarter of its zinc output. Mexico converts about 90% of its lead and about 60% of zinc into refined product. It is thus in a better position than Peru as far as integration of production lines is concerned.

Most of the lead and zinc mines in the developing countries are captive mines of larger corporations in the industrial world, have little capital and practically no chance for their own integration. The lead and zinc business in these last five years has been very bad, with prices very low and capitalization of companies at a minimum. Besides, many companies became involved in indebtedness, which practically ruins them.

Transfer of technology from the industrial world has been very limited and, except for the introduction of the Kivcet Process in Bolivia and Italy, little has been reported about other advances introduced into the developing world lead business. Most mines just work to sell their concentrates on international markets to developed nations.

TABLE 11

LATIN AMERICAN PRODUCTION OF LEAD AND ZINC

(in thousand metric tons)

LEAD - MINE PRODUCTION

	1980	1981	1982	1983	1984	1985
Argentina	32.6	32.7	32.8	31.7	28.5	29.0
Bolivia	15.9	16.8	12.4	11.8	7.4	7.8
Brazil	21.8	21.6	19.4	19.0	19.2	19.2
Chile	0.5	0.2	1.5	1.7	4.5	2.7
Honduras	13.3	12.6	15.1	19.3	20.5	20.4
Mexico	147.2	148.9	170.2	184.3	183.3	181.3
Peru	189.1	186.7	175.8	205.1	198.4	216.2
Total	420.4	429.5	427.2	472.9	461.8	477.0
World Output	3,583	3,447	3.550	3,473	3,412	3,528
LA as % of World	11.7	12.4	12.0	13.6	13.5	13.5

ZINC - MINE PRODUCTION

Argentina	33.7	35.1	36.6	36.6	35.1	36.4
Bolivia	50.3	47.0	45.7	47.1	37.8	40.2
Brazil	70.0	71.0	86.7	88.0	79.3	72.0
Chile	1.1	1.5	5.7	5.8	19.2	23.9
Guatemala	0.1	4.5	1.0	--	--	--
Honduras	16.0	16.2	24.6	38.0	41.5	46.5
Mexico	235.3	206.6	242.3	266.3	290.2	302.6
Peru	487.6	496.7	507.1	553.1	568.3	623.9
Total	894.1	378.6	949.9	1034.9	1229.0	1145.4
World Output	6,165	6,114	6,445	6,510	6,736	6,788
LA as % of World	14.5	14.4	14.7	15.9	18.2	16.9

If conditions allowed, the QSL Process could be tested on Peruvian and Mexican concentrates, but these nations are so indebted that few chances exist for these countries to afford to buy such a new process. Besides, it still remains unproven by an independent large producer from the developed world, and developing countries are not the best place for first attempts to test new processes and introduce new technology.

So, in the area of lead and zinc, research work should be probably confined to improvement of mining and milling technology, modification of reagent formulae for the flotation process, intensification of byproduct recovery and, probably more than anything else, in followup of complex processes such as sulphatation roasting and leaching of bulk concentrates, which should bring about higher overall metal recoveries. In this context, cooperation with Canmet, the Canadian Center for Mineral and Energy Technology in Ottawa, and with the Brunswick pilot plant test unit could be very useful. Incidentally, in Brunswick pilot plant test work are intimately involved a Chilean PhD and a chemical engineer from the University of Concepción, Chile, and an important mineral processing research program is also available.

8.43 Tin

Although tin production in developing countries accounts for about 70% of the total world output, as shown in attached Table 12, and thus is one of the very important activities for a number of countries, the present state of affairs is rather desperate because of the collapse of the International Tin Council's buffer stocks policy and consequent collapse of tin prices to less than one half of their previous levels. With this, many tin mines are simply out of business because they cannot operate with any margin of profit at the present price level. On the other hand, tin stocks accumulated by ITC, and now in the hands of banks and other creditors, are

TABLE 12

TIN PRODUCTION IN DEVELOPING COUNTRIES

(in metric tons of metal content)

	1980	1981	1982	1983	1984	1985
Malasia	61404	59938	52342	41367	41307	35200
Indonesia	32527	35391	33806	26553	23225	22133
Thailand	33685	31474	26109	19943	21960	18000
Burma	1290	1438	1681	1642	2028	2000
Bolivia	27291	29830	26773	25278	21100	18667
Brazil	6377	8253	9293	13418	17700	24400
Zaire	3159	3221	3144	2930	4120	2400
China	14600	15000	15000	15000	15000	17500
Total	180333	184545	168148	146131	146000	140300
World Total	247300	253113	237176	211620	209000	198000
DC/W %	92.92	72.91	70.89	69.05	70.07	70.85

too large to give hopes that this situation will change in a short time. Years will pass before the tin market gets back to order.

In these circumstances there is little that can be done to alleviate this situation. The standing problem for some countries is the vertical integration of their industries in order to reach markets under more favorable conditions. In this respect, however, all major tin producers from the developing world, such as Malaysia, Indonesia, Thailand, Bolivia and Brazil, have already reached the necessary integration.

The other problem, growing in its importance, is the depletion of high grade alluvial tin deposits and the necessity to mine each time lower and lower grades of ores with a great contamination of fines. It is here that traditional gravity concentration methods will probably give way to flotation technology, and this subject must be studied with anticipation.

If a research effort is developed in this area, the logical location of a research center would be probably one of the South East Asian countries with enough infrastructure to give enough scientific and technological support to the project.

8.5 SUGGESTED POINTS FOR DISCUSSION AT EXPERTS MEETING

We suggest that action in the non-ferrous base metal area be first circumscribed to copper, as an initial stage. Then experience from this project can be expanded to lead, zinc and tin. This idea is justified in our opinion by the fact that, fist, the copper production impact in developing world is far more important than that of any other non-ferrous base metal; second, in this area there is much more potential to get practical results through existing and emerging new technologies.

Moreover,in this area there is important experience among the developing countries which can be effectively shared through South/South exchange. The potential benefit of some $ 500 million in reduction of production costs is really impressive and will be attractive to governments, international organizations and participants. It will also involve the greatest number of developing nations in these projects.

Secondly, we think, some kind of coordinating committee or council should be organized to pursue this project through convincing governments, international institutions and professional organizations to participate in it. Potential benefits to producers, engineering firms, suppliers of equipment and research media should be demonstrated by presenting a work program to cover all areas of copper production.

Thirdly, discussion may be opened on potential centers of excellence which could execute this program, with the eventual possibility of visiting such centers before making any decision.

Also, it must be provisionally determined as to what kind of professionals will be involved in such a project, where and how all necessary equipment and working groups will be assembled, what kind of budget may be necessary for all of this, and who is going to subsidize it.

Basic rules of operating and financing can be also discussed. In this respect, it should be clarified as to who is going to benefit from the new technologies and in what way. Should industrial companies interested in all this advance line up with an initial quota for expenses and then automatically receive benefits when such appear through research? Or should new technologies be discovered first or adapted to specific cases and then sold on the basis of patents? What kind of stimulation will creative workers enjoy in research in order to avoid bureaucratic and unmotivated approaches to the whole problem?

And, for that matter, how will working groups be selected and put together?

Careful consideration should also be given to relations with engineering firms which are normally intermediaries between new ideas and their practical application. Will there be consultants for different projects on the basis of a retainer, or will only staff members be involved in the different projects?

Then the legal problems related with all this international setup must be carefully considered in order to avoid future troubles, particularly regarding copyrights, patents, commercial transfer, etc.

The project must be clearly formulated as a program, with clear specifications of basic rules for participation, formal obligations and possible rewards obtained through new discoveries and their applications.