COMMISSION DES COMMUNAUTÉS EUROPÉENNES

200, rue de la Loi - BRUXELLES

COPPER DOSSIER

VOLUME 1

BASIC ELEMENTS

BUREAU DE RECHERCHES
GÉOLOGIQUES ET MINIÈRES
B.P. 6009 - 45018 Orléans Cédex
(France)

A.M.M.I. Spa
19 Via Po
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MAIN ABBREVIATIONS (§ 1,2,3,4,6,9)

t	:	metric tonne (all tonnages are metric, unless otherwise stated)
kt	:	thousand metric tonnes
Mt	:	million metric tonnes
tpd	:	metric tonnes per day
tpy	:	metric tonnes per year

\$, ¢, (M\$)	:	US dollars or cents (million US \$)
F (kF, MF)	:	French Francs, (thousand FF, million FF)

1. COPPER GEOLOGICAL CHARACTERISTICS

1. COPPER GEOLOGICAL CHARACTERISTICS

1.1 Types of deposits and world reserves

1.1.1 Types of deposits

There are several classifications of the various types of copper deposits, and many of them date from some time ago. Most of them distinguish a large number of types and sub-types, according to the host-rock, the geotectonic environment, shape, mineral association, etc. These distinctions are useful for scientific reasons, and frequently also for formulating precise prospection guidelines. However, in this introductory chapter of the EEC study, we will present a simplified classification and a brief description of the principal types in order of their economic importance.

1.1.1.1 Copper porphyries and associated mineralizations

Copper porphyries have, quite recently, become of considerable and increasing economic importance. The Canadian figures are very significant in this respect since, in 1970, copper from this type of mine represented 5.7 % of accumulated production and 54.2 % of reserves [1]. Table 1 shows that, on the world scale, these deposits represent 50 % of production and more than 60 % of reserves of copper content, and this figure continues to increase. There are, therefore, many studies and publications on porphyry copper, and,in spite of some divergencies, the type is quite well defined.

The geological context : Porphyry deposits appear in orogenic zones. They are located in porphyritic intrusions of monzonitic to dioritic composition. The latter occur at average to shallow depths, and probably late in the orogenic evolution. The original deposit is, therefore, generally deformed only by faults (late tectonic phases).

The host-rock of the "disseminated" mineralization consists of the porphyry itself and/or the country rock, frequently including intermediate volcanic rock (andesite). The associated skarns and fissure veins are outside of the porphyry, and are clearly conditioned by the nature of the host rock (carbonated, etc.) and its deformations.

The porphyry mineralization itself consists of a stockwork of veinlets,sometimes with brecciated zones and/or disseminations in the rock itself. The primary ore contains chalcopyrite, bornite and pyrite(1), often in large quantities. Other recoverable metals are Mo, Au, Ag, as well as Re, Se and Bi. Lead and zinc are found in the periphery of the deposits, in the skarns and fissure veins forming mineralized bodies, which are much smaller but often have high grades.

Around the mineralized body, hydrothermal alteration creates relatively regular and concentric zones, very useful as prospection guides. In the complete pattern there are, from the center outwards, potassic, phyllitic, argillitic and propylitic zones.

The mineralized body has an irregular, often roughly cylindrical shape. It is of large size, with a surface of 1 to 2 km^2, but the body is sometimes broken into several parts, either as a result of previous and contemporary tectonic deformations, or by late faulting. The size of the mineralized body (or bodies) is very large. An average for 148 porphyry copper deposits in different provinces shows that the metal content is 2.3 Mt, equivalent to 500 Mt of ore with 0.46 % Cu, or 200 Mt with 1.15 % Cu. Very large deposits have several tens of million tonnes of copper content.

The ore grade is generally low, between 0.4 and 1.2 %, with an average of 0.80 %, and this figure declines as new discoveries are made. The grade never reaches 2 % since the richest known porphyry copper is the El Teniente (Chile) deposit, which contains 1.96 % Cu. Certain parts of the

(1) See table on page 3-2 for composition, grade and density of the main copper bearing minerals.

deposit at the summit can, however, exceed this grade because of a relatively frequent secondary enrichment process forming chalcocite.

In this groupe, S.E. Kestler, et al, [2] have made a distinction which seems to be justified between two sub-types :

- a gold-bearing sub-type, associated with island arcs linked to slightly more basic intrusions (quartziferous diorite) in a strongly volcanic environment;

- a molybdenum rich sub-type in more sedimentary and cratonic environment, linked to slightly more acid intrusions (quartziferous monzonite).

The principal porphyry copper belts are :

- The western American cordillera from Alaska to Chile, especially in British Columbia , the western part of the U.S. and Mexico, Central America and the Caribbean, Colombia, Ecuador and Peru;

- At least part of the Alpine Chain from Yugoslavia to Pakistan, especially in Rumania, Bulgaria, the Caucasus Mountains and Iran;

- The island arcs of South-East Asia : Indonesia, the Philippines, New Guinea, the Solomon and Fiji islands.

This list includes all areas with large copper production and reserves, except for Poland and the Central African copperbelt. This is not by chance that all the orogenic zones involved are of Post-Jurassic age. Certainly there are copper porphyries in Paleozoic Chains (especially in Kazakhstan, with the large Kounrad deposit, and the Caledonides deposits south of the St. Lawrence), and there may even be Precambrian deposits; but the majority are recent to very recent on the geological time scale. It can be supposed either that the metallogenic efficiency of the "porphyry system" has increased from Precambrian to

Tertiary time, or that the deposits of ancient orogenic zones have been eroded because of their rather superficial position. Anyway, the relative youth of porphyry copper, which has been stressed many times and most recently by J.B.E. Jacobsen [3], is an important factor for previsional research.

1.1.1.2 Stratiform deposits in sedimentary cover

Up to recent years, cupriferous schists, marls and sandstones produced about 25 % of the world's copper. Table No. 1 seems to show that their economic importance has slightly decreased, since they contain only 21-22 % of reserves, but this trend is less evident if recoverable copper, rather than copper content, is considered since deposits of this type are favored by their high grades.

The geotectonic background is very different from that of copper porphyries. These deposits are located in intracratonic zones, subject to extension phenomena. The geological environment is mainly sedimentary, and the associated basic volcanism is often not very large and not closely linked to mineralization.

Paleogeographical investigations are of primary importance, for the deposits are located in lagunal-deltaic series at subsident basin borders. More exactly, ancient shores, paleoreliefs and paleochannels play an important role in concentration.

The host-rock of the mineralization varies and it is possible to distinguish two sub-types :

- Cupriferous schists and marls (kupferschiefer), where very extended and conformable mineralization overlies fine detrital sediments (shale);
- Cupriferous sandstones (red beds), where the copper is located in coarser sediments, siltstones, sandstones and conglomerates, where the deposit is often less regular and peneconformable.

The two sub-types are generally associated, at the district scale, and several mineralized horizons are contained in the same stratigraphic unit. The host-rock generally has a reducing character, but the incorrectly applied term,"red beds", is due to the red layers which are often found around the deposits, as well as occurrencies of evaporites.

In general, the sediments after mineralisation are only affected by slight folding, but there are many, usually early, faults. On the other hand, in the Copperbelt of Zambia and Zaïre there is much major folding, as well as overthrusting, due to later orogenesis.

Disseminated mineralization cements the host-rock, and is characterized by relatively low sulphur and iron, and therefore pyrite contents. Chalcocite and bornite have an important place in paragenesis. Native copper and domeykite (in an arsenicated environment) are frequent, and chalcopyrite is relatively abundant. The principal by-products are cobalt (especially in Zambia-Zaïre), uranium, lead (Lubin), and the chemical spectrum is very broad, including Ag, Ni, Be, Se.

There are often two types of zonation, which still have not been well explained : in the deposit itself, horizontal and/or vertical zonation appears with two poles, i.e. iron (pyrite) and copper (chalcocite), which seems to be linked to the concentrating trap. At the belt scale zonations with Fe, Ni, U, Cu, Co and Pb-Zn appear, which are probably linked to the source of these elements.

The mineralized body is essentially bidimensional, although certain mineralized horizons exceed 10 m in thickness, especially in detrital environment. The size of the deposits varies a great deal, but the grade is almost always high : 1 to 5 % Cu and more. H. Pélissonnier [4] calculated a general average of 2.20 % Cu, quite close to our African average of 2.36 % (Table No. 2), which is dominated by the Copperbelt.

Two large metallogenic provinces are dominant for this type : Firstly, the Copperbelt of Zaire and Zambia and, secondly, the Central European Zechstein area (D.D.R. and Poland), which together contain almost 75 % of the known reserves of cupriferous schists and sandstones. The first province is of Upper Precambrian age, which also includes the deposits of Udokan (USSR), White Pine (U.S.) and deposits in the North-West of the U.S. and British Columbia, etc. The second is of Permo-Triassic age, like several other deposits in the Urals, in the Epi-Hercynian cover of Europe and in the South-South-West of the U.S. There are other deposits in the Devonian-Carboniferous (Djezkazgan, USSR), the Cretaceous of Africa, and the Oligo-Miocene of Bolivia (Corocoro). However, the Upper Precambrian and Permo-Triassic periods seem to be the most favorable, as is confirmed by recent discoveries at these two levels, in the East of Greenland.

1.1.1.3 Massive sulphide deposits in volcano-sedimentary series (sensu lato)

This type of deposit, which had a restricted meaning and well established characteristics a few years ago, is now being given a broader definition. It is therefore difficult to determine its economic importance exactly. It can be roughly stated that it represents 15 % of accumulated production, and nearly 10 % of the copper reserves in the world. However, its real economic value is much greater because the associated metals are of importance and sometimes even have a predominant role.

The geological domain for these deposits largely coincides with, and overlaps, that of porphyry copper. Volcano-sedimentary formations are involved, which are laid down in the first phases of geosynclinal evolution, and, therefore, in zones of major folding. The deposits are thus much deformed, more or less metamorphosed and intruded by masses, sills, dykes of acid and basic rocks.

The enclosing formations can be :

- Acid and/or basic volcanic series with abundant lavas (pillow lavas), mainly deposited in a marine medium;
- Geosynclinal sediments with less developed volcanics (pyroclasts);
- Marine series without contemporaneous volcanic manifestations, at least in the immediate environment of the deposit.

These variations are used for distinguishing sub-types, which can be roughly divided as follows :

- Massive sulphide deposits, in the strict sense, which are generally polymetallic and are enclosed in essentially volcanic series. They consist of more or less conformable deposits of massive pyrite ore, overlying a mineralized pipe or stockwork;
- Deposits associated with volcanism, often predominantly Pb-Zn, in thin beds or bodies which are sometimes unconformable and located in sedimentary series, with discrete volcanism. In the "hydrothermal sedimentary deposits", the volcanic activity can be reduced to hydrothermalism without volcanics.

The distinction between proximal and distal mineralizations is linked to the above sub-types, but has still not yet been well defined. Depending upon the authors, it is based upon the distance from the volcanic vents, or from the mineralizing source.

In almost all cases, there are associated chemical sediments, which are usually siliceous sediments (jasper), and hydrothermal alteration with quartz, carbonate, black chlorite and sericite.

The mineralization also varies : massive ore with pyrite-pyrrhotite bodies containing Cu, Zn, Pb, Au, Ag and Bi in varying proportions; networks of veinlets which are often quartzose and richer in copper in the underlying stockwork; conformable beds, stockwork, massive or dissemintaed deposits, and even associated veins in "hydrothermal sedimentary deposits" with Pb-Zn predominant. The chemical spectrum is often complex, with base and precious metals and Sb, Cd, Te, Sc, Co, Ni, Sn.

Within the same deposit, the spatial distribution of the various mineralizations varies, but there is a general and clear zonation with copper in depth and central position, and the Pb-Zn located in the upper part and/or on the periphery.

The mineralized body takes various shapes, which have been briefly described above, and a strong tectonic deformation often makes it difficult to mine, except in the case of true massive deposits. This handicap is often compensated by high metal contents. Copper grades vary from 0.5 to 2 %, with an average of 1.35 % Cu, and, in general, the copper content is greater when volcanism is predominantly basic. Both these polymetallic deposits often have higher Zn and Pb grades, and the ore frequently contains more than 10 % metal with valuable amounts of Ag and Au. As far as copper is concerned, an average deposit of this type is of medium to relatively small size. Deposits with more than 1 Mt of copper content are quite rare (Rio Tinto, Mount Isa), and most are between 50 and 500 kt. However, the Pb-Zn content is often much larger, and in certain, rare cases, exceeds 10 Mt (Broken Hill, in Australia, Sullivan, Mount Isa, Bathurst), while it often is more than 2 Mt.

The number of these volcano-sedimentary deposits is very large, and they are found in many provinces and for all periods. The best known groups are the sulphide bodies of the Archean and Proterozoic Canadian shield, the deposits of Caledonides, the German Devonian deposit of Rammelsberg, the Carboniferous deposits of the South Iberian district, the Japanese Miocene Kuroko type deposits, and even some very recent mineralizations of the Philippines arc.

1.1.1.4 Ni-Cu sulphide deposits in basic-ultrabasic complexes

From the economic point of view, copper is a by-product of these deposits where the Ni/Cu ratio varies from 1 to 15, and even more. Even though the copper content is definitely below that of the three principal preceding types, it is not negligible and represents 3 to 4 % of world production, and 2.5 to 3 % of reserves (Table No. 1).

These deposits constitute a well defined group, characterized by the close link with basic-ultrabasic complexes, paragenesis with pyrrhotite, pentlandite and chalcopyrite, and the absence of gangue other than the host rock. The geotectonic background varies, depending upon whether the zone is cratonic or geosynclinal, and in the latter case, whether ante-, syn- or late orogenic. The classification of sub-types is based upon the support complexes, especially on their origin, mode of deposit, size and the relative importance of mafic and ultramafic rock, which are often respectively uralitised and serpentinised.

The size of the deposits and the Ni/Cu ratio are linked to these characteristics. Copper grades vary from 0.2 to 1 % Cu, with an average of 0.7 %, but there are large tonnages of ultrabasic rock with low nickel sulphide contents, and practically without copper (Mount Keith, Dumont). In addition to the predominant nickel, the associated products are platinum metals and, to a lesser extent, Co, Au and Ag. Fourteen elements, especially Rb, Te and Se, in addition to those mentioned above, are extracted at Sudbury, by means of extensive metallurgical processing.

As is shown by Table No. 3 on reserves, deposits of this type are concentrated in several countries which, in order of importance, are as follows : Canada, which has all types of deposit in Ontario (Sudbury), Manitoba (Lynn Lake, Thompson), British Columbia (Giant Mascot), Quebec (Ungava district); the USSR, with the deposits of the Kola peninsula (Petchenga, Montchegorsk, Allarechensk) and the Norilsk district; the U.S., with Stillwater (Montana) and the Duluth gabbro (Minnesota); and western Australia (Kambalda). Almost all these deposits are of Archean and Lower and Upper Proterozoic ages.

1.1.1.5 Types of deposits of secondary importance

This group of deposits contains about 5 % of the known stock of metal, with 5 to 6 % of past production, and 3 to 4 % of the copper reserves. It contains two kinds of deposits :

- those with little economic importance, but of a well defined type other than the four preceding ones;
- various deposits which are either not well known, or which have special characteristics, so that up to now they have not been linked to known types.

The following can be distinguished amongst the former :

- Native copper deposits in basic lava : Mineralizations of this type are widely spread, but they are rarely of economic grades. Only the basalts of the U.S. Lake Superior district (Michigan) have been actively worked. Known reserves are therefore low, less than 1 %; but these ores constitute potential reserves which are probably not negligible. H. Pélissonnier[5] links them to cupriferous schists and sandstones, with which they have affinities : stratabound mineralizations, poor in sulphur and located within a similar geotectonic context.

The classical and only important example is that of the Lake Superior district, which has produced about 450 Mt of ore at a grade of 1.27 % Cu, and with a content of more than 5.6 Mt of metal. Similar subeconomic mineralizations are, however, known in Canada, the U.S., Latin America (Brazil, Colombia), Europe (Norway, Germany, Poland, Yugoslavia), the USSR, Australia, etc.

- Cupriferous tin deposits contain 0.7 % of the known world copper content, but more than 60 % has already been exploited. The future importance of the copper of this type is therefore very marginal, but it has had a great deal of historic importance because of the Cornwall belt and the Japanese deposits.

These deposits have various characteristics, but their common denominator is the presence of igneous acid rock : leucocratic granite plutons and subvolcanic to volcanic structures. They overlie the contact between intrusions and the almost always paleozoic country rock series. In the classical districts, they are quartzose veins with definite plutonic zonation : cassiterite (and wolfram) with deep and intraplutonic greisenization; peripheral tin-copper zone, which is especially rich in copper on the outside and on top.

The Cu-Sn-W (Mo, Bi) association may have a certain importance again, because of the recent discovery of the so-called porphyry tungsten deposits as, for example, at Mount Pleasant (New Brunswick, Canada) and in Mexico.

- Copper-siderite veins, with negligible world importance, but which are well characterized and abundant in Hercynian and Alpine zones of Europe and North Africa. Of the many deposits of this type, only one is worth mentioning, i.e. that at Mitterberg (Austria) with its 170,000 tonnes of metal produced from ore with a 2 % Cu grade.

Deposits also exist which are sometimes large, but which cannot be classified amongst the types defined above and whose interpretation is still under discussion. Amongst those deposits which are classified as "miscellaneous", are those at Kipushi (Zaïre) and Tsumeb (Namibia), which are very close and for which a new type may be created, or Phalaborwa (South Africa), which is linked to an alkali intrusion. Most of the other deposits of undefined type can be fitted into the simplified classification above, provided it is remembered that there are transitional types, for example between cupriferous porphyries and sulphide bodies, or between deposits linked to volcanism and "kupferschiefer".

These ambiguities in the classification and variations in the degree of knowledge, probably explain the differences in Table No. 1 showing data on the economic importance of the various types. The differences are moreover usually small, except in the case of the cupriferous schist and sandstone reserves. This is, to some extent, explained by the fact that, according to H. Pélissonnier, this type includes native Cu deposits in basic lava, and especially by the use of a slightly less restrictive notion of reserves (in the case of the African Copperbelt, for example).

TABLE N° 1

ECONOMIC IMPORTANCE OF VARIOUS TYPES OF DEPOSIT

(In million tonnes of Metal)

TYPES OF DEPOSIT	ACCUMULATED PRODUCTION		RESERVES		AVERAGE ORE GRADES
	Mt of Metal	Percentage	Mt of Metal	Percentage	
PORPHYRY COPPER	95,- - -	50,5 % 53,2 % -	305,- - 340,2	59,5 % 63,6 % 61,8 %	0,80 % 0,5-2 % -
CUPRIFEROUS SCHIST AND SANDSTONE	50,- - -	25,75 % 23,4 % -	150,- - -	28,3 % 20,8 % -	2,20 % 0,7-5 % -
MASSIVE SULPHIDE DEPOSITS S.L. IN VOLCANOSEDIMENTS	29,- - -	15,5 % 12,5 % -	37,- - -	7,2 % 8,5 % -	1,35 % 0,8-2 % -
Ni-Cu SULPHIDE DEPOSITS	6,5 - -	3,45 % 4,0 % -	9,- - 15,4	1,75 % 3,0 % 2,8 %	0,70 % 0,3-1 % 0,42 %
MISCELLANEOUS	8,25 - -	5,1 % 6,9 % -	10,25 - -	3,0 % 4,1 % -	- - -
TOTAL	188,75 - -	100,3 % 100 % -	511,25 - 550,0	99,75 % 100 % -	1,03 % - -

Each square contains three lines. The figures are as follows :
- the first line is from the inventory of H. Pelissonnier 1972 ;
- the second line is from B.F.B. ("Kupfer" 1972) ;
- the third line is from the BRGM (Unpublished Inventories 1975-77).

1.1.2 World reserves

The results of the study of the literature on this subject are given in two tables. Some details and remarks are presented hereunder, and the question of copper resources is then briefly considered.

All studies on reserves at regional and, a fortiori, world scale, are confronted to the problem of definitions. It should be stated that the reserves given in the tables correspond roughly to measured and indicated reserves. The figures thus include all proved reserves, most of probable reserves and, no doubt, a small part of the possible reserves. In other words, these data include reasonably measured tonnages mineable under market and technological conditions not far from the present ones.

1.1.2.1 Table N° 2 : Copper mine reserves

This table gives the uncorrected results of a BRGM documentary study on the copper mines (situation in 1975). By copper mine, should be understood all mines in which copper is the principal metal or an important constituant, except for Ni-Cu mines, which are the subject of a separate study. Some well known prospects, not yet worked, are also included.

The number of mines inventoried is shown in the columns to the right. In the western market economy countries, these are most often mines but in the socialist countries a unit is most often a group of nearby mines integrated into a complex. In general, the figures for these countries are much less reliable because statistics are scarce.

1.1.2.2 Table N° 3 : World copper reserves

Some geographical units are wider than in Table N° 2, and the following remarks should be made :

- "Others - Latin America", includes Mexico, Central America, Caribbean Islands and all the countries of South

TABLE N° 2

WORLD COPPER MINE RESERVES

(expressed in million tonnes
contained in copper mines now being worked or in preparation at the end of 1975)

COUNTRIES REGIONS or CONTINENTS	ORE TONNAGE	AVERAGE ORE GRADE	COPPER CONTENT	% WORLD TOTAL	PRINCIPAL ASSOCIATED BY-PRODUCTS	NUMBER OF MINES		
						being worked	in prep.	TOTAL
US	14 882,0	0,56 %	82,9	17,2	Mo,Ag,Au (Zn)	83	17	100
CANADA	4 763,2	0,51 %	24,5	5,0	Mo,Zn,Au,Ag,Pb	69	15	84
MEXICO	2 731,9	0,70 %	19,2	4,0	Ag(Mo)	10	3	13
OTHER CENTRAL AMERICA (+ CARIBBEAN)	3 593,7	0,52 %	18,8	3,9	Au,Ag,Zn	6	6	12
TOTAL CENTRAL AMERICA	6 325,6	0,60 %	38,0	7,9		16	9	25
PERU	4 142,5	0,70 %	29,1	6,0	Mo,Ag,Zn,Pb	24	5	29
CHILE	9 085,8	1,02 %	92,9	19,3	Mo,Ag,Au	39	10	49
OTHER SOUTH AMERICA	2 311,0	0,56 %	13,0	2,7	(Mo)	10	3	13
TOTAL SOUTH AMERICA	15 539,3	0,87 %	135,0	28,0		73	18	91
ZAIRE	709,5	4,07 %	28,9	6,0	Co (Zn,Pb)	22	2	24
ZAMBIA	1 100,5	2,59 %	28,5	5,9	Co	21	9	30
SOUTH AFRICA	823,4	0,51 %	4,2	0,9	Nb,Ta,P-(Au,Ag,Zn)	16	1	17
OTHER AFRICA	102,1	2,94 %	3,0	0,6	(Ag,Pb,Zn,Co)	23	9	32
TOTAL AFRICA	2 735,5	2,36 %	64,6	13,4		82	21	103
IRAN	1 635,5	0,78 %	12,8	2,7	(Mo)	1	9	10
OTHER MIDDLE EAST + SOUTH ASIA	946,1	0,77 %	7,3	1,5		10	12	22
CHINA,KOREA,MONGOLIA	700,0	0,88 %	6,2	1,3		11	2	13
JAPAN	550,2	1,07 %	5,9	1,2	Zn-Pb(pyrite-Fe)	18	-	18
TOTAL ASIA	3 831,8	0,84 %	32,2	6,7		40	23	63
NEW GUINEA ISLANDS	1 380,3	0,55 %	7,6	1,6	Au(Ag)	3	3	6
PHILIPPINES	2 915,3	0,45 %	13,1	2,8	Au,Ag(Zn,Mo)	33	11	44
AUSTRALIA	157,5	1,58 %	2,5	0,5	Pb-Zn(Au,Ag,Bi)	16	5	21
OTHER OCEANIA	206,0	0,73 %	1,5	0,3	Ag,Au	4	2	6
TOTAL OCEANIA	4 659,1	0,53 %	24,7	5,2		56	21	77
USSR	2 017,0	1,86 %	37,5	7,8	Mo	28		28
POLAND	1 036,0	1,76 %	18,3	3,8		5		5
RUMANIA	2 000,0(?)	0,33 %	6,6	1,4		2	2	4
YUGOSLAVIA	1 370,6	0,60 %	8,2	1,7	Ag,Au	5	1	6
OTHER EAST EUROPE	625,0	0,49 %	2,8	0,6		9	2	11
TOTAL EAST EUROPE (without USSR)	5 031,6	0,71 %	35,9	7,5		21	5	26
EEC	105,3	0,86 %	0,9	0,2	Zn,Pb,pyrite(Sn)	7	1	8
OTHER WEST EUROPE	645,1	0,82 %	5,3	1,1	Zn,Pb,pyrite(Au,Ag)	34	2	36
TOTAL WEST EUROPE	750,4	0,83 %	5,2	1,3		41	3	44
WORLD TOTAL	60 535,5	0,80 %	481,5	100,0		509	132	641

America, except Chile and Peru.

- China block includes China, Mongolia and North Korea.
- Cyprus and Turkey are included in "Other western European countries", as in Table No. 2.

The columns can be divided into three parts : the three columns to the left show elements included in total reserves :

- Column No. 1 gives figures from Table No. 2 (copper mines);
- Column No. 2 gives figures which have been corrected in terms of two criteria :
  - . copper is a secondary but recoverable by-product of many mines, not included in Table No. 2. The correction is approximate, but was made on the basis of certain examples for which the by-product copper/total copper ratio is known, and geological and mining criteria.
  - . the quantity and quality of available information varies a great deal, depending upon the country concerned. In addition, there were some omissions in the general study, which is summarized in Table No.2 and new discoveries have been reported since.
- Column No. 3 concerns copper reserves for Ni-Cu sulphide deposits (Type 4), which are the subject of a separate inventory. Certain figures in this column correspond, at least for a small part, to subeconomic reserves (for example, part of the Duluth complex in the U.S.).

The three center columns are the general results of the BRGM survey, with :

- Column No. 4 : total reserve (columns No. 2 and No. 3 added),

TABLE N° 3

WORLD COPPER RESERVES (In million metric tonnes of copper content)

COUNTRIES ZONES or CONTINENTS	B.R.G.M. figures (1975-77) Copper Mines Table n°	Revised figures	Ni-Cu Deposits	TOTAL RESERVES	% WORLD TOTAL	RESERVES PRODUCTION RATIO	OTHER PUBLISHED FIGURES B.F.B. 1971 (Kupfer)	U.S.B.M. 1975 (Min.Facts Prob.)	U.S.B.M. 1977 (Comm. Data Sum.)
US	82,9	88,1	3,4	91,5	16,7	65	77,1	82	93,0
CANADA	24,5	29,3	6,2	35,5	6,4	47	33,9	36	34,4
NORTH AMERICA	107,4	117,4	9,6	127,0	23,1	59	111,0	118	127,4
CHILE	92,9	102,0	-	102,0	18,6	116	63,0	82	93,0
PERU	29,1	32,0	-	32,0	5,8	149	22,7	27	32,9
OTHER-LAT.AMERICA	51,0	56,0	-	56,0	10,2	543	10,2	27	*
LATIN AMERICA	173,0	190,0	-	190,0	34,6	159	95,9	136	
ZAMBIA	28,5	29,0	-	29,0	5,3	42	27,2	27	31,8
ZAIRE	28,9	31,0	-	31,0	5,6	63	19,1	18	28,2
OTHER AFRICA	7,2	8,2	0,8	9,0	1,6	30	7,0	9	* 91,0 *
AFRICA	64,6	68,2	0,8	69,0	12,5	47	53,3	54	
AUSTRALIA-OCEANIA	24,7	26,9	0,1	27,0	4,9	43	10,9	17	36,7
SE ASIA-MIDDLE EAST	26,0	28,0	-	28,0	5,1	117	18,7	?	*
"CHINA BLOCK"	6,2	13,0	?	13,0	2,3	110		?	o
TOTAL ASIA	32,2	41,0	?	41,0	7,4	115	43,5	27	25°
USSR	37,5	45,3	4,7	50,0	9,1	65		36	40
OTHER EAST EUROPE	35,9	39,5	-	39,5	7,2	81			o
EEC	0,9	1,0	-	1,0	0,2			18	*
OTHER WEST EUROPE	5,3	5,3	0,2	5,5	1,0	32	4,7		*
TOTAL EUROPE	78,7	90,1	4,2	95,5	17,5	67		54	
WORLD TOTAL	481,5	534,6	15,4	550,0	100,0	76	338	406	506

- Column No. 5 : percentage of each geographical unit in relation to world total,
- Column No. 6 : reserves/production ratio. The production considered is an average of annual production of copper content for the last three years (1974-1976). The figures in this column are therefore for reserve life, at a constant production rate. A forecast can be made on this basis, in terms of the annual rate chosen for production growth.

The three columns to the right give reserve figures published by other bodies in recent years. On the whole, reserves have grown rapidly, so that their lifetime remains constant. In details, some disparities can be observed, which are probably due to differences in information sources, but the rapid growth of reserves in the South American Cordillera, Oceania and the eastern European countries, appears clearly.

1.1.2.3 Remarks on resources

The notion of resources, which used to include all concentrations, whether they were explored or not, and whether workable or not, is changing and becoming less comprehensive for certain authors. We will exclude here economic reserves, and resources will then include subeconomic reserves as well as hypothetical and speculative ones [6] .

The subeconomic reserves are rather precisely measured, but as far as we know, no detailed inventory on the world scale has been published, and it is impossible to give even approximate figures.

The U.S. Bureau of Mines has published general figures for resources, and H.J. Schroeder [7] gives an estimate of about 1,500 Mt. This is only an order of magnitude since the zone distribution seems to have been arrived at by extrapolation of reserves corrected in terms of the degree of available knowledge, and the extent to which the geological context is promising. It is thus only an approximate forecast showing that the relative importance of South America and Europe could decline and that of Asia increase, in the remote future. The USBM table also shows polymetallic nodules as a new copper source.

Copper resources in manganese nodules on the seafloor

The "copper dossier" deals with the copper in the EEC countries. Nevertheless we will briefly summarize some data about seabed mining which could have some medium and long term effects on copper reserves and prices and could constitute a supply source for the EEC.

Because of the deadlock in the Law of the Sea negotiations, the legal situation is not known. It is probable that the sea zones under national jurisdiction will be uniformly extended to 200 nautical miles from the coast. The international zone thus established constitutes a surface of about 260 millions km^2 and contains almost all known fields of nodules. In this zone, the legislation for future seabed miners is completly uncertain.

Technical aspects present a clearer picture but many figures about ore bodies, mining and processing are still doubtful.

Prospection results can be summarized as follows :

+ nodules are spherical or ovoïd concretions with a diameter from several millimeters to several centimeters which overlie in a single layer the sediments of the abyssal plains. More rarely they occur in crust form. Their origin is still doubtful and four theories emphasize the role of seawater solution, volcanism, micro-organic-action and diagenesis.
+ Location : nodules are found almost everywhere on the deep seafloor but fields with high density of nodules are found mostly in the Pacific and the Indian Oceans.

It is very difficult to be more precise at the present time because the US groups, very active in this field, have not published recent data. But it seems that the best known promising area is in the Pacific Ocean where it covers a surface of about 8 millions km^2, between 7 and 15° N and 120-160° W.

+ <u>Grades</u> : generally speaking the Mn content of the nodules are between 15 % and 25 % and the iron content between 7 % and 20 %. The average Mn/Fe ratio seems to be higher in Pacific Ocean (19/15) and in Indian Ocean (18/16) than in Atlantic Ocean (16/21).

  Minor elements contained are Ni, Cu, Co, Ba, Zn, Mo. Average grades in percent of dry weight are 0.4-1.3 % Ni, 0.15-1.15 % Cu and 0.2-0.8 % Co.

  In a southern belt of North Pacific of more than 1 million km^2, there seem to be large surfaces containing nodules with 1.2 % Cu, 1.4 % Ni and 0.2 % Co. Grades of over 3.25 % Ni + Cu have been found locally. But no mine site has yet been identified since exploration is still insufficient.

+ <u>Mining methods</u> have been studied and proposed. It is better to use the word dredging for an operation which consists to collect nodules froms selected areas of the seafloor.

  Most of the economic studies consider that a mineable ore body should be around 50-75 Mt with 2.5 % base metal content to dredge at the annual rate of 3 Mt. A mine site should be then an area of 30,000 km^2 with a concentration of 10 kg of nodules per m^2 grading about 1.2-1.4 % Ni, 1-1.2 % Cu and 0.15-0.25 % Co.

  But this definition is much theoretical. Concerning the minimum concentration of 10 kg nodules per square meter,some definitions speak in dry weight, some others in wet weight (30 % difference). Cut-off grades are sometimes known (2 % Ni + Cu + Co), sometimes not specified. In an eventual mine site, the percentage of the area unsuitable for mining due to outcrops is still very speculative. It is the same for the percentage of recoverable nodules, whatever the type of collector is (buckets, hydraulic types, ...). In conclusion, the lack of experience lead to a cautious attitude and, according to Metallgesellschaft, exploitation possibilities are far from brilliant.

+ Resources : they can not be considered as reserves, at least not before a necessary technological progress in knowledge (prospecting) and sampling and mining methods.

High figures have been proposed but without any serious basis. H.J. SCHROEDER (7) put forward the figure of 400 Mt of coppercontent, which can be compared with other data such as 250 Mt (M. DUBS KENNECOT. 1974), 46 Mt for the first generation of operations and 275 Mt for the second (A.A. ARCHER, IGS, 1973). The latter estimates were considered excessively prudent by the US delegation to the Conference on the Law of the Sea. According to these sources, it can be supposed that nodule copper resources are roughly between 250 and 500 Mt, i.e. between one half and total known reserves on continents. Other much less optimistic figures were suggested by some Europeans. Afernod (8) speaks of 24 Mt for example.

These figures, very different from ones to others, show clearly the uncertainty about fields of nodules. Whatever the truth may be, the possible influence upon copper production before the year 2000 will not be very great, in view of the necessary technological progress and the large investments involved. It was calculated that 50 units each producing annually 3 Mt of dry nodules and recovering 90 % of the copper content could supply only 7 % of the 1990 world copper consumption. Nevertheless long term potentialities are far from negligible and European countries must participate in this field to the development of knowledge of nodules and seabed mining technology.

1.2 Deposits and occurrences in the EEC

There are very few active copper mines in the EEC. Avoca, in Ireland, is the only one whose principal product is copper. Elsewhere, copper is a byproduct of tin, as at the Wheal Jane (U.K.), of Pb-Zn at Rammelsberg (West Germany) and Funtana Raminosa, Campiglia Marittima and Fenice Cappane (Italy) and, lastly, of gold and bismuth at Salsigne (France). These mines, along with some recent interesting mines such as Gortdrum in Ireland, are well known and economic and geological data are easily available.

Table No. 4 shows that the present copper mine production in the EEC is very low, and this is also true for the reserves shown in Table No. 5, especially if the figures are compared with the world figures in the preceding tables.

However, Table No. 4 also shows that accumulated production, for certain countries, is far from negligible. The usual statement that there is no copper in western Europe should therefore be questioned, especially since the study of the geological characteristics of the EEC, which is summarized below, shows that there are many contexts which are theoretically promising. For these reasons, and in accordance with the orientation of this report towards future perspectives, we have, in the following pages, given more emphasis to favorable indices and districts than to deposits which are either exhausted or now being worked, particularly as the relevant data is often very rich and easily available.

We will consider the cupriferous districts of the EEC countries according to their order from North-West to South-East, as follows :

- Greenland (Denmark)
- Ireland
- United Kingdom
- Benelux countries
- West Germany
- France
- Italy

We will then briefly describe copper mineralizations in territories belonging to the EEC, but located outside of Europe.

1.2.1 Denmark (Inset fig. 1)

There are no base metal occurrences on the European territory of Denmark, except - perhaps - for the little island of Bornholm, and there seems to be no chance of discovering copper. The case is different for Greenland, which is partly linked to Europe in view of certain geological characteristics. For this reason, we will consider it here, while other extra-European territories of Community members are treated separately.

Greenland is a large island of 2,186,000 km^2, but more than 80 % of its surface is covered by inland ice and glaciers. The zone which can be prospected is therefore limited to 350,000 km^2, and even to 250,000 km^2, if the part located to the North of the 75th parallel is eliminated, where the climate reduces the working period to several weeks, or even several days per year.

The Geological Survey of Greenland and the Danish mining companies have long been at work on the geological study and systematic prospection of the country. The general results have recently been published , and we will describe the cupriferous mineralizations in accordance with the principal geological units which are distinguished.

1.2.1.1 The Archean Craton

The craton crosses the southern part of Greenland between 61° and 66° North, and can be linked to similar complexes in Labrador and the North-West of Scotland. To the West, it contains large iron deposits (magnetite, itabirite at Isua), and chrome-bearing horizons (Fiskenaesset basic rock). Ni-Cu-Pt sulphide deposits have also been found on the West coast, towards 65° North, to the South of Sukkertoppen, between Fiskefjord and Sondre Isortoq.

There are many known norite bodies of variable size (0.1-4 km) in an arc of 75 x 15 km. About fifty deposits formed by magmatic segregation were prospected by aeromagnetic surveys, geological and ground-geophysical studies and boreholes. The largest mineralized bodies contain 2 Mt of disseminated ore and 200 to 400 kt of massive ore. The Ni/Cu ratio is 2 to 5, but the grades have not been published and it was not thought profitable to exploit them at the present time. There are also quartzose cupriferous veins enclosed in green schists, near the South-West border of the craton (Sermiligârssuk).

1.2.1.2 Lower and middle Proterozoic mobile belts surround the Archean craton. Three principal units are distinguished :

- The Ketitidian belt, to the South between Cape Farvel and 62°, contains, on its North-West edge, the famous cryolite mine of Ivigtut. The basement consists of large syntectonic and intrusive granite plutons in a folded and metamorphosed series of sediments, volcanics and basic intrusions. Many copper occurrences are associated with veins and faults contained in this series. In the middle Proterozoic, these formations were covered by a platform series with basalts, sandstones and conglomerates. The latter surround the Nanortalik occurrence, which consists of disseminated sulphides with local grades of 2 % Cu + Pb + Zn. These formations were later intruded by large alkali complexes to which are linked the Ivigtut cryolite and many occurences of U, Th, Be, Nb, F, etc.

The copper occurrences themselves, in this belt, are of little interest, but they indicate favorable contexts in which important mineralizations could have been concentrated by a complex geological history. The only copper workings in Greenland were in this area at the beginning of the century : Kobberminebugt, located in metavolcanics, produced 90 t of metal, and there still remain 2,000 to 3,000 t of ore containing 30-40 t of copper, while several other occurrences are known nearby. At Julianehaab, 15 t of high grade ore was extracted from the Frederik VII mine, which is linked to a quartzose pegmatite body.

- The Nassugtoqidian belt extends in a North-East South-West direction to the North of the craton, between 69° and 63°30'. It consists principally of refolded Archean series with some more recent patches. Many sulphide occurrences exist on the West coast between Holsteinberg and Umanak, and the most interesting of these are in the Lersletten region, where graphitic metasediments contain stratose lenses of dissiminated pyrite-pyrrhotite-chalcopyrite and sphalerite mineralizations. However, grades are low (maxima of 2 % Zn and 0.5 % Cu) and erratic.

- In the Rinkian belt further North, and on the West coast alone, no copper occurrences are known, but there is the Pb-Zn mine of Marmorilik (Black Angel), which is the only mine now active in Greenland.

1.2.1.3 There are two Paleozoic orogenic belts : In the North of Greenland, an East-West belt extends the Innuitian fold belt of the Canadian arctic islands. The climate, North of the 81st parallel, makes it very difficult to prospect this zone and no copper occurrences are known.

- The Caledonian belt occupies the northern half of the East coast and should be linked to the European Caledonides, from which it was separated by the opening of the Atlantic. Old little nuclei appear in the middle of Upper Precambrian and Cambrian-Silurian geosynclinal deposits, in the western part. The group was folded after the Middle Ordovician and later, in the East, molasse sediments accumulated, after which basins formed during a period which remained unstable from Devonian to Triassic times.

Stratiform cupriferous mineralizations were discovered rather recently (1973-1974) in this environment, at two levels : Upper Precambrian and Permo-Triassic.

The Upper Precambrian is represented by a very thick (15,000 m) and gently folded sequence, in which can be distinguished the Eleonore Bay Group (E.B.G.) and the Tillite Group above it. The three cupriferous horizons are in the upper third of the E.B.G. in shales, silts and quartzites, located between a thick detrital formation at the base and a carbonate formation at the top.

. The lower horizon is known to the North, over a limited surface, and contains pyrite and chalcopyrite (0.5-2.5 % Cu) dissiminated through 5 to 20 cm of quartzose schist.

. The middle horizon, which appears in some sections to the South, consists of silty gray-green shale with little chalcopyrite (0.5 % Cu max.) over 10-20 cm, and sometimes up to 1 m.

. The upper horizon is thicker and has been explored over more than 100 km. A sequence of 20 to 50 m of variegated siltstones and quartzites, has 1 to 5 mineralized levels of various thicknesses (0.2 - 1 m and sometimes 5 m) with tetrahedrite, chalcopyrite and native Ag. Locally, grades of 6 % Cu have been found, but in the best studied zone, the average grade is 1.4 % Cu, 0.1 % Zn, 1.1 % Sb and 24 g/t Ag.

The Upper Permian and Triassic correspond to a transgressive period during which a thick series (1,000-2,000 m) with varied lithology was deposited. Stratiform Cu-Pb-Zn mineralizations were found at seven levels, and the four most important ones for copper are :

- The uppermost Upper Permian calcaro-dolomitic bioherm, with chalcopyrite-tennantite-galena-sphalerite-barite-fluorite infillings. This mineralization is known over

50 km^2, with a thickness of 0-50 m and contents of 1-2 % Cu, 1-25 % Pb, 0.1-0.3 % Zn, 25-150 g/t Ag, 11-12 % Ba and 0.4-1 % As.

- A lower Triassic arkose showing, over 100 km^2, two irregular mineralized horizons with thicknesses of up to 30 m. The disseminated chalcocite-bornite-galena ore has contents of 3 % Cu, 3 % Pb, 150 g/t Ag and 0.3 % Ba

- A Middle to Upper Triassic horizon with a thickness of 1-2 m, is known over more than 1,000 km^2 and contains low grade cupriferous mineralization (0.1-1 % Cu), with chalcocite and bornite disseminated in dolomite shales and fine gray or black sandstone.

- Lastly, in the Upper Triassic there are banks,in the meter range, of fine gray calcareous sandstone enclosed in red marl and containing,over more than 1,000 km^2, a spotty mineralization with little sulphur (native Cu, native Ag, domeykite, chalcocite). Rich samples contain 10 % Cu, 300 g/t Ag and 0.2 % As.

One can say, then, that on the East coast of Greenland there are stratiform cupriferous mineralizations, located in similar geological contexts and roughly at the same levels as those of the Central African Copperbelt and the European kupferschiefer. The most promising beds are the Upper horizon of Precambrian and the Permo-Triassic horizons 2 and 3, described above. These mineralizations, which have been prospected at a relatively slow rhythm because of the climatic conditions, have still to be studied in detail.

1.2.1.4 <u>Eocene magmatism</u> is the last important event for the Greenland metallogeny. It occurs around the 70th parallel on the East and West coasts, where it is indicated by the thick "basaltes de plateaux" and various intrusions, espacially in the East. The copper occurrences associated with this phase are insignificant, but the discovery of porphyry molybdenum at Malmbjerg (117 Mt with 0.15 % Mo), encouraged prospection for porphyry copper deposits and lodes associated with intrusions.

1.2.2 Ireland (Inset fig. 2)

The only active copper mine is Avoca, since the closing of Gortdrum in 1975. The enclosed map and list of copper mines and occurences do, however, show that much prospection and exploitation has been carried on in the past.

Rocks of the Irish Republic are of Pre-Triassic age, except for rare and small Mesozoic patches. In this essentially paleozoic context, it is possible to distinguish several group of cupriferous mineralizations, according to the type of deposit and the environment.

1.2.2.1 Metomorphic Calédonian belt in the North

Located to the North-East of a Galway-Belfast line, these formations made up of intrusive granites and gabbros, and of a gneisses,schists,quartzites,and limestones series, were folded and metamorphosed by the Caledonian orogeny. They outcrop in Donegal County in the North, where no copper occurrences have been mentioned, and in the coastal part of Connaught where 8 occurrences are known (No. 1 to 8). The latter were mined in the 19th century, but production was very low. The geological context is not well known, but there is a clear link between the presence of copper and faults or amphibolites. This type of deposit does not, therefore, seem very promising, but the following should be noted :

- There has been less prospection in this area than in others;
- Recently, discoveries in similar contexts have been made in Scotland;
- Prospection guides exist : basic volcanics and Dalradian marble horizons, either crossed by North-West faults or intruded by granites, some containing dissèminated molybdenite (Sheskinarone).

1.2.2.2 Volcano-Sedimentary Caledonian belt of South-East

Roughly speaking, this part of the Caledonides consists of a Cambrian and principally Ordovician volcano-sedimentary series, which was folded and slightly metamorphosed at the end of the Ordovician. Late orogenic granites, including the large Leinster batholith, intruded a little later. There are two copper mining districts in this environment, Avoca in the North, and Bunhamon in the South.

The Avoca district (No. 9 to 15) has been mined for more than 200 years. The history of this mine and of the geological interpretations given by J.W. Platt are still very interesting, for it is often forgotten that the Avoca copper mine is, in fact, a polymetallic deposit with some Pb-Zn reserves. Much progress has been made in the understanding of the mine since its reopening in 1970.

The mineralized body is located in an Upper Ordovician acid volcanic series, with some metasedimentary intercalations. These formations have been isoclinally folded (North-East South-West direction, South-East dip) and have been affected by North-South faults. The details of these complicated folds has not been well explained. From North-East to South-West, can be distinguished the old Kilmacoo, Connary, Cronebane and Tigroney mines, forming the new "East Avoca" and then Ballygahan and Ballymurtagh, which make up "West Avoca". The whole structure is more than 4 km long and 500 m wide.

The stratigraphic sequence containing the mineralization consists mainly of pyroclastic material with discontinuous siliceous chemical deposits and small dolomitic shale lenses. Extrusive or intrusive rhyolitic complexes are intercalated in this series.

There are several types of mineralizations and, from the bottom to the top of the series, the following are found :

- Stratiform bands of copper-rich massive sulphides (3 % Cu) which have been exhausted, enclosed in altered volcanic ashes and black schists;

- Brecciated parts of the pyritic rhyolitic complex, locally containing a rich Cu-Zn-Pb mineralization (sometimes more than 10 % metal);

- At the summit, the ores now exploited are of two types : First, the stockwork, made up of small bodies and quartz veinlets with pyrite and chalcopyrite (0.7 - 1.1 % Cu) enclosed in volcanic siliceous schists, 15 to 30 m thick. Second, the massive banded ore in very chloritic and pyritic, finely bedded shales, with a thickness of 3 to 20 m and containing 1 to 1.5 % Cu. In the upper part of this horizon, the ore becomes richer in Pb-Zn, and this is also true laterally in dolomitic shales. This is the well known zonation of the volcanic-exhalative type.

Cronebane, a supergene enrichment zone (1.41 % Cu), was exploited by open pit methods from 1971 to 1975, and studies are now under way for exploitation by in situ leaching of the low grade sulphide parts.

Accumulated Avoca production amounts to about 75 kt of copper, and there are proved reserves of 8 Mt of ore with 0.7-0.8 % Cu, which is one of the lowest grades in the world now being mined by underground methods. The potential is much greater with at least 40 Mt, with a content of 0.45 % Cu for East Avoca alone.

The Bunhamon district (No. 16 to 19). The Avoca volcanic band is small compared with the belt from Arklow to Waterford extending over 130 km. The Bunhamon district is at the South-West end of the latter in Ordovician volcanics and shales. It was mined a long time ago (1730-1869), and only limited geological data is available. The primary Pre-Devonian mineralization is chalcopyrite and little galena in North-West South-East quartz veins, some of which have been worked up to a depth of 300 m. Total production amounted to 22 kt of metal extracted from a high grade ore (10 % Cu).

The little mine of Beauparc (27) and the occurrence at Salterstown (28), in the Lower Paleozoic North of Dublin, are comparable to these mineralizations, but very small.

Prospection was carried on for a long time up to quite recently in this very promising Ordovician volcano-sedimentary context. No economic discoveries have been made, but the potential is still high around Avoca and in the volcanic belts. Studies further defining the geology and metallogenic context (including magnetite, gold and lead-zinc) would probably be very useful for focusing further prospection. Diorite intrusions to the South-East of the granite of Leinster should be included in the possible metallotects, because of the resemblance to Coed y Brenin recently discovered in Wales.

1.2.2.3 Devonian sandstones of the South West (West Carbery District)

The Caledonian orogeny was followed by emergence and general erosion. The Devonian lies unconformably and includes principally the detrital, continental and deltaic, "Old Red Sandstone". To the South of the "Mallow Thrust Front" it has been folded by the Hercynian orogeny in an East-West to East-North-East West-South-West direction.

In this Hercynian part and to the South-West of a Killarney-Clonakilty line, there are many cupriferous veins which have produced about 27 kt of metal from 1810 to 1880.

Two regions are usually distinguished :

- The West Carbery region itself to the South, includes a very large number of veins and at least 20 small old mines (No. 29-47) on a very small surface (around 400 km^2). The ore consisted of chalcopyrite with a quartz --barite gangue. There was galena at Roaringwater (No. 46), and the iron cap of Kilcrohane (No. 44) contained gold. The principal mines seem to have been Coosheen (No. 36) and Horse Island (No. 42), especially in view of the high grades respectively up to 18 % and 55 %, and Kilcrohane (No. 44) and Mount Gabriel (No. 45), but total production was relatively small.

- The Bearhaven mines are in a surrounding zone (No. 48 to 54), where the veins are more scattered but also larger. Allihies to the West, was the most important mine, and from 1810 to 1883 an ore rich in chalcopyrite (10 % Cu) was mined. The veins are enclosed in Upper Devonian strata, and the largest (Mountain Mine) is more than 500 m long and 10 to 20 m thick and the mineralized columns were stripped to a depth of more than 500 m. To the North, at Ardtully (No. 53), a field of 8 veins contained a different paragenesis with bornite and tetrahedrite. To the East, copper was associated with manganese at Glandore (No. 50) and was a by-product of barite at Duneen (No. 51).

Attempts to resume work on the vein mineralizations have all failed, but reserves of 1.5 Mt with a minimum of 1.6 % Cu are known at Allihies, and there are also disseminated stratiform chalcocite, bornite, chalcopyrite and pyrite mineralizations, which have never been worked. The latter seem to be located principally, if not exclusively, in the Kiltorcan beds of the uppermost Devonian, which also contain most of the veins. There is no doubt a genetic link between the two types of mineralization, but geochemical prospection of the red beds is complicated by the presence of peat-covers, and geological studies do not seem to have defined paleogeographical or sedimentological guides.

1.2.2.4 The Central Plain

The gently folded and strongly faulted Paleozoic series consists of Silurian-Devonian anticlines overlaid by Carboniferous limestones. Base metal mineralizations have been actively prospected and recent discoveries are remarkable, especially for Zn-Pb. Except for small occurrences, copper is found in the central South-West part, between Galway and Glonmel where the following two types of mineralization are known :

- Veins enclosed in Devonian strata, which are comparable to those of West Carbery. Identical paragenesis are found mostly with chalcopyrite, but also with bornite and tetrahedrite at Pallaskenry (No. 22). The isolated Bolinglanna occurrence (No. 26) in the North-West, corresponds to veins in Devonian sandstones with chalcopyrite associated with galena and pyrite.

These small but often high grade Lower Paleozoic veins were mined in the 19th century. They have no economic importance, but help to explain the Post-Caledonian metallogeny of Ireland.

- Lower Carboniferous limestones. In this formation are located the principal Pb-Zn-Cu concentrations which, in the South-North order, are as follows : Mallow (Cu, Ag), Aherlow (Cu, Ag), Gortdrum (Cu, Ag, As, Sb, Hg), Silvermines (Pb, Ag, Zn, Cu), Tynagh (Pb, Zn, Cu, Ag), Moate (Zn, Pb), Keel (Zn, Pb, Ag), Ballinalack (Zn, Pb), Navan (Zn, Pb) and Abbeytown (Zn, Pb). The Pb+Zn/Cu, Zn/Pb and Cd/Zn ratios indicate a general zonation which has been emphasized, and the copper in these deposits is concentrated in the South-West, and a link, not yet well defined, exists probably between these copper deposits and the stratiform and vein copper in the

Devonian of the South-West of Ireland. This region can be described as a cupriferous belt.

- <u>Mallow</u> was discovered in 1973 by geochemical and induced polarization methods. Reserves are 3.6 Mt, with 0.70 % Cu and 25 g/t Ag. The deposit is located on the North limb of a faulted anticline and is enclosed in a series of 100 m of shales and sandstones and calcarenites of the lowermost Carboniferous limestones, which are transitional between the Devonian Old Red Sandstone and the massive Waulsortian limestone. Volcanic tuffs and breccias are known nearby. The mineralization, which is known over a length of 380 m and to a depth of 150 m, is located in shale beds themselves and in a calcite vein system. It consists mainly of finely disseminated chalcopyrite, bornite, chalcocite and tennantite with As, native Ag and traces of Pb and Zn.

- <u>Aherlow</u> has reserves of 5.5 Mt with 0.89 % Cu and 40 g/t Ag. There are many similarities between this deposit and Mallow as concerns the paragenesis and the host rock, but the role of structure is much more clear. The economic mineralization is located in a shear zone,cross-cutting subvertically 200 m of stratigraphic series. It is surrounded by a low grade disseminated chalcopyrite halo. There is little Sb, As and Hg, especially in comparison with Gortdrum.

- <u>Gortdrum</u> was mined by open pit methods from 1967 to 1975, and produced 40.7 kt of copper, 98.5 t of silver and 280 t of mercury from 3.35 Mt of ore with 1.19 % Cu and 30 g/t of Ag.

Mining faced troubles due to the presence of Sb (tetrahedrite)and As (tennantite) impurities and the unstability of the open pit walls which did not make it possible to extract the base of the mineralized body. The mineralization irregularly invades the lowermost 300 m of the Lower Carboniferous in the collapsed North limb of a large

North-South-West East-North-East fault.It shows clear vertical zonation with chalcopyrite-sulphosalts at the surface and chalcocite-bornite-cinnebar in depth. Tectonics are complex and involve several phases, including overthrusting towards the North. Basic rock dikes cross-cut the series.

On a more regional scale, Gortdrum is to the South-East of the acid and basic volcanic complex of Limerick, and several km to the West there is Oola (No. 61), which is another small copper deposit which was exploited in the middle of the 18th century.

- At Silvermines, copper is not recovered even as a by-product of the Zn-Pb-Ag ore, and is located at the base of the stratigraphic series. It is interesting to note that the southern compartment of the fault contains disseminated Cu in Devonian sandstones.

- At Tynagh, Pb-Zn production began in 1966, but copper is recovered only since 1973. From that date, 14.5 kt of copper concentrates containing about 2.8 kt of metal were extracted from ore located near the fault (0.2-0.3 % Cu in tennantite and chalcopyrite).

- Ballyvergin (No. 59) is another small deposit located to the West of the Tulla Volcanics and worked in the 19th century. It was prospected again in 1961 and a low grade disseminated chalcopyrite halo has been indicated around a high grade body (Cu, Pb, Ag).

- There are also two groups of occurrences which were once worked to the South of Killarney (No. 62-64) and to the North of Dublin (No. 65 and 66).

In conclusion, the Carboniferous deposits of Cu, Pb and Zn all have the following characteristics : they are located in the lower 400 m of the Carboniferous series

(Tournaisian) along and to the North of structures faulted East North-East. The role of the structure, as trap, is clear in the North and becomes less so in the South. The lithology of the country rock varies and does not seem to have a determining effect, but there is often a halo, always dolomitized, sometimes silicified and even chloritized at Mallow.

The paragenesis is essentially made up of primary Pb, Zn and Cu sulphides with barite and pyrite. The trace elements are Ag, Cd, As, Sb and Hg. Economically, the copper content in the Pb-Zn deposits is negligible (except at Tynagh) and vice versa, but there is regional zonation with a Cu-Ag (As, Sb, Hg) pole to the South-West and a Zn-Pb (Cd) pole to the North-East. Further the copper deposits are limited to the lower 200 m of the Carboniferous.

1.2.3 The United Kingdom (Inset fig. 3)

The present structure of the United Kingdom is dominated by Caledonian trends and, roughly speaking, North-East South-West formations divide the United Kingdom into increasingly young bands from the North-West to the South-East, from the Precambrian of Scotland to the Tertiary strata of the London basin. Geological contexts for cupriferous mineralizations vary with the geographical units and are described below in this order.

1.2.3.1 The Precambrian Basement

The United Kingdom presents a complete Caledonian orogenic belt between the two Precambrian shields. To the South-East, St-George's Land outcrops only in the Leicester region through some small windows in the Mesozoic cover. To the North-West, the Hebrides shield occupies a larger surface in the Hebrides Islands, and in Scotland to the West of the Moine Thrust.

The Lewisian is an old gneissic complex of varying composition and origin, containing some small base metal occurrences which seem to be without interest. The most promising mineralizations in the sector are, in comparison with Scandinavia (Laisvall), those of the Epi-Lewisian cover : Torridonian in broad subhorizontal zones and Cambrian following a narrow band between two major overthrusts (Moine Thrust and Sole Thrust). These sediments contain copper occurrences : disseminated chalcocite in Torridonian shale and sulphides in Upper Cambrian limestone.

1.2.3.2 The Caledonian belt

The Caledonides outcrop extensively in Scotland, Ulster, the North-West of England and Wales, and the orthotectonic belt in the North in the Highlands and the paratectonic belt in the South are usually distinguished.

In the North, the metamorphic series located between the overthrust over the shield and the Highland border fault includes the Moinian of Precambrian age to the North and the Dalradian of Upper Precambrian to Cambrian age in the South. This series is unconformably overlied by Devonian sandstones and traversed by numerous intrusions.

- The Moinian, which is very metamorphic, is composed principally of arenaceous and argilaceous rock with some carbonate horizons. Only negligible copper occurrences have been found until now.

- The Dalradian is less metamorphic and, from top to bottom, appears to consist of a series of quartzites with a tillite level, a carbonate sequence and overlying volcano-sedimentary formations. Copper occurrences, as well as small deposits of various types are known there.

- Small Ni-Cu lenses to the South-West linked to the basic rock : Inveraray.

- Copper and polymetallic (Pb-Zn) veins and bodies, along a central North-East South-West belt containing various occurrences, including Kilfinan in the Middle Dalradian limestone and extending into the Shetland Islands and Ulster.

- Various disseminated Cu or Cu-Mo mineralizations associated to the Newer Granites of Scotlant and recently described : Ballachulish Granite (Cu-Mo) Kilmelford (Cu-Mo) in Argyllshire and Tomnadashan (Cu) further East.

- Occurrences in Old Red Sandstone (ORS), especially in the Orkney Islands and on both sides of Moray Firth. Mineralizations are syngenetic and linked to fluviatil and lacustrine facies of the middle Old Red Sandstone, but in spite of active uranium prospection, only small occurrences have been found (Cu, Ba, Pb, Zn). The Sandloge (Shetland) little mine exploited copper in carbonate veins linked to a Post-Devonian magmatic activity.

The most promising strata in the metamorphic Caledonides is the Upper Dalradian calcareous and volcano-sedimentary series, as in the Irish Republic. Drill-holls have recently been made in the Shetland Islands on cupriferous occurrences located in this geological environment at Vidlin.

- The Midland Valley is a collapsed zone between the Highland border fault to the North and the Southern Upland fault to the South. On its North edge, it contains a broad Devonian band which extends into Ulster to the East of Lower Lough Erne. These formations are invaded by volcanic rocks in which are enclosed the small lodes Bridge of Allan (Cu, Ba) and Alva (Ag, Co, Cu) considered as telethermal and late Hercynian.

- The South part of Caledonian belt includes several districts to the South of the Southern Upland Fault and in the West of the United Kingdom : the Southern Uplands and their extension to the South of Belfast, the Isle of Man, the Lake District and part of Wales. They consist essentially of a Cambrian-Ordovician series, but reworked Precambrian patches appear in Wales, on the one hand in the East faulted zone, and, in the other hand, in the extreme North-West connected with the geanticline of the Irish Sea. Along this North-East South-West axis, the thickness of the series decreases and there is much volcanism.

- In the Southern Uplands and Ulster, the mining districts of Leadhills, Newton Stewart, Down and Monaghan consist of small veins in Cambrian-Silurian anticlines. But these mineralizations are considered of Post-Caledonian age, and copper often occurs as traces in lead-zinc ore.

- In the Isle of Man, the two deposits of Laxey and Foxdale are similar, but the country rock is solely Ordovician (schists and graywackes) and tetrahedrite is supposed to be more abundant.

- The Lake District is also principally lead and zinc-bearing with barite, but copper has been worked in many small mines including Coniston. The many veins in the district are in the Ordovician strata, consisting of schists, sandstones and tuffs of Skiddaw, and the thick volcanic series (basalt to rhyolite) of Borrowdale. Copper is known in many veins, but there were only two productive mines :

. Roughton Gill, in the North of the district, is a vein more than 3 km long which cross-cuts the Skiddaw-Borrowdale contact. It has produced approximately equal quantities of Cu and Pb (15 kt of concentrates);

. Coniston is a group of small mines in the South of the district. Veins of quartz-pyrite-chalcopyrite (+ Pb, Zn and traces of Bi, Ni, Co) are enclosed in the volcanics and especially the rhyolites of the Borrowdale series. The richest veins were mined (1854-1919) to a depth of more than 400 m. Recorded production was 53 kt of concentrates, with 5-13 % Cu obtained from a high grade ore (3 to 9 % Cu in the vein) and containing about 5 kt of metal.

All the veins are considered as of Post-Caledonian age but the Caledonian copper, whether stratose or in veins, is closely associated with Ordovician volcanism, and the hypothesis has been formulated that this spatial link indicates a genetic link.

- In the North-West of Wales, there are many cupriferous mineralizations, which are sometimes large and of various types :

. The peripheral Pb-Zn (Cu) lode districts of Llanvrst Berwyn, Shropshire and Plynlimon are analogous to the preceding ones;

. To the North-West there are more cupriferous veins which, from the South-West to the North-East, include : Llanengau inthe Ordovician base sediments, which has produced around 1 kt of Cu for 15 kt of Pb-Zn-Ag; Drws y Coed, where 15 kt of copper were extracted from four veins encased in Ordovician schist and Llandudno to the North.

- The disseminated mineralization of Coed y Brenin. There, the superficial cupriferous peat was worked a long time ago and the prospect has been studied again by Rio Tinto Zinc since 1970. Drill holes indicated 200 Mt with a content of 0.3 % Cu. This mineralization is disseminated in Upper Cambrian Ffestiniog Beds (argillaceous sandstones and banded greywackes), and in intrusive dioritic sills. It has been interpreted as porphyry copper since the sulphides, including chalcopyrite, are contained in microspecks and veinlets, and there are traces of Au and Mo and zones of porphyritic, phyllitic and argillic alteration. However, there is neither considerable silicification nor potassic alteration and the question is still open.

Although it was not possible to mine for economic and ecological reasons, this discovery remains an interesting one for it shows that Cambrian greenstones and associated volcanism can contain Cu, Au (and Fe) deposits.

- Parys Mountain, the volcano sedimentary deposit in Anglesey, is probably the largest copper mine in the United Kingdom. It was mined from 1773 and the exact production figures are not known. A minimum figure of 74 kt of metal has been given, but the figure of 130 kt of Cu has also been proposed solely for ores (3 to 7 % Cu), and without taking into account precipitates recovered from mine water which represent 19 kt. In addition, minimum reserves of 160 kt of metal content in 30 to 40 Mt of ore with 0.41 % Cu, have been indicated by drilling. The metal stock of this mine (production + reserves) is therefore about 300 kt.

There are considerable Precambrian outcrops in the Island of Anglesey, overlaid by an Ordovician and Silurian volcano-sedimentary series with doleritic sills. The detailed structure and stratigraphy are still not precisely known at Parys and Mona. The mineralization is located on the North limb of a syncline oriented towards the South, in Upper Ordovician and Silurian volcanics and shales, and extends over 1,800 x 400 m. It has been worked to a depth of 300 m. The whole is approximately conformable and includes various types of ores : the cupriferous and silicified stockwork located in Ordovician felsite and the "bluestones", which are silicified Silurian black shales impregnated with sulphides (with up to 30 % Zn, 10 % Pb and 1 % Cu), constitute the main ores. However, quartzose lenses located at the Ordovician felsite-shale contact have been worked,and there are also sulphides disseminated through acid volcanics (mainly pyrite) and in a chloritized green rock sill (pyrite and chalcopyrite).

In view of its context, form and mineralizations, Parys Mountain is comparable to Avoca in Ireland, since the two deposits are massive sulphide deposits s.l. linked to volcanism.

1.2.3.3 Cornwall and Devon

This well known tin and copper belt has a very long history and was even worked in the prehistorical period, and has played a considerable role in the mining development of England. It is the largest region of this type in the world, with nearly 2,000 mines and a production estimated at 2 Mt of cassiterite and 200 kt of Pb + Zn. Recorded copper production for the principal mines since 1815, amounts to 410 kt of metal, while the total production has been estimated at 1.2 Mt (Dines , 1956) and the metal content at 1.4 Mt (Pelissonnier, 1972). Wheal Jane, a rather recent mine in the belt, is the only deposit now producing copper in the United Kingdom. Its resources amount to about 15 kt of metal, which is approximately equivalent to the Mount Wellington's reserves (new project).

The geology of this province and its mines is thus very well known and there are many publications on the subject. It is also one of the regions in the world in which there has been the most prospection and it is not probable that there will be significant discoveries. For this reason, we will limit ourselves to a brief review of the principal characteristics.

Like the South of Ireland, this region was affected by the Hercynian orogeny. Devonian shales and sandstones were deposited in a mobile belt, and there was also underwater volcanism, especially at the end of the period, as well as greenstone intrusions. Following the Culm deposit, these formations were folded and slightly metamorphosed and then intruded by a batholith which outcrops as eight intrusives granites with concentric structure. Porphyry dikes indicate the latest phase of activity.

The very numerous mineralized veins and greisen, pipes and pyrometasomatic deposits are located near the roof of the batholith most frequently outside of the granite, but the genetic link is very clear.

Around the granites, there is a clear zonation between the tin in the center and in depth, and the copper on the periphery and closer to the surface. The Cu/Sn ratio varies greatly for the same structure. The main copper producing districts were from West to East, St Just, Camborne and St day, Par (St Austell), Caradon (Liskeard) and Callington and

Tavistock. For the period since 1815, 20 to 25 mines had a production of more than 5 kt of Cu.

In Devon, a stratabound cupriferous body is known to exist to the North of Belstone-Ramsley. This body in Lower Carboniferous host-rock has been interpreted as a skarn. Small manganese deposits are indicated at the same level, and the transitional series between Devonian and Culm corresponds to maximum volcanic activity. By analogy with the South Spain belt of Huelva and the closer Brittany, the possible presence of deposits associated with volcanism can be examined. It should also be noted that the iron deposits in northern Devon contain a little copper.

1.2.3.4 Carboniferous limestones outcrop from the Midland valley in the North to Derbyshire, in the South-West of Ulster and around the periphery of Wales, in the South. In Ulster, they are covered by glacial formations and prospection has not yet been completed. Irish zonation also makes it possible to suppose that there is a better chance of finding zinc than copper. In England, the Carboniferous limestone contains important Pb-Zn, Ba and F mineralizations in the North Pennines and Derbyshire districts. Cupriferous mineralizations are, however, discrete. The occurrences at Middleton Tyas to the North-East of the Askrigg Block, near the Permian cover and especially the Ecton mine in the South-West of the Derbyshire district, should be mentioned. About 60 kt of high grade ore were extracted there, with a copper content of approximately 8 kt Cu. Lastly, there seems to be no copper in the Pb-Zn district of the North-East of Wales.

1.2.3.5 The Permo-Triassic Strata of the Center

- The North-South Permian belt from Nottingham to the North Sea. During Zechstein time, the transgressive sea covered Yorkshire, and several authors have formulated the hypothesis that the Central European kupferschiefer might reoccur at the base of the Permian magnesian limestone. The latter consists of alternations of dolomitic and argillaceous limestone and of marls with small quantities of evaporites. The lithology seems promising and we have seen that the underlying Carboniferous contains copper at Middleton Tyas. In the Permian itself, Cu, Pb, Zn indices were found in basal schisty marls and the overlying magnesian limestone. They are accompanied by trace elements (Mo, V, Co, U), recalling the paragenesis of kupferschiefer, but these mineralizations are very minor occurrences.

- Cheshire : The well individualized Triassic basin to the East of Liverpool contains some old copper mines located in the area of transition between Keuper sandstones and marls. The mine of Alderley Edge is the largest with a production of 170 kt of ore with 2.1 % Cu (1857-1877). The Cu-Pb-Ba mineralization (with many trace elements) is disseminated in three layers of "Waterstones" conglomeratic sandstones (transition series). During remobilisation stage, faults acted as traps and became the principal metallotects at Grinshill, a small Cu-Ba deposit in the South. There are other occurrences on the South-West border of the basin.

- Outside of Cheshire, the Triassic contains many occurrences and small Pb-Zn, Ba mines in Midlands, Mendip Hills South of Bristol and in Glamorgan Vale in the South of Wales. Copper occurrences are small but quite numerous, especially to the West of Leicester. It is evident that all these mineralizations, whatever their form may be, are located near the limit of the Upper Triassic saliferous basin : at the edge of sandstone facies and evaporite series which could constitute a regional guide.

1.2.4 The Benelux Countries (Inset fig. 4)

Belgium has a large copper processing industry, using only imported concentrates and blister with Zaire as the largest supplier. There are no copper mines which are actually being worked in the Benelux countries, and past production was negligible in Belgium and Luxemburg, and nil in the Netherlands. There are however several old mines and occurrences which are described hereunder.

- Sedimentary formations in the Benelux countries can be divided into three groups separated by two principal unconformities :

. The Cambro-Silurian,affected by the Caledonian orogeny, outcrops in a narrow band from Charleroi to Liege and in some massifs of the Ardennes at Rocroi, Givonne, Serpont and Stavelot. The latter is traversed by the only known intrusions in the Benelux countries (tonalite-diorite);

. The Devono-Carboniferous crops out extensively to the South of the East-North-East valleys of the Sambre and Meuse, and has been relatively strongly folded during the Variscan orogeny.

- The most recent formations are horizontal or slightly rolling. It is possible to distinguish a Permo-Triassic and Liassic unit overlying the Ardennes to the South of a Sedan - Diekirch line, and the post-Liassic cover of the North overlying the massif of Brabant and the Flanders-Netherlands group.

The latter unit is barren and, for this reason, there are no known base metal occurrences in the Netherlands, except for the South part of Dutch Limburg. There, disseminated Cu-Pb-Zn mineralizations are known in the walls of faults in Westphalian slates and sandstones.

- Among small Cu deposits and occurrences in the South of Belgium and Luxemburg, it is possible to roughly distinguish three types :

. Two small Devonian deposits, which were once mined and are encased in detrital formations :

- Rouveroy, near the Franco-Belgian frontier, 10 km South of Mons, consists of copper oxide ores infilling Lower Devonian sandstones joints. Production goes back to the 19th century, and amounted to about 5 kt of copper. The whole ore deposit has been estimated at 2 Mt of ore with 1 % Cu.

- Stolzembourg is located in Luxemburg, near the German border and to the North-North-West of Diekirch.This ore deposit consists of some small NNW-SSE veins enclosed in Eodevonian shales. The paragenesis mainly consists of chalcopyrite, pyrite, ankerite, barite with some quartz, silver and zinc.

These veins are known since the 17th century and were mined again during brief periods in the 19th and 20th centuries. The main lode was intensively explored, but production was very low and can be estimated at 200 tonnes metal from a rich ore (5 - 20 % Cu).

Nearby, there are two other small lodes in the same formations, which are approximately parallel : the barite vein of Bivels (Luxembourg)and the quartz-chalcopyrite and barite vein of Bauler (Germany).

These sporadic mineralizations recall those of the South-West of Ireland, but they are much smaller and there is no associated tratiform dissemination.

- Copper sulphide veinlets, often associated with galena and sphalerite have been indicated by coal mining in the coal basins. As in the case of Limburg in the Netherlands, they are encased in Namurian and Westphalian detrital formations.

- There are two types of occurrences in the massif of Stavelot : some quartzose veins contain sulphide and oxide copper. The latter case is known principally in the South of the massif (Lierneux and Salm occurrences) near the manganesiferous district of the Lienne river.

- The sill-like intrusion of La Helle, located near the Belgian-German frontier, took place in Cambrian quartzite and contains a disseminated Cu-Mo mineralization. The chalcopyrite and molybdenite are in a network of quartzose veinlets in surrounding tonalite and quartzite. Some recent drill holes have shown low grade ores : 0.1 - 0.2 % Cu. An analogous mineralization is thought to occur in a similar intrusion located at Lammersdorf, in Germany.

The latter mineralization suggests porphyry copper type aside from the question of economic importance. The Devonian occurrences seem the most promising and the lower detrital series with red formations should be studied. No copper is associated with the large Pb-Zn deposits in Devonian and Dinantian limestones. Prospection results concerning the only known copper occurrence in this environment (Dolembreux S of Liege) were negative.

1.2.5 West Germany (inset fig. 5)

At the present time Rammelsberg is the only mine in West Germany producing copper. It has a production level of 1,500 to 2,000 tonnes per year as by-product of the zinc and lead ore. This mine probably holds the world record for its lifespan since it has been worked without interruption for more than 1,000 years (it was probably opened in 968). At the present production rate, however, it should be exhausted around 1985.

Nevertheless future perspectives in West Germany are probably more encouraging than they seem. Past production (about 345 kt of metal) and known reserves (430 kt of metal content in ore with an average grade of 1 %) are relatively large. Many occurrences also exist and in the recent past (1955) there were at least 5 mines producing copper in West Germany. We will briefly describe below the principal mines and districts beginning with the Variscan massifs and closing with deposits in the Permo-Triassic cover.

1.2.5.1 Oberharz

There continues to be considerable mining and prospecting activity in the Harz and particularly in the West German part. The principal metals produced are Pb, Zn and Ag especially in the old Lautenthal and Claustal-Zellerfeld mines and in the present-day Grund and Rammelsberg mines. Only the latter mine produces copper.

The Rammelsberg mine is located near the faulted border of the Harz. The mineralized bodies are in the

"Wissenbacher Schiefer" which is a series of 600 m of Middle Devonian gray and black shale with sandy and carbonate beds and many tuffaceous levels. The latter are especially important as they indicate volcanic activity which is not very evident at the level of the deposits but is more marked in the diabases of the upper part. They make it possible to link Rammelsberg to the massive sulphide type and the mine has many of the characteristics of these deposits. The mineralized bodies are concordant with the schist and are composed of very finely crystalline and mixed massive sulphides. The paragenesis includes mainly Zn, Pb, Cu, Ag and Au but also many other elements (Sb, As, Cd, Co, Ni, Sn...) some of which are recovered.

The deposit and its country rock have been intensely folded by the Variscan orogeny. The mineralized bodies are now in a syncline which is compressed, faulted and overturned to the NW. There are three principal orebodies, the "Old Ore Body", the "New Ore Body" and the "Grey Ore Body". The first two are large folded lenses with a maximum diameter of 700 m and with a maximum thickness of 30 to 50 m in the center. They are composed of two types of ore. There is the massive "Rich Ore" with an average grade of 18 % Zn, 8.5 % Pb, 1.3 % Cu, 160 g/t Ag and 1.2 g/t Au with 23 % Ba SO_4. There is also the "Banded Ore" with 8 % Zn, 4 % Pb, 0.5 % Cu, 60 g/t Ag and 0.5 g/t Au with 5 % $BaSO_4$. The "Grey Ore Body", located between the two preceding ore bodies is smaller and consists of a poorer ore with much barite (up to 85 %).

Another type of ore exists which originally was on the periphery and stratigraphically under the others. This is the mineralized "Kniest" which is a type of stockwork rich in chalcopyrite (3 % Zn, 1.4 % Pb, 1.3 % Cu) with little barite and no silver or gold. It has been worked only locally.

The total metal stock is around 7 Mt in 30 Mt of ore. Apart from the Kniest the present reserves are about 2.5 Mt and will be exhausted around 1985 at the present rate of exploitation, which is 280 kt/y.

Copper occurrences of another type are known in the Harz. Geochemical and geophysical prospection and drilling have indicated lenses of Ni-Cu sulphides disseminated in ultrabasic rock in the Harzburg area. The grades are quite high (reaching 3 % Ni + Cu) but prospection was abandoned because the tonnages are too small (600 kt for the largest lense). Mention is also made of the Fe, Ba, Cu vein of Grosser Knollen (Mittelharz) which is in Carboniferous country rock.

1.2.5.2 The Rhine schist Massif ("Rheinisches Schiefergebirge")

This large region between the meridian of Frankfurt, the Ruhr and the frontier consists of a Palaeozoic series (Ordovician to Dinantian but mainly Devonian) folded by the Variscan orogeny in a NE-SW direction. It is also affected by numerous longitudinal and transverse faults and contains many mineralized districts in which copper is of secondary importance. Production was last recorded for 1958 (449 t).

In general terms, it is possible to distinguish the following districts beginning with the right bank of the Rhine :

- The central district of siderite veins of Sieger and Wied : Many siderite veins have been worked in this region. Originally and sometimes even recently, mining was directed towards non ferrous base metals but these are now considered to be undesirable factors in the processing of the siderite.

These veins are commonly thick and generally show quite clear vertical zonation with an upper part containing Pb-Zn-Cu and a lower part with Cu (+ Ni, Co). Some have produced several Mt of siderite but for copper the most important seem to have been :

- Lohmannsfeld near Altenseebach and Stahlberg near

Musen in the northern district which since 1850 has produced 150 kt of copper ore with a grade of more than 5 % Cu ;

- Kolbach and Steimelhagen ;

- San Fernando-Wolf, Betzdorf and Anxbach in the southern district which was worked in the 20th century until 1965 and produced 60 kt of rich copper ore ;

- Georg and Sielberwise in the Wied district.

Around this central siderite district there are many quartz veins without a significant siderite content. They are generally of small size but are sometimes rich in non ferrous metals.

- In the 19th and 20th centuries the Ems district to the S produced 19 Mt of ore with a 10 % metal grade. The 1.9 Mt of metal content is as follows : 1,160 kt of Zn, 640 kt of Pb, 96 kt of Cu and 664 t of Ag. At least half of this is from the Ems district itself and this district should not yet be exhausted.

In the Taunus on the other hand production has been lower because the Pb-Zn-Cu veins are small and become unproductive at depth. Silbersegen and Alter Kaiser which were closed in 1920 and 1935 respectively were the most important.

- The basins of the Dill and the Lhann to the SE are well known for their iron deposits. The Devonian and Culm phtanites, sandstones and diabases also contain many Cu-Pb-Zn-Ag veins which were worked at various times. Total production probably did not exceed 10 kt of copper. The most important seem to have been Mehlbach, characterised by supergene enrichment of gray copper rich in silver, Boxbach near Nanzenbach and Hohensolms which were principally cupriferous deposits. All activity has ended in this sector but 360 mining concessions still exist.

- To the N, Sauerland has little copper. There is no Cu in the pyrite and barite deposit of Meggen and traces only are found in the complex ore of the Ramsbeck district. The exception is the Melusina mine near Meggen where studies have shown there are small reserves with a concentrate equivalent of 4.5 kt with 65 % Pb, 5 kt with 50 % Zn and 7.5 kt with 12 % Cu.

- In the Ruhr about 100 base metal occurrences are known in the coal beds. Small Pb-Zn mines have been worked and the veins are locally rich in copper. This is the case for the "Primus Sprunges" in the Pluto and Hannover mines near Bochum. As in the case of French, Belgian and Dutch coal basins, little attention has been given to the occurrences and a study on the European scale seems to be necessary.

- Lastly, along the Rhine valley several cupriferous occurrences are known near Wipperfurth, at Eitorf and especially at Muhlenbach (= St. Marienberg) to the East of Koblenz. The latter mine was worked from 1847 to 1960. From the 2.13 Mt of exploited ore, 117 kt of Zn, 38 kt of Pb, 2.9 kt of Cu and 47 t of Ag were extracted. Copper production is therefore very low but this mine was closed for economic reasons and new reserves probably exist in the northern extension of the vein.

- On the left bank of the Rhine copper deposits and occurrences in the Variscan formations are much less numerous. The only district with some importance is Hunsrück where several hundred concessions contain many veins. The last mine, Gute Hoffnung, in the St Goar district closed in 1961 after having produced 1.2 Mt of ore with 3 % Pb, 5 % Zn and 0.14 % Cu. On the whole, this district produced 135 kt Zn, 77 kt Pb, 5.25 kt Cu and 62 t Ag.

A large number of veins have therefore been worked in the Rhine schist massif but no mine produced a large amount and copper is almost always of secondary importance in relation to Pb-Zn. The Ems district seems to have been the richest but total copper production did not reach 150 kt.

1.2.5.3 The Basement to the East of the Rhine Rift

The graben of the Alsace plain extends up to north of Frankfurt. On the eastern border the Paleozoic and Precambrian basement appears in three massifs which from N to S are Spessart, Odenwald and the Black Forest (Schwarzwald).

- In the Spessart in addition to the kupferschiefer in the cover a small Co, Ni, Cu, Fe, Mn vein at Bieber can be mentioned.

- The Odenwald is also poor in copper. We can mention the two veins at Silberberg (Cu, Co, Ag) and Reichenbach (Cu, Pb, Ag, Ba) which were worked some time ago and were unsuccessfully reopened from 1935 to 1944.

- The Schwarzwald also contains quite a few occurrences but copper production has been negligible. Four groups of occurrences can be distinguished :

- Two small districts of vein occurrences containing Cu, Ba (Ag) with NNW trend to the North of Wolfbach and with a NNE trend to the East of Sulzburg ;

- The district of veins with complex paragenesis (Ag, Co, Cu, Ni, Pb, Zn, Sb, U) of the Wittichen and Wildschapbach area.

- The Ba-Cu veins linked to the faults on the SW border of the massif at Badenweiller and Schlachtenhaus ;

- Two Ni-Cu occurrences at Horbach and Todhoos.

In addition there is also between Freudenstadt and Badteiwach a district of NN veins in which the Neubulach veins is the most important. The occurrences are barite veins with some copper contained in Bundsandstein of the NE border.

1.2.5.4 The Basement at the Czechoslovakian Frontier

Three types of copper deposits are known there :

- Exhalative stratiform deposits in the Ordovician. The three most important are Kupferberg to the north (Frankenwald), Bayerland near Waldsassen (Fichtelgebirge) and Bodenmais (Bayernwald).

- The Kupferberg region between Frankenwald and Fichtelgebirge was once intensely worked with several shafts. Copper production ended in 1903 because the ore grade had fallen but later, and again recently, the extensions of the deposits were explored by many drill holes. Almost all the drillings crosscuted the mineralized level but the grade and thickness are still too limited for exploitation. The deposit has the shape of a synclinal basin and is enclosed in a metamorphic series of ancient basaltic lava and ferruginous limestone. It is partly located in the contact aureole of the Reisengebirge granite and in addition to the stratabound deposit there are cupriferous quartz veins.

- Bayerland was mined up to 1971 and produced 300 to 500 t Cu per year up to 1961 and afterwards about 100 t per year. The ore bodies are highly metamorphic but are conformable with the surrounding micaschist. One of these bodies consists of pyrite and the other one of pyrrhotite and both contain small quantities of chalcopyrite (0.55 % Cu), sphalerite, galena, and magnetite. They also contain Pb, As and Sb sulphides. The concentrate obtained was copper pyrite with a grade of 0.55-1.2 % Cu.

- Bodenmaïs is located further south in the Bavarian forest. It has been known and worked since the 14th century and was producing up to 1965 but no production figures are available. Its lenticular pyritic masses are exhausted and were located in Moldanobikum quartzites, schists and gneisses. The Fürstenzeche deposit has the same characteristics.

- There are also several small vein deposits including Ebendorf (Oberfranken). The Pb-Zn-Cu barite veins are enclosed in a granite and gneiss basement. They were worked from the 16th to the 19th century. To the North at Issigau there is also a pyrite, copper and fluorite vein in Paleozoic country rock.

There are Ni-Cu occurrences in the dunites of the Höher Bogen. These have been prospected but the greater part is located on the other side of the frontier in Czechoslovakia.

1.2.5.5 The "Rotliegendes" of the Nahe district (Pfalz)

In the valley of the Nahe, thick detrital and continental layers were deposited in the Lower Permian. Especially in the middle parts they are accompanied by basic volcanics (porphyrites and basalts). Many copper occurrences are known in the Middle Rotliegende which seem to be linked to basic lava. They occur in various form such as veins, cupriferous shear-zones and lenticular impregnation zones in agglomerates. The paragenesis with bornite, chalcocite, native copper and some chalcopyrite is poor in sulphur and contains in addition little Hg and Sb in some deposits.

The principal mines were Fischbach, Obermoschel-Lemberg, Kircheim-Bolangen and Donnersberg. The district had its best period before 1615 when the Thirty Years War ended operations. Studies at Fischbach (1934-37) showed that there were no reserves of importance.

To the East there are also stratiform sedimentary deposits at Gollheim (gray schist at the top of the Rotliegende with chalcocite concentrations) and Imsbach which were exploited in the 15th century but are exhausted.

1.2.5.6 The marine Lower Zechstein kupferschiefer

The kupferschiefer of Central Europe are one of the most classical and best studied examples of sedimentary stratiform copper deposits. The copper shales form a thin layer between the detrital Lower Permian (Rotliegende and Weissliegende) and the Zechstein limestone. The sequence is therefore transgressive and at that period the South limit of the Zechstein sea crossed all of Europe from the North of the UK (East of the Pennines) up to Poland. The boundary went through the Netherlands, the Ruhr, the East border of the Rhine schist massif as far as Heidelberg in the South and through the South of Thuringia. In West Germany this sea therefore took form of a large gulf within which are found all known deposits and occurrences.

The kupferschiefer are the principal form of the mineralization which can locally invade the footwall (conglomerate) or the hanging wall (limestone). These are sheet marls containing about 40 % carbonate and evaporite, 40 % silica (detrital quartz and silica gel), 10 % organic matter and 10 % miscellaneous including sulphides. The chemistry of the mineralization is very complex and contains Cu, Fe, Ag, Pb, Zn as the principal elements and Ni, Co, As, Mo, Bi, U, Pt, Pd, Re, Au, Vd as secondary elements. For the base metals the grades in the present day Sängerhausen mine are 2-3 % Cu, 0.1-1 % Zn and 0.1-1 % Pb. There is also horizontal zonation determined by the paleogeography (islands, shoals) at the time of the transgression and this chemical and mineralogical zonation is used in prospection for locating drillings. In general, there is also vertical zonation with copper predominating in the lower part while lead and zinc are more abundant in the upper horizons.

The principal productive zones are located in Eastern Europe.

The Mansfeld-Sängerhausen district of East Germany in the SE of the Harz contains about 3.5 Mt of metal. It has been exploited without interruption since the 12th century and still produces 20 kt/Y. Further East the Lublin district began to be worked in more recent times and makes it possible for Poland to produce about 250 kt per year. The characteristics of the Mansfeld deposit are very like those the West German occurrences which are a valuable potential source. There are however many mining problems because of limited grades and especially because of the narrowness of the mineralized horizon. At Sängerhausen the average thickness is 35-40 cm. Mining methods do not allow work to be carried on for less than 22 cm and the maximum thickness is locally 80 cm. There are two criteria which determine whether a sector can be worked. The cutoff grade is fixed at 0.55 % Cu and the metal content must be at least 10-12 kg/m^2. The mining of this sector would probably be abandoned if market conditions were the determining factor.

In West Germany the marine Lower Zechstein crops out widely and many deposits and occurrences are known. Along the ancient coastline on the East border of the Rhine schist massif, and from North to South the following can be cited :

- The small occurrences of Schafberg and Huggel near Osnabrück ;

- The occurrences and deposits to the West of Kassel : Stadtberge near Niedermarsberg, Thalliter near Korbach, Giershagen and Frankenberg ;

- The occurrences around Spessart including Bieber.

Around the Hercynian massifs, near the frontier with East Germany, the following can be mentioned from North to South :

- The South border of the Harz between Lauterberg and Steina (Hahausen) ;

- The region between Witzen Hausen and Eschwege to the SE of Kassel, to the S of the Werra Gebirge
- The region of Sontra and the Richelsdorfer Gebirge to the South (North and South Mulde).

This is not a complete list and from the exploration point of view the problem of kupferschiefer in West Germany should be studied on the regional scale since in most of the central German gulf the Lower Zechstein is at a depth of less than 500 m (at Sängerhausen the mine goes down to 600 m). For this surface it has been calculated that the resources are 12 Mt of copper in a mineralized horizon 20 cm thick with 1 % Cu. At the present time these resources are not exploitable but there could be districts with greater thickness and higher grades.

Considerable work and even mining has been carried on in several places. Stadtberge has some special characteristics for in addition to the copper shales, faulted zones have been exploited in the Zechstein and the underlying Carboniferous. Since the 12th century, 3 Mt of ore with 1.6 % Cu have been extracted on an intermittent basis and there are known reserves of 600 kt of ore with a content of 10 kt of Cu. Elsewhere there have been small mining operations for shorter periods. The largest was probably at Thalliter while the Frankenberg mine stopped operating in 1818 and unsuccessful attempts to mine again the deposit were made in 1870 and 1938. In the latter mine, the grades are low (0.8-1.2 % Cu and 11 g/t Ag) in spite of surface enrichment. Production amounted to 5 kt Cu and 8-10 t of Ag.

At Bieber, North of Spessart, the exceptionally large mineralized thickness can be locally as much as 2.5 m but the average grade is low i.e. 0.4 % Cu, 0.9 % Pb and 40 g/t Ag.

In the Richelsdorfer Gebirge two mineralized zones have been explored to the South of Sontra, called the "North basin" (14.7 km^2) and the "South basin" (36.3 km^2). The thickness is 20 to 40 cm and the metal content is 10 to 20 kg/m^2. The arenacerous ore i.e. the sandstone and conglomerate on the kupferschiefer wall, is the principal copper carrier. The two North and South deposits contain about 1 Mt of copper but reserves were calculated at 40 Mt with a 1 % Cu grade (+ traces of Pb, Zn, Ag) i.e. 400 kt of metal. Operations lasted only from 1939 to 1955 because they showed a deficit and could only be continued because of government grants. Mining was interrupted in 1955 by a catastrophic inflow of water. During this period 2 Mt of ore were exploited from which 15 kt of copper were extracted.

A minimum of 100 kt of copper were thus extracted by the small kupferschiefer exploitations in West Germany. There is a very large potential and the geological characteristics seem to be more varied than in Mansfeld. Exploration and mining methods and ore processing treatment studies should make it possible to transform part of this potential into economic reserves.

1.2.5.7 Triassic Deposits

These are principally showings. Production has therefore been very small but they are not uninteresting for the future. They are all located in the SW part of West Germany around the Rhine schist massif and the Black Forest.

We have already mentioned around the Black Forest the field of veins of Neubulach which is enclosed in Bundsandstein. At Eschbach to the NE of Villingen there is also a small stratiform copper mass at the top of the Lower Trias.

There are several deposits on the periphery of the Rhine schist massif :

- At Maubach, the mineralization is located in small veins and masses in the Devonian and in impregnations in the Triassic. The mine which had been worked for centuries was modernised in 1956 and in the 14 years period up to 1969 produced 10.4 Mt of ore with 1.93 % Pb, 0.80 % Zn and a little Ag and Cu. 166 kt of Pb, 39 kt of Zn, 850 t of Cu and 49 t of Ag were extracted. The mine is exhausted and the content is mainly Pb-Zn as in the very large low grade prospects at Mechernich.

- To the East of Stadtberge and Thalliter there are copper impregnations in the wall and roof (Culm and Bundsandstein) of the kupferschiefer. There were once small exploitations at Wrexen, Rhoden, and Twiste but drilling in 1940 and 1942 indicated grades which were too low ($<$ 1 % Cu).

- In Saarland as in Lorraine (France) irregular copper disseminations with low grades are located in the sandstone facies of the "intermediate beds" of the Lower Triassic between the "Main conglomerate" and the "Volzia sandstones". Copper was locally exploited at Wallerfangen and Duppenweiler since the time of the Romans. Recently a drilling South of Sarrebruck cut through a thin copper rich horizon (20 cm) at - 229 m. This shows the continuity of the productive level but the mineralizations known up to now are not economical.

1.2.5.8 Conclusions

In West Germany therefore many mines exist which have produced copper. However the majority are small veins where copper is a by-product of Pb-Zn-Ag or siderite. These veins contained in the Hercynian massifs have produced only several kt to several tens of kt of copper and are usually exhausted. There are no true porphyry copper deposits or large Ni-Cu deposits linked to ultrabasic rocks. The most important types of deposit for copper are :

- Sulphide deposits associated with vulcanism either in the Ordovician (E Bavaria) or in the Devonian (Rammelsberg in the Harz) ;

- The sedimentary stratiform deposits of the Permo-Triassic Epihercynian cover. The marine Lower Zechstein kupferschiefer are by far the most important but there are deposits in the Rotliegende (Nahe district associated with basic volcanism) and in the Bundsandstein (periphery of the Rhine schist massif).

Rammelsberg will soon be exhausted and there has been very active prospection for volcanosedimentary deposits. The known sedimentary stratiform deposits are thin beds with low grades. They nevertheless represent a very large potential, especially the kupferschiefer, and the possibility for West Germany of producing large quantities of copper ore depends upon solving the exploration, exploitation and processing problems presented by these deposits.

1.2.6 France (Inset fig. 6)

France is one of the smallest producers of copper ore. Its past production amounting to about 50 kt is negligible in relation to the world total, and even in relation to the EEC total its share is only 2.6 %. At the present time the only active mine is the Salsigne gold mine where copper is recovered as a by-product at the rate of 400-500 t per year.

There are however many old small mines and occurrences and the recent discovery of the new Brittany province containing massive sulphide deposits shows that future possibilities are far from negligible. The known resources already amount to 200 kt Cu and the potential is probably much larger. In spite of the low copper grades the ore in these deposits could probably be worked because of the Pb Zn Ag content.

As in the other EEC countries the deposits and occurrences (except for Alpine ophiolite) are located in Prehercynian massifs and their Permo-Triassic cover. The copper mineralizations are described below in accordance with these geological units which in France constitute well defined geographical entities because they are separated by Mesozoic and Tertiary basins.

1.2.6.1 Brittany

There are no known copper occurrences in the cover in which the Permian and Triassic strata are missing except to the N between Caen and Cherbourg. Six years ago the same thing could have been said about the Armorican massif since only a few insignificant occurrences were known. There have since been prospected with geochemical and geophysical (electrical and gravimetric) methods as well as drilling and three deposits were discovered while many anomalies are under study. It is now possible to speak of a Brittany province. This province is difficult to study geologically because there are insufficient outcrops and the deposits in Brittany are of a new type with complex tectonism. This study has been greatly facilitated by underground exploration work at Bodennec.

The recent discoveries are classified as being of the polymetallic massive sulphide type in the broad sense. The links with more or less strong volcanic activity are now well established. The stratigraphy of the Precambrian and Paleozoic formations of the Armorican massif and the tectonic phases through which they have passed are still not well known. This is largely due to the fact that there are very limited outcrops, that the Precambrian to Dinantian lithology is monotonous to a certain degree and that three principal orogenies exist i.e. Infracambrian, Caledonian for part of Brittany and Hercynian. The formations consist of alternating sandstones and shales with frequent but local volcanic episodes often with predominant spilite and quartz-keratophyre.

The mineralizations are concordant or subconcordant and more rarely in vein form and are located in several stratigraphic levels i.e. Brioverian (= Upper Precambrian) for Rouez and possibly Tremuson, Gedinnian for Bodennec, La Porte aux Moines and Saint Rivoal, Upper Devonian for Huelgoat and Poullaouen and Dinantian for some occurences. It is possible that there are copper occurences and deposits at other levels.

The three best known deposits are Rouez, Bodennec and La Porte aux Moines. Little information has been published on the Rouez deposit which was discovered by airborne geophysical prospection and is located in Briocerian acid volcanics and pelites. This is a true massive sulphide with a thickness of 20 to 60 m and it contains several tens of Mt of ore consisting of fine and massive pyrite and pyrrhotite with low and erratic grades for Cu (0.2-1 %), Zn (0.5-3 %), Pb and Ag. A figure of 25 Mt of reserves with a grade of 0.7-0.8 % Cu and 1 % Zn + Pb has been announced.

Bodennec and La Porte aux Moines are of Gedinnian age. Ash at Bodennec and breccia at Porte aux Moines are interstratified at the deposit's wall and show discontinuous volcanic manifestations from unknown centers of emission. The principal mineralization is polymetallic and consists of massive sulphides rooted in a low grade disseminated mineralization and veinlets in a chloritic and siliceous gangue. The two deposits have very different characteristics however. Porte aux Moines is apparently a massive sulphide crosscut by drillings on thicknesses of 15-50 m, enclosed in chloritic black shales in which only Hercynian trends appear. Bodennec is a thin bed with decimetric to metric thickness which is very rich (30 % metal). The tectonism is complex for there are at least two trends for folding one of which is probably Caledonian and was reworked by the Hercynian orogeny.

Quantitatively the volcanics are of little importance in these two deposits but since acid ash and tuffite are present in the wall and there are partly exhalative schist and quartzite in the roof they can be classified in volcanosedimentary deposits family.

Current estimates for reserves may very well be revised by the work now in progress. These figure are 1.5 Mt with 4.6 % Zn, 4.4 % Pb, 1.9 % Cu and 90 g/t Ag at Bodennec and 2.2 Mt with 8.96 % Zn, 2.12 % Pb, 0.7 % Cu and 100 g/t Ag at La Porte aux Moines. The copper content of these two deposits is therefore 40 to 45 kt but it can be expected that there will be extensions and at the provincial scale that there will be new discoveries of deposits of this type.

Most of the copper vein occurrences in Brittany have no economic importance. The copper-tin vein network at Lanmeur near Morlaix which has an apparent resemblance to the Cornwall deposits should be noted. Copper in association with tin is however exceptional in Brittany and here its presence is due to the fact that the granite is enclosed in gabbro. The vein field contains about 1 Mt with 0.5 % Cu and 0.5 % Sn and does not reach the economical level.

1.2.6.2 The Massif Central

There are many copper mineralizations of various types in the Massif Central but they have negligible economic importance although the only mine now active is located on its South border.

- Salsigne is principally a gold mine. At the present time 300 to 500 t of copper per year are recovered from a complex ore with an average grade of 10-12 g/t Au, 20-35 g/t Ag, 400-450 g/t Bi and 2-2.5 kg/t Cu. Since its beginnings, the exploitation over an extension of 700 m and a depth of up to 360 m has produced 6.5 Mt of ore from which 67 t of Au, 185 t of Ag, 1,330 t of Bi, 280 kt of As, 280 kt of H_2SO_4 and 18 kt of Cu were extracted. Estimations for gold reserves are 40 t and copper reserves are probably around 15 to 20 kt.

Geologically, the mineralizations are located in a volcanosedimentary sequence of the "X schist" (probably Lower Paleozoic) to the South of the Montagne Noire. The intense tectonism with overthrust and overturned series cut by many faults is still not well understood. The metallogenic characteristics of the mineralized bodies are therefore not well determined.

There are two groups of mineralized bodies i.e. the vein bodies located in NS fractures and stratiform bodies occuring around the former. The vein bodies were formed by the infilling of open fractures or by the cementation of fault breccia. The stratiform mineralizations are disseminated or massive but in this case they have a banded appearance owing to alternating of thin sulphide rich and quartz rich layers.

In these massive bodies there is on the average 40 % mispickel + pyrite + pyrrhotite and 60 % quartz. There are also small quantitie of Cu, Ag, Bi, Au in these complex sulpho-arseniate mineralizations whose paragenesis includes more than 25 mineral species.

Although the relations between the Infradevonian acid volcanism and the mineralized bodies have not yet been well established (volcanics are unknown in the mine) a new metallogenic interpretation of the gold, bismuth and arsenic district could be proposed. Thus the mineralization considered as Late Hercynian might in fact be contemporary with the volcanism. In the mining district,copper associated with siderite is more abundant in the E part (Rameles, lower levels of Fontaine de Sante).

In the Paleozoic in the S of the Massif Central (Le Vigan district, Monts de Lacaune and S edge from Salsigne to Bedarieux) there are many copper occurrences in veins or of other types which for the moment are of academic interest but whose context is favorable for the presence of Pb, Zn, Cu deposits.

- Deposits in the NE of the Massif Central :

Most of the deposits in this region are linked to the presence of Devono-Dinantian volcanism but this relation is more or less evident depending upon the intensity of later tectonic deformations and especially upon remobilisations of the mineralization by Hercynian intrusions.

- The Massive pyrite deposits of Sain Bel are intercalated in the Brevenne group which is a submarine volcanosedimentary group probably of Devono-Dinantian age. The massive sulphide lenses are located in white sericite schist or to a lesser extent in the underlying volcanics with predominant spilite and quartz-keratophyre. They are linked to acid vulcanism and they have an exhalitive-sedimentary origin. The massive ore is almost pure pyrite or pyrite with quartz cement with traces of chalcopyrite and blende. This mine has been known since Gallo-Roman times and produced only a small quantity of copper (several kt). Before its recent closint it produced about 20 Mt of pyrite. The figures for copper are very approximate since the mine is old and up to recently the pyrite residues were processed in Germany.

- The Chessy mine was exploited intermittently up to 1878. It is richer in copper (chalcopyrite in the sulphid deposit ; carbonate, cuprite and tenorite in the "blue mine") and production may have reached 10 kt.

- Chizeuil is another pyrite mass located in

Tournaisian quartzite penetrated by granite and pegmatite. In this mine which was long exploited for pyrite, a band of massive ore with enargite, pyrite, bornite, chalcopyrite and galena was found in 1958 on the roof of one of the masses. Copper production ended in 1961 and the total production was 1,038 tonnes.

- The other mineralizations were of mixed origins, firstly volcanosedimentary and then magmatic. This is the case for the sulphide impregnations of the Beaujolais where five sulphide masses of small size were remobilised on the SW edge of the metamorphic aureole of the Odenas granite. To the N this district also has intragranitic veins with Cu, Pb, Zn, Ba, F. (Montchonay, les Valettes). Charrier is a copper tin and iron (magnetite) deposit with high grades enclosed in a Devono-Dinantian volcano-sedimentary series : schists and quartzites with volcanic base (basalt, andesite, spilite). This series is traversed by a Visean granite accompanied by granophyre and microgranite cutting through the series. The hydrothermal phenomena linked to the intrusion of the granophyre must have remobilised the copper of the basic lava and formed the present masses and veins with complex paragenesis. From the end of the 19th century to the present this mine produced about 2 kt of copper concentrates with 20-30 % Cu and 800 t of cassiterite.

Around Charrier and further South, on the East border of Limagne there are many copper veins which probably indicate possibilities in this region. The other vein showings of the Massif Central accompany other mineralizations i.e. gold at St Yrieix, fluorite in the coal belt and Pb Zn in the South, but the copper has no economic importance.

- The Southern Permian Basins

The Permian basins of Brive and the W border of the Causses (Rodez, Saint Affrique and Lodeve from N to S contain stratiform occurrences and have been actively prospected without results up to the present time for copper since the Lodève basin contains large uranium deposits. These occurrences are impregnations of sandstones, coarse conglomerates or fine, gray or green, schisty sandstones levels in red detrital series. In the Brive basin the oxydized copper is in shear zones located in the Lower Stephanian and Saxonian sandstones and was mined to a slight extent around 1870 and 1910. In the Lodève basin the occurrences are stratiform and at the Permo-Triassic contact but they have never been worked. The mineralized outcrops are extensive (up to 9 km) while the thickness is 0.4 to 0.9 m and the grade is 1 to 1.7 % Cu (0.4-0.9 % for a grade based on 1 m). In the Saint Affrique basin the cupriferous lenses intercalated in the Saxonian schists and sandstones are discontinuous with a thickness of about 1 meter and with a low grade (0.3-0.8 % Cu). The Raminier vein is included in this series and there was a recent unsatisfactory attempt to exploit it. These Permian cupriferous red beds in the SW of the Massif Central have long remained at the occurrence stage and will probably continue to do so for a long time.

1.2.6.3 The Pyrenees

The Pyrenees as a whole and especially the French part are poor in copper. The Ordovician and Devonian volcanosedimentary series contain Mn and Pb-Zn occurrences and deposits but no copper. The old exploitations involved small veins.

- Banca Baigorry is located in the Basque Country, in Ordovician schists and quartzites. The vein has a siderite gangue with chalcopyrite, boulangerite, pyrite, galena and sphalerite. In the 19th century it produced 1.2 to 1.5 kt of copper with 10 to 12 kg/t of silver. A study made in 1960 showed possible reserves of 120 kt of run-of-mine containing 3.5 to 4 kt Cu and 25-30 t Ag. There are many occurrences and other small deposits in the French and Spanish Basque Country.

- Alzen is located in the Ariège. There are small veins and breccia masses in the Devonian limestone related to microdiorite dikes. The paragenesis is identical to that at Banca Baigorry. At the end of the 19th century and the beginning of the 20th this small mine produced 340 t of copper and 20 t of silver principally from the cementation zone with rich ore. There are no known reserves.

- Padern is another small vein located on the South border of the Massif of Mouthoumet. This quartz vein containing chalcopyrite, galena and boulangerite is enclosed in sandstones and production from 1910 to 1920 was negligible.

1.2.6.4 The Vosges

As in the Black Forest, the Vosges contain many copper occurrences of various types. These can be divided into several districts but their economic value is slight.

The only one of any importance is the vein field of Sainte Marie aux Mines and Sainte Croix aux Mines which was exploited for a very long time up to the 19th century. Part of it produced mainly Pb Ag and the other part produced Cu and Ag. The total Ag production was estimated at 240 t and the copper production probably amounted several kt. These veins are enclosed in gneisses and have a complex paragenesis with chalcopyrite, tetrahedrite, ruby silver, native silver, skutterudite and nickeline. There are many small veins of the same type to the N especially at Lubine, Urbeis and Triembach.

A small district with a quartz-vein containing Cu Bi exists around the Schlucht pass in the center of the Vosges massif. Further South there are Cu Bi veins once again at Büssang near the Urbes-Mollau district consisting of 25 cupriferous veins enclosed in Visean schists, graywackes and tuffs. The most important mines which were exploited in the 16th and then in the 18th centuries were Mollau-Wesserling, Storkensohn, and Urbes but their production is unknown. Copper was also recovered from the copper-lead-silver veins in the Southern part of the neighbouring district of Giromagny-Auxelles.

There are other kinds of copper mineralizations amongst which can be mentioned the Cu Mo stockworks at Chateau Lambert and Saint Maurice on the North edge of the Ballon d'Alsace granite, and the occurrences in the Permian linked to shear-zones at Nayemont but of stratiform sedimentary type at Anozel and Courmont.

1.2.6.5 The Saint Avold district is located near the German frontier and the mineralizations found there have the same characteristics as Saarland occurrences of Wallerfangen and Duppenweiler. These Pb and Cu deposits were worked long ago and studies,made in 1957-58 and at the present time,make it possible to determine their context. These stratiform mineralizations are located in Middle Triassic sandstone facies and in threshold zones formed by the substratum. The copper deposits are small, lenticular and irregular and have low grade and present day prospection is directed towards lead. There are about fifteen known copper deposits the most important of which are Hautbois, Steinberg, Hellering and Falck. These deposits are much smaller than the neighbouring lead deposits of Bleiberg and Castelberg.

1.2.6.6 Maures and Esterel and their cover

The Maures and Esterel basement has no copper but there are some occurrences and a stratiform deposit in the Permo-Triassic cover. Cap Garonne is located near the coast in the Var. This is a stratiform deposit in conglomerate in the Triassic base overlaying Upper Permian red sandstone. The mineralization contains tennantite, bornite, chalcopyrite, sphalerite, galena and several altered minerals. This small deposit was discovered in 1862 and was worked from 1862 to 1867 and then again between 1880 and 1914. It produced only 1.9 kt of copper from 65 kt of ore with a grade of 2.92 % Cu with traces of Pb and Ag in an ore body with an extension of 200 x 200 m.

1.2.6.7 The Alps and Corsica

The copper deposits can be roughly divided as follows :

- The Permo-Triassic of the "Dôme de Barrot" and the South Border of the Mercantour contain several stratiform copper deposits. These disseminations are located in a schist-sandstone layer of the Upper Permian and in a coarse conglomerate of the Triassic base (Werfenian). The mineralization is in the form of veinlets, nodules and specks and contains principally carbonate ores with a little chalcocite and covellite.

The largest mine is Le Cerisier which from 1864 to 1884 and then very sporadically at the beginning of the 20th century produced around 4 kt of copper extracted from 200 kt of ore with a grade of 2 % Cu. There are also reserves which have not been estimated and which are too small for a reopening of the mine to be considered.

More than 10 stratiform occurrences with copper alone or with associated uranium and several vein occurrences are known in this Permo-Triassic stratum but they do not seem to be more interesting than the occurrences in the Permian basins of the Massif Central.

- Veins in Basement. These are rare in the Mercantour which contains only 2 or 3 small vein occurrences to the NW but they are more abundant in the northern Alps. Mont Blanc and the Aiguilles Rouges contain many quartz veins with chalcopyrite and tetrahedrite but none are of economic importance. The case is different however in the Belledonne where in the Aiguebelle district many veins have been worked including the large vein at Saint Georges d'Hurtières (Les Fosses) from which 1.5 Mt of siderite and 400 t of copper were extracted. Copper is therefore a by-product and is irregularly distributed in the siderite vein. The other BPGC veins of Belledonne, the Grandes Rousses and Pelvoux are without economic interest as concerns copper. The stratiform deposits located in the Triassic cover of these Alpine massifs to the N (La Plagne, l'Argentière La Bessée) are Pb Ag (Zn) deposits without copper and are considered exhausted.

- The deposits of Alpine Ophiolites. As in the Italian Alps greenstones are found in the monotonous series of "schistes lustrés" and were deposited at the Lower Cretaceous-Upper Jurassic age. They crop out in the neighbourhood of marble and radiolarite and represent an ophiolitic series consisting, from bottom to top, of spilite, serpentinized ultrabasic masses, gabbro and diorite and then basic pillow lava.

Although they are less numerous than in Italy there are copper occurrences in this series and the Saint Veran mine produced intermittently between 1902 and 1959 around 1 kt of copper. The mineralization which is rich in bornite and native copper is stratiform and located in quartzite with riebeckite at the "schistes lustrés"-serpentine contact.

The "schistes lustrés" zone continues in Corsica in the N-E part where many occurrences of this type are known. Several mines produced small amounts at the beginning of the century i.e. Tama in the commune of Vezzani (600 t of Cu), Saint Augustin (= Piana) in the commune of Castifao (30-40 t) and Ponte Leccia. In these small mines and occurrences are found the same types (masses in serpentine, veins in gabbro, pyrite masses in prasinites) as in Italy but their economic value seems to be even smaller.

In Corsica there are also occurrences and small deposits in the basement to the NW the largest of which are the Cu As vein of Lozari and the Cu Pb Zn Mo mass of Prunelli.

This brief survey of the copper occurrences and small deposits in France shows clearly that this country cannot be an important copper producer. It should not however be concluded that the chances for new discoveries are so small that prospection is not justified. There has been significant renewal of prospection for metal mines only in the last 20 years and it is only in the last five years that this has been directed specifically towards copper. The discovery of the new deposits in Brittany was encouraging and this favored prospection for volcanosedimentary deposits in the Hercynian massifs. The possibilities of Alpine ophiolite and the Permo-Triassic cover should not however be forgotten.

1.2.7 Italy (Inset fig. 7)

The Precambrian is almost unknown in Italy and the principal structural characteristics date from the Alpine orogeny. The geological history is however very complicated and the contexts containing copper occurrences or mines are very varied. We will arrange them below in terms of geological and geographical criteria.

1.2.7.1 The Ivrea Verbano Zone (Dioritic-kinzigitic zone)

In the Alpine context this constitutes a very special unit with a complex history. It was highly metamorphosed during the Caledonian orogeny and was reworked by the Variscan and Alpine tectonics. It now forms a NNE-SSW band 100 x 10-20 km which is made up roughly as follows from W to E :

- A large differentiated ultrabasic complex (peridotites, pyroxenites, norites, gabbros) with small Ni-Cu (Co,Pt) sulphide deposits ;

- A metamorphic series of paragneisses (kinzigites and stronalites) of marbles, quartzites and amphibolites enclosing pyrite lenses with Cu and Zn, small deposits of manganese (quartzites) and barite (marbles).

- The small Ni-Cu deposits are conformable lenses of massive ore in the center, disseminated minerals on the periphery and with classical paragenesis (pyrrhotite, pentlandite, chalcopyrite, etc.) with a Ni/Cu ratio of 3 to 4. They are enclosed in ultrabasic rocks and in the transition zone to metagabbros. About ten deposits and prospects are known,the most important of which are the following : Campello Monti , La Balma, Monte Capio, Alpe Laghetto, Balmuccia and Fei di Doccio. The grades are 0.5-1.5 % Ni and 0.1-0.7 % Cu sometimes with 1 % Co and the tonnages mined have remained low (about 50 kt grading 0.8 % Ni + 0.1-0.4 % Cu).

- The Fe-Cu (Zn) deposits are located further to the East near Lago Maggiore in gneisses with amphibolite intercalations but they are also conformable lenses of massive ore and disseminated ores with pyrrhotite, chalcopyrite, sphalerite and traces of molybdenite. The best known of the old mines are Migiandone, Nibbio and Alpe Collo, dimensions of which are very limited. Grades were no doubt high, but past production was low (1-3 kt Cu).

A more detailed geological study appears to be necessary in order to limit and orient prospection but deposits will be hard to work because of the rugged relief and it does not seem probable that a target can be found with more than few millions tonnes with 2 % Cu which is a small objective.

1.2.7.2 Deposits in the Eastern Alps

In the Eastern Alps various types of copper occurences and small deposits are known. Predoi, the most important of them is linked to ophiolite and is dealt with in the next paragraph while the following types are distinguished :

- Paleozoic metamorphic deposits which are stratiform polymetallic deposits linked to a particular horizon of the south-alpine basement of unknown age. Calcarenica, Vetriolo and Valle Imperina were worked a long time ago by the Republic of Venice and there are many occurences of this type ("Kieslager") in the basement to the NE of Bolzano. Past production was about 20-30 kt of Cu. The largest ore bodies are about 1 to 3 Mt of pyrite with Cu, Pb, Zn (1-1.5 % Cu).

- Deposits linked to the Devono-Carboniferous paleosurface. In the Paleocarnican Alps, the Carboniferous is transgressive over a Devonian carbonate mass which had emerged a long time before. The summit of these carbonates contains mineralized beds and impregnations and karstic pockets with tetrahedrite, sphalerite,galena and barite. The only deposit of this type which has been worked is Monte Avanza.

- Deposits linked to Variscan magmatism (Bressanone crystalline rock). There are Pb-Zn-Cu veins with occasional trace elements in a quartz-barite and fluorite gangue which may predominate (Caoria, Torgola). Quartz veins bearing Cu-Mo are known at Montefondoli in a dioritic intrusion. In spite of the presence of a potassic halo it is not a porphyry copper. Recent prospection has shown extensions (40-50 kt Cu) in the SW of the old mine difficult to exploit for ecological reasons.

- Deposits linked to Triassic magmatism : the Mount Mulat deposit is a stockwork with scheelite and chalcopyrite (with a little pyrite and stibnite) linked to a porphyritic vein crosscutting a Triassic monzonite. It was worked in the past, then opened again in 1947 and since has been sporadically prospected. The proved reserves are 80 kt with 0.8 % Cu and 0.25 % W but the potential exceeds 1 Mt.

1.2.7.3 Mesozoic Ophiolites

These appear especially in the Alpine arc to the

West of the meridian of Genova but they also extend into the Eastern Alps (Hohe Tauern, Vale Malenco), in the Appennines of Liguria, Emilia and Tuscany and even much further south, in northern Calabria.

The stratigraphic sequence is the same as that of all ophiolitic complexes with roughly the following from bottom to top :

- An ultrabasic (serpentine) and basic (gabbro) intrusive group ;

- A group of basic volcanics (diabase masses and dikes with associated breccia and submarine spilite with pillow-lava at the top) ;

- A sedimentary series beginning with an often manganesiferous jasper or chert level overlaid by limestones and carbonated schists. The cupriferous mineralizations are of two types :

- Simple occurrences in the base complex. Small low grade disseminations of pyrrhotite and chalcopyrite in ultrabasic rocks (Mt Ramazzo in Liguria) or pyrite-bornite-chalcopyrite veinlets and impregnations in tectonised gabbros (Bargone and many showings in Liguria, Vallone delle Miniere in the Alps) ;

- Deposits in basic volcanics where the old mines are located which worked relatively stratiform pyrite and chalcopyrite lenses. Several sub-types are distinguished depending upon the paragenesis and context. In particular, there are differences between the Alpine and Appennine domains as concerns the metamorphism and tectonics characterising the mineralizations.

In the Alps there are stratiform pyrite-chalcopyrite lenses (with traces of Zn, Au, Mo, Pb) in the greenstones of Alpine "schistes lustrés" ("calcescisti con pietre verdi")
Metamorphism of the greenschist facies has little trans-

formed the deposits in which the massive sulphide volcano-sedimentary type can be clearly recognised. There are many occurrences of this type and the deposits contain from 20 kt to 1 Mt of ore. The best known are Alagna Valsesia (=Fabbriche), Preslong-Ollomont, Herin-Chamdepraz, Chialamberto (= Fragne) and Beth-Ghinivert in the Western Alps and San Valentino di Predoi in the Eastern Alps.

Alagna Valsesia. The mineralization is very conformable and appears as long slabs of massive ore several centimeters to 1 m thick located in a 6 m enclosing horizon. Tectonics and metamorphism later led to fragmentation and formation of boudinage structure in the mineralized body. The known deposit potential (production + reserves) is 1.5 Mt with 1-1.5 % Cu, but will be probably increased by the actual workings.

At Chialamberto the ore is enclosed in a talc-chlorite schist horizon located at the upper and lower contacts of a stratose prasinite body. There are thus two mineralized bodies the largest of which (the lower one) has the following dimensions : 700 x 200 x 0.5-4.5m. In addition to the usual minerals the paragenesis contains enargite and tetrahedrite but no data is available concerning tonnages and grades of extracted ore and reserves.

San Valentino di Predoi on the upper Adige, is located in chlorite-schist lenses enclosed in calc-schist. The mineralized bodies are also lenticular and more or less regular and extended. Here again there are no figures available as to production while proved reserves amount to 50 kt with 2 % Cu and there are 200 kt of probable and possible reserves.

Metallogenic activity associated with Alpine ophiolite is therefore important and this type of deposit may be of future interest. Estimates of this type of deposit are 50 kt Cu for Past production and 30 kt Cu for reserves. However the stratigraphy and mapping of the "calcschisti" should be carried further so as to keep prospection within the host-rock formations. At the present time prospection would be at the strategic stage with geochemical and geological studies being carried on over large surfaces.

<u>In the Appennines</u> the mineralized bodies are located in the upper part of the spilite with pillow-lavas or in diabase olistoliths, which are transformed to varying degrees, and in locations which are stratigraphically unknown. These are stratose lenses of massive sulphide sometimes associated with a stockwork and disseminated ore. Paragenesis is often simpler with pyrite, chalcopyrite (1 to 2 % Cu) and sphalerite (3-4 % Zn locally) sometimes with a little gold. This is the case at Vigonzano in Emilia, for all East Liguria, where a well known enclosing horizon contains many occurrences, and in other sectors. These mineralizations become of some economic interest if special conditions concerning paleogeography (depressions) and supergene alteration occur as at Libiola (Liguria) and Montecatini Val di Cecina (Tuscany).

<u>At Libiola</u> the mineralization is in a roughly cylindrical, very fractured zone of pillow lava (85 x 60 m) which is known to a depth of 80 m. Production amounted to 15 kt of metal from a run-of-mine ore with a content of 1.5 % Cu. The primary paragenesis contained bornite and a cemented zone was enriched in chalcocite.

<u>At Montecatini Val di Cecina</u>, the cementation phenomena are even clearer. The mine produced 50 kt of metal during 75 years of operations from an ore with a 7 % Cu content. It is not possible to describe briefly the tectonic history of this very complex sector. The deposit was discovered as a result of the fact that there was an outcrop of a lode several meters thick. The paragenesis predominantly consists of cupriferous sulphides with a little pyrite, sphalerite and

galena. However the largest and richest part was concealed and consisted of a mass of chloritized and argillaceous plastic rock located at the contact with spilite. This mass 700 m long and 20 to 50 m thick (maximum 100 m) contained globular pockets of massive copper sulphides ranging from several cm to several dm with a concentric structure (chalcopyrite core with bornite and chalcocite halos). The role of the argillaceous wall as a mineralization trap has been stressed.

The very complex geological context of this type of deposit makes prospection for hidden deposits very expensive but many occurrences are known.

1.2.7.4 Southern Tuscany

The geological background of the deposits has been much complicated by tectonic activity (folding and thrusting due to Alpine movements, recent faults) and by Miocene-Quaternary volcano-magmatic activity.

The latter has played an important role in the pyritic orebodies genesis. Many deposits are masses located at the contact with quartz monzonite intrusions with associated skarn facies. One even suggests that the base-metal mineralization is contemporaneous of the magmatic activity which remobilized S and Fe in the intruded series.

But published sections of the deposits very often show (Niccioletta, Boccheggiano, Monte-Argenterio and even parts of Fenice Cappane) stratabound lenses at, or near, the contact between the Triassic "evaporite formation" and cavernous Norian-Rhaetian limestone. In more precise terms, the mineralized lenses are either at the phyllite-cavernous contact or within the "evaporite formation" at the phyllite-evaporite contact. This suggests a possible sedimentary metallotect.

In addition, deposits of this type in Tuscany were known principally as massive pyritic ones. With the new developments at Fenice Cappane and Campiano, the mixed sulphide mineralizations (Cu, Pb, Zn, Fe) become more important. There are mined at Campiglia Marittima and Fenice Cappane and a drill hole at Campiano has crosscut, below the pyritic orebody, 40 m grading 12 % base metals (2 % Cu) and 200 g/t Ag. These new facts do not fit very well with the genetic explanation scheme.

It is thus suggested that the importance of faulting and intrusions has been exagerated, if not for the present form of the deposits, at least for the genesis of mineralization. There could be some metallotects linked to the pre-tertiary geological history of the area and if it is confirmed, it could open intersesting perspectives for prospection in Tuscany, where past production is 20-30 kt Cu and where known reserves are about 70 kt Cu.

- Campiglia Marittima is a skarn linked to an intrusion in Triassic limestone of a porphyry vein at a granite apex. The mineralization consists of sulphide impregnation in the skarn : disseminated chalcopyrite, galena and sphalerite (+ Magnetite) in the form of spots, nodules, veinlets in irregular zones. There is zonation with copper predominant in the center (1.2 %) and Pb-Zn on the periphery (4 % Zn + 2 % Pb). Pyrite, pyrrhotine, mispickel and tetrahedrite are also present. Production before World War II is not known but since the war 1 Mt of ore has been mined from which 30 kt of metal were extracted with a content of 7,5 kt Cu. Proved reserves amount to 250 kt with 0.8-1 % Cu to which can be added 500 kt of possible reserves.

- Fenice Cappane. The deposit which is now being worked is a large mineralized quartz vein which was deposited in a very faulted zone. It is approximately 1,500 m in length and it has an average width of 10 m (1-20 m). It is at the contact between Triassic limestones and phyllites and liguride flysch in which it has formed a small cementation zone. Below, sulphide grades increase with depth and horizontal zonation appears between the Zn rich southern part and the Cu rich central and northern parts.

.This lode mineralization is interpreted as a remobilization due to heat of intrusions of a Triassic concentration.

The present average grade is 5.20 % Zn, 1.1 % Pb and 0.29 % Cu (pyrite and iron + Ag and Cd in traces). Total production is unknown. Run-of-mine production in 1976 was 130 kt containing 9.75 kt of sphalerite, 1,625 t of galena, 825 t of mixed sulphides with 10 % Cu and 3.9 kt of pyrite. Present mine reserves are 900 kt of mixed sulphides predominantly Pb-Zn but the new mine now being prepared for working the lower part of the deposit contains 6,345 kt of cupriferous ore and 1,800 kt of mixed sulphides (75 % are proved reserves and 25 % probable reserves).

1.2.7.5 Sardinia

Sardinia is known principally for its Pb-Zn and Ba mines but it also has several types of cupriferous deposits.

The Sulcis-Iglesiente district in the SW contains large Pb-Zn and Ba deposits enclosed in Cambrian carbonate of the "Metalliferous" : Monteponi, San Giovanni, Campo Pisano, Masua, etc. Some of these ores have undergone several phases of genetic evolution. There is little associated copper but it is sometimes recovered as a by-product (for example around 100 t/y of recovered copper contained in a mixed Pb-Zn-Cu concentrate at Rosas).

The ores of Sulcis (Rosas-Sa Marchesa, Truba Niedda, etc.) often contain a little copper (0.3-1 %) in ore with 4-10 % Pb + Zn. Copper is more rare in the mines of Iglesiente but a cemented zone with copper oxide was worked at Sa Duchessa and deep veins at Montevecchio seem to be enriched with copper (towards - 250 m). Lastly, in the North, Perda S'Oliu (district of Fulminese) is a copper-pyrite deposit which seems to be in the Siluro-Devonian and should therefore be compared with the SE sector.

The SE quarter of Sardinia (Barbagia-Ogliastra) is made up largely of series of Ordovician-Devonian age. The Silurian which overlies sandstone quartzites and basic volcanics, is made up of graphitic black schists with lenticular levels of black limestones containing

Orthoceras in the upper part. In this horizon, where appear the first carbonates, are located many small conformable lenses (2 to 50 kt) of ore which is rich but fine and hard to process. The deposits of Tertenia Talana and Torpe further to the North are located in this context as well as the Funtana Raminosa mine.

This is the only active mine in the sector. The Silurian series is unconformably overlaid by Sericitized phyllites with quartzite banks and has been affected by three orogenic cycles. In addition to Caledonian metamorphism and Alpine regional faults, Variscan magmatic activity with granite and porphyritic vein intrusions has, more locally, led to contact metamorphism and intense deformations of the series.

The mineralization is distributed over a thickness of 100 m in small lenticular and irregular bodies in average 3 m thick and 10 m long. Type of deposit and genesis are still hypothetical. Average ore grades are 1 % Cu, 1.4 % Pb and 3 % Zn. There are no figures for early production (about 10 kt Cu) but from 1958 to 1970, 450 kt of ore were produced. Annual production is now 47 kt containing 350 t Cu, 500 t Pb, 1,100 t Zn and 1,900 kg Ag and is increasing. Proved reserves are 243 kt and probable reserves are 924 kt.

Mineralisations linked to magmatism in the NW. Disseminated mineralizations are known to exist in trachyandesite tuff beds of Alpine age. They have been worked with handicraft methods at Bosano and other occurrences are known but the largest deposits are in the Calabona area.

In the past, there were sporadic small-scale workings of high grade oxidized copper at Vessus and Salondra but recent prospection has indicated a mineralization of the porphyry copper type. Under thick Oligocene trachyandesite cover, varied sediments appear with ages ranging from

Silurian to Liassic which have been traversed by a porphyritic intrusion whose age is supposed to be Upper Cretaceous. Drillings showed that this porphyry contained low grade disseminated copper (0.2-0.3 % Cu) and hydrothermal zonation characteristics of porphyry copper. The mineral association is quite complex and contains gold while the old workings were contact deposits with Cu and Zn (Argentiera).

This deposit cannot be worked for economic and environmental reasons but its existence makes the other occurrences and anomalies in the NW and NE of Sardinia of interest.

Disseminated copper mineralizations linked to intrusions are also known in Calabria and have to be prospected to define their economic potential.

1.2.8 Overseas possessions of the EEC

Apart from Greenland with its large surface and which has already been studied with Denmark, the overseas possessions of the EEC countries are numerous but their area is small. They consist mainly of islands or coastal enclaves for the most part with no importance for copper. A few, however, may have some future possibilities in this field, especially the Solomon Is. and New Caledonia. These territories will be briefly reviewed below by geographical zone.

1.2.8.1 Possessions in the Atlantic

- The Antilles : some small islands of the Antilles belong to France (St. Martin, St. Barthelemy, Guadeloupe, Martinique), to Great Britain (St. Thomas, Anegada, Virgin Islands, Sombrero, Anguilla, Antigua, Montserrat, Nevis, Barbuda, Dominica, St. Vincent and St. Georges) and to the Netherlands (Bonaire, Curaçao, Aruba). The geological context of the island arc is favorable and there are many copper deposits associated with volcanism and Laramian intrusions in Haiti, the Dominican Republic and Puerto Rico.

In the Lesser Antilles, the most promising area for copper is the outer arc. But this geologically favorable zone is known only in the North (Leeward Is.) where copper occurences are known to exist at St. Martin, St. Barthelemy and Antigua. There does not seem to have been any exploitation, however, and the emergent part is so small that there are almost no chances of finding copper.

- Islands along the edge of North America : the Bahamas and Bermuda belong to the United Kingdom but are of no importance from the point of view of copper. St. Pierre and Miquelon Is. (France) are located to the South of Newfoundland and are part of the eastern extension of the Appalachian chain. About ten copper occurrences are known to exist there but the Cu-Zn (Pb) veinlets associated with volcanic series and gabbros or situated in Langlade

Cambrian sediments do not seem to have any economic importance.

- French Guyana : since Surinam (previously Dutch) and Guyana (previously British) have become independent, French Guyana is the only territory dependent upon the EEC in Latin America. Guyana covers a large enough area (91,000 km^2) for it to be possible to expect discoveries of minerals but the Guyana shield in general has, up to the present time, shown itself very poor in base metals. There are, however, several known copper occurences in Surinam and the volcanosedimentary context of Paramaca series (Middle Precambrian) do not seem to be unfavorable. The territory is now being systematically prospected and Pb-Zn and Cu anomalies have been discovered in this unit.

- The British Islands on the Mid-Atlantic ridge : these islands (Gough, Tristan da Cunha, St. Helena and Ascension) are small volcanic islands with no copper prospection possibilities.

1.2.8.2 Possessions in the Indian Ocean

- Kamaran Is. : these British uninhabited islands are in the Red Sea near the coast of Yemen. They have no significance in themselves but could turn out to be of importance if the Red Sea metalliferous muds (especially those bearing copper) were to be exploited, although present feasibility studies concern more northerly zones between 18° and 23°.

- The other islands listed below seem to be of no interest :

. Kuria Muria (G.B.) near the coast of Oman on the extension of Dhufar Qara ;
. the Amirantes, Seychelles and Chagos (Diego Garcia) Is. which belongto the U.K. ;
. Reunion and Mayotte (Comores) which are recent volcanic

islands ;

. the Southern French islands : Crozet, Nouvelle Amsterdam, St. Paul and Kerguelen. Copper occurrences have been noted in the latter but are only of academic interest since they are not linked to a spilitic series, as had been believed, but to the contact between Tertiary basalt and later intrusions.

1.2.8.3 Islands of the Pacific

There are a large number of British and French islands in the Pacific. The following can be distinguished :

- the large islands of the West, located in extension of the Indonesian arc and which are very interesting for mining exploration ;

- the small islands of the Central Pacific which are small volcanic islands without any real interest for prospection.

The large islands of the West

Since the Fiji Is. (previously U.K.) and the Franco-British condominium of the New Hebrides have become independent, the only remaing possessions of EEC countries are the Solomon Is. and New Caledonia.

- New Caledonia is well known for its production of lateritic nickel. It is less well known that it has produced about 12 kt of copper and that the figure of 25 kt has been proposed for the probable reserves. These figures are small but the minerals were worked with pre-industrial methods from 1885 to 1930 and very rich ores (5-25 % Cu) were involved. Recent studies have also shown that these are not veins but volcanosedimentary deposits.
  About thirty copper occurrences, including six old mines (Ao, Pilou, Balade, Murat, Meretrice and Fern Hill) with polymetallic ore (Cu, Pb, Zn, Au, Ag) are known in the Cretaceous volcanosedimentary

series in the valley of Diahot in the North of New Caledonia. The stratiform mineralization is associated with black carbonaceous schists and acid metatuffs in distinct Senonian horizons with intercalated levels of diabase, and silexite. This constitutes a quite well delimited sector in which new prospection could be successful. Some others volcanosedimentary environments (Permian and undated central masses) with copper occurences and some Cu-Mo disseminated mineralizations linked to granodioritic intrusions of Southern New Caledonia could be of some interest. On the other hand the results have been negative for the prospection of Ni-Cu sulphide deposits linked to ultramafic rocks.

- The Solomon Islands are located on the East end of the porphyry copper belt of the Philippines, Malaya (Mamut), New Guinea (Ertsberg, Ok Tedi, Frieda River, etc.), New Ireland, Bougainville (Panguna) which extends to the SE up to the Fiji Is.. These islands do contain mineralizations of this type especially in New Georgia and Guadalcanal. The most important prospect seems to be the porphyry copper deposit at Koloula (Guadalcanal I.) described by A.R. CHIVAS, and R.W.T. WILKINS in Economic Geology (vol. 72, 1977, pp. 153-169). It is still difficult to say what the economic impact of this type of mineralization could be on the Solomon Is.. The latter is certainly the overseas dependency of the EEC with the largest copper potential especially since the presence of massive sulphide linked to volcanism cannot be theoretically excluded as is the case for the Fiji Is..

The small islands of the central Pacific are largely French or British i.e. for the U.K., the Gilbert, Phoenix, Line, Tonga, Pitcairn Is. and for France, the Wallis, Horn, Society, Marquesas, Tuamotou and Toubouai

Is.. These are volcanic archipelagoes whose future development depends more on tourism than copper prospection !

Clipperton I. which is an isolated island off the coast of Mexico belongs to France. In itself, this island is certainly of no interest but it might become important later in connection with the exploitation of nodule fields located on the nearby deep sea floor to the West

1.2.8.4 Antarctica

The statute of Antarctica is still under discussion. France's share consists of Adelie Land between 136° and 142° and the United Kingdom has a large part of the continent between 20° and 80° including the island arc stretching from the Falkland Is. to the South Sandwich, South Orkney and South Shetland Is.. This island arc is the extension of the Western Cordillera of Latin America which is also known throughout the continent of Antarctica (Graham Land, Ellsworth Highland, Marie Byrd Land) and connects with New Zealand by means of the Macquary-Balleny undersea ridge. The geological context of this Late Cretaceous and Early Tertiary orogeny is very rich in copper in Latin America. In Antarctic Lands, little is known of the geology and mineralizations because of the climatic conditions which are as severe or even worse than those of northern Greenland. The area could however become interesting in the remote future.

We have not discussed Hong Kong and Gibraltar which are much too urbanised to be of any interest for prospection. The conclusion of this brief survey is that the only areas which have surfaces which are large enough and suitable geological contexts so that there might be a significant contribution to the copper supply of the EEC are the Solomon Is. and New Caledonia. Guyana, the Lesser Antilles and Antarctica are more remote possibilities.

1.2.9 Conclusions

1.2.9.1 EEC production and reserves tables

Table No. 4 concerns productions and most of the figures were supplied by the state offices of the EEC member countries. Production since 1901 is known relatively exactly. This is also true for production for a part of the 19th century to a greater or lesser extent depending upon the countries. However only estimates can be given for the periods before statistics began to be collected in the various countries. The figures given for these periods are probably often too low. It can therefore be concluded that Cu production up to now from ores mined in the EEC amounted to a minimum of 2 Mt of metal.

The distribution amongst the producer countries shows that four of them have had a large share. In order of importance these were the United Kingdom (>50 %), West Germany and Italy (17-19 %) and Ireland (9 %). The main mining regions of Cornwall, Harz, Tuscany and Southern Ireland accounted for about 3/4 of production.

Table No. 5 concerns "reserves". This term is in parentheses because part of the figures concerns subeconomic resources (cf. the remarks in the table). The table is also incomplete since only proved and published reserves (and resources) are listed. There are reserves —which are generally small however— situated in extension of old mines or in new prospects. for which tonnages and grades have not been published.

As a first estimate, it can be stated that proved reserves (and resources) are approximately equivalent to production in the past (2 Mt of metal). However

TABLE N° 4

COPPER PRODUCTIONS IN THE E.E.C.

in tonnes of contained copper in ores (recoverable)

Periods and years / countries and deposits	Before 1901	1901 1910	1911 1920	1921 1930	1931 1940	1941 1950	1951 1960	1961 1970	1971	1972	1973	1974	1975	1976	From the beginning to 1976 TOTAL	From the beginning to 1976 % E.E.C.
ITALY	160,000	123,000	22,200	4,565	3,738	6,220	4,014	13,939	1,537	1,048	859	912	767	823	345,000	17.7 %
FRANCE	30,000	1,109	290	1,165	4,846	3,393	4,834	3,283	248	358	338	354	400*	500*	52,000	2.6 %
of which SALSIGNE	-	-	-	168	4,462	2,919	3,737	3,162	248	358	338	354	400*	500*		
WEST GERMANY	← 320,000* →						21,746	16,100	1,484	1,321	1,436	1,734	1,961	1,600*	365,000	18.7 %
of which RAMMELSBERG	← 220,000* →						8,741	13,373	1,383	1,321	1,436	1,734	1,961	1,600*		
DENMARK (GREENLAND)	10	-	90	-	-	-	-	-	-	-	-	-	-	-	100	-
NETHERLANDS	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
LUXEMBURG (STOLZEMBOURG)	100	10*	10*	-	80*	-	-	-	-	-	-	-	-	-	200	-
BELGIUM (ROUVEROY)	5,000	-	-	-	-	-	-	-	-	-	-	-	-	-	5,000	0.3 %
UNITED KINGDOM	1,000,000	6,350	2,300	875	460	185	150	50	-	225	450	440	425	500*	1,012,000	51.9 %
of which PARYS MOUNTAIN	130,000	1,500*	1,100*	700*	460	185	140	50								
of which CORNWALL-DEVON	870,000	4,000*	1,000*	50*	-	-	-	-	-	225	450	440	425	500*		
EIRE	70,000?	100	200	100	-	-	7,072	23,535	11,625	14,166	12,273	13,599	11,262	4,318	170,000	8.8 %
of which AVOCA	20,000	-	-	-	-	-	7,072	8,090	6,660	9,576	8,332	6,435	4,400	3,697		
GORTDRUM	-	-	-	-	-	-	-	15,445	4,965	4,590	3,415	5,883	6,464	-		
TYNAGH	-	-	-	-	-	-	-	-	-	-	526	1,281	398	621		
TOTAL	← 1,765,000 →						37,816	56,907	14,894	17,260	15,420	17,090	14,920	7,750	1,950,000	100 %
OVERSEAS TERRITORIES of which NEW CALEDONIA	← 12,000 →														12,000	
TOTAL																

* estimated up to 1950
Approximate after

TABLE N° 5

RESERVES IN THE E.E.C.

DEPOSITS AND COUNTRIES (Currently mined deposits are underlined)	PROVED AND INFERRED RESERVES			BY OR CO-PRODUCTS	REMARKS
	TONNAGE (kt)	GRADE	CONTAINED COPPER kt		
AVOCA (1.1.1977)	8,438	0.86 % Cu	72,5	200 kt 8-9 % Pb + Zn Pyrite	Recovable reserves estimated at 4.2 Mt with 0.74 % Cu but the potential is much larger
TYNAGH (12.31.1976)	2,450	0.18 % Cu	4,4	+ 3,89 % Pb + 3,75 % Zn + 28 g/t	About 3,500 t recovable
GORTDRUM (estimated)	1,000	1,2 % Cu	12,0	Hg	Not worked because of problems of open pit flank stability
MALLOW	3,600	0.70 % Cu	25,2	+ 25 g/t Ag	Not worked
AHERLOW	5,500	0.89 % Cu	49,0	+ 40 g/t Ag	
BALLYVERGIN	140	1.00 % Cu	1,4	(Pb)	Can be increased by 50 %
ALLIHIES	1,500	1.6 % Cu	24,0		The potential is probably larger
TOTAL EIRE	22,628	0.84 % Cu	188,0		
WHEAL JANE	5,000	0.3 % Cu	15,0	Sn	Cornwall, Cu grade of Mt Wellington supposed to be 0.3 %
Mt WELLINGTON	5,000	0.3 % Cu	15,0	Sn	
COED Y BRENIN	200,000	0.3 % Cu	600,0	Mo (Pb + Zn)	Subeconomic reserves
PARYS MOUNTAIN	40,000	0.41 % Cu	160,0	Zn ?	The estimate might be increased by recent drilling
TOTAL UNITED KINGDOM	250,000	0.32 % Cu	790,0		Very low grades
SONTRA	40,000	1 % Cu	400,0		
DILLENBURG Area	30	1.5 % Cu	0,5		
NIEDERMARSBERG	600	1.6 % Cu	10,0		Calculated grade
BAYERN	300	0.55-1.2 % Cu	2,5 ?		Estimated metal tonnage
RAMMELSBERG	2,700	0.6 -0.7 % Cu	17,5	Pb - Zn - Ag	Higher potential
TOTAL WEST GERMANY	43,630	0.99 % Cu	430,0		
FUNTANA RAMINOSA	1,150	about 1 % Cu	12,0	1.4 % Pb + 3 % Zn	(Mixed sulphides)
CAMPIGLIA MARITTIMA	700	0.8 - 1 % Cu ?	6,0	(Pb - Zn)	Inferred reserves
FENICE CAPANNE	9,000	0.3 - 2 % Cu	100,0	(Zn - Pb)	The metal content is very approximate and may be underestimated.
ALAGNA VALSESIA	150	1 - 1.5 % Cu	2,0		100 kt of proved reserves
PREDOI	250	2 % Cu	5,0		50 kt of proved reserves
BEDOVINA (Mt MULAT)	1,200	0.8 % Cu	10,0	+ 0.25 % W	100 kt proved
TOTAL ITALY	12,750	1.09 % Cu	139,0		
BELGIUM (ROUVEROY)	1,500	1 % Cu	15,0		Resources
SALSIGNE	6,000	0.25 % Cu	15,0	Au, Ag, Bi, As, S	By-product of gold
DOME DU BARROT	200	2.0 % Cu	4,0	(+ Ag)	Insufficient
BANCA BAIGORRY	120	3.0 % Cu	4,0	+ 25-30 g/t Ag	Insufficient
BRITTANY PROVINCES (BODENNEC, LA PORTE AUX MOINES, ROUEZ)	28,700	0.6 % Cu	218,0	Pb-Zn (1 to 10 %) Ag	The potential is larger than these official figures indicate especially at Rouez (with grade < 1 % Cu).
TOTAL FRANCE	35,020	0.69 % Cu	241,0		
GENERAL TOTAL	365,528	0.5 % Cu	1,803,0		

apart from the reserves of existing mines amounting to about 250 kt, most of the other resources are not mineable under present technological and market conditions either because they are in mines which are too small or because the grade is too low.

The total reserves and resources are distributed amongst five countries which in order of the size of the metal content are as follows :

- low grade resources in the United Kingdom (45 % with Coed y Brenin and Parys Mountain) ;
- West Germany (22 % with Sontra but this does not take into account the kupferschiefer potential which is mainly unknown) ;
- the polymetallic reserves of France (12 % with the Brittany province) ;
- the reserves of Ireland (10 % with Avoca and the SW deposits) ;
- the reserves of Italy (8 % principally at Fenice Cappane).

The proved reserves are thus mainly in the old production zones except for the West German kupfer-schieffer which have been worked only slightly and the new volcanosedimentary province of Brittany.

1.2.9.2 General conclusions

The production of copper from ores mined in the EEC seems today to be negligible. In 1976 it did not even reach 8 kt i.e. around 0.1 % of world production and 0.4 % of EEC copper consumption. The EEC is therefore obliged to import, in one form or another, almost all the copper that it processes and consumes.

This situation leads to the expenditure of foreign exchange and a great dependence upon supply sources. The latter point which is of little significance in the present period of overproduction and very low prices could become a very serious matter in the medium and long term.

This situation cannot be improved to any considerable extent in the next few years unless an attempt is made to prospect for new resources. The total proved resources of the EEC are not negligible but amount to only a year of consumption. It is therefore necessary to prospect more intensively for copper. The expenditure for exploration should not be too closely linked to copper prices. The latter change very quickly while there is generally a period of 8 to 10 years between the discovery of a new ore body and the starting of mining production. It is also true that one of the major conditions for success in metal exploration is perseverence and continuity in the work. It would therefore be wise to use the present period with its depressed market and favorable external supply conditions for prospection investments, to find copper ore in EEC.

Are there sufficient prospects to justify this exploration ?

The EEC countries have the reputation of being poor in copper and this seems to be confirmed by the figures given above. This is however only partly true :

- production in the past amounting to about 2 Mt of metal was not negligible and this is an honorable score for a surface of 1.5 million km^2, most of which consists of barren post-triassic sedimentary basins ;
- there are many occurrences and small deposits in the Preliassic formations and the Alpine chain. In addition, even regions which are poor in mineralized showings should be explored again since shortly before the discovery of the deposits of Brittany,it was possible to write that "one of the most marked characteristics of the metallogenic province of the

Armorican Massif is its copper poverty". This phrase expressed the state-of-the-art rather than the mining possibilities of this region as later discoveries have shown ;

- since 1950 there have been many discoveries of new deposits or extensions of old mines i.e. Avoca, Gortdrum, Mallow, Aherlow in Ireland, Coed y Brenin in Wales, Fenice Cappane in Italy and Porte aux Moines, Bodennec and Rouez in Brittany. These new deposits have the largest part of the proved reserves. These discoveries are especially promising since they were the result of new effort of prospection in the EEC countries, after a period in which attention had been concentrated on countries known to be richer in raw materials and on ancient colonies of some of the EEC members.
- lastly known figures (cf. 1.2.10) allow to state that the cost of a discovery in the EEC (at least in Ireland and France) during the last twenty years has been lower than in some others countries (Canada and Australia, for example).

The above remarks indicate that there can be a reasonable expectation of discoveries at prices which will not be prohibitive and, that the application of modern prospection methods to zones considered favorable, will very probably lead to new economically feasible discoveries. The latter point will be further developed in the general conclusions of the report where the types of deposits and prospection zones will be defined as well as the prospection methods to be used and if possible improved.

1.3 Notes on exploration costs and expenses

Unit costs for exploration, such as prices per km of airborne geophysical survey, for geochemical sampling or for one meter of drilling, are variable but are within a relatively limited and well known range in the European context. These figures are very useful for the exact definition of a programme on one prospect, but cannot be used for estimating total exploration expenses for a given country.

Costs per square kilometer are a first approach to a general evaluation. They also vary a great deal depending upon the geological and climatic conditions and the advancement phase of the prospection under consideration.

Aside from drillings, the necessary expenses per km^2 can roughly be estimated at $ 100 for regional geological and geochemical exploration and at $ 1,000 for more detailed prospection on a smaller surface containing anomalies and occurrences. As far as we know, however, few general reports have been published concerning prospected surfaces and methods used for exploration in a given country and no data of this kind is available at all for Europe.

At the later stages of target study and deposit development, reconnaissance drilling and mine work are a much larger expenditure and the idea of expense per km^2 no longer has any sense. Drilling prices in Europe vary from 200 to 800 FF per meter depending upon the drilling-machine and the depth of the hole.

The total exploration expenses are almost the only figures which are available in the literature. A high degree of prudence is however appropriate as to their meaning and exactness. The degree of accuracy varies considerably depending upon the private companies which consider this information relatively confidential. The public shares themselves are not always very well known since this expenditure is distributed amongst various government agencies and private companies. With some exceptions, there is thus a certain degree of uncertainty as to the total exploration expenses for the various countries. This uncertainty become greater in the case of prospection for copper alone.

There are, however, certain publications on this subject and D.R. DERRY proposes figures on the worldwide scale. According to him, exploration expenses for metals were about 2 % of metal mine production at the beginning of the decade of the 1950's and 4 % at the end of the 1960's. This percentage has probably increased in recent years. It is usually assumed that exploration expenses for a deposit must not exceed 10 % of the metal content.

Some relatively exact data exists for certain countries, especially Canada and Australia. In Canada, one of the main mining countries, published figures are relatively complete for the period 1950-1976. According to W.E. ROSCOE and D.R. DERRY, total exploration expenditures have increased 10 times in 25 years (13.2 M$ in 1951 and 132 M$ in 1975 in constant 1976 dollars). During the same period, the average cost of a discovery rose from 15 M$ (1951-1956) to 28.6 M$ (1966-1976). The average for the first twenty years shows that the total prospection costs amounted to 4.6 % of the accumulated turnover of the mining companies.

The statistics published in Australia are less complete but it is still possible to deduce that from 1965 to 1971, the average cost of a discovery was 24 M$ which is comparable to the Canadian figure.

A remark should be made as to this "discovery cost". It is not certain that all the publications which refer to it mean "discovery of a deposit with a grade and tonnage sufficient to justify production in normal conditions for metal prices and capital return" as defined by D.R. DERRY. In addition, the figure obtained is of great interest for comparing the efficiency of the prospection methods used, but not for judging the mining potentiality of a given country. In order to do this, it would be necessary to take into account the value of the metal contained in these discoveries and its appears that this study remains to be made.

For the EEC, certain figures have also been published or can be obtained by a documentary study, especially for Ireland and France. In Ireland, total exploration expenses up to 1973 amounted to 14.8 M£ and this made it possible to discover four new deposits. The average cost of a discovery can therefore be extimated at 8 M$ which is thus much lower than the Canadian and Australian costs.

In France, a recent study by J.P. HUGON shows that mining prospection expenses for French companies (except for hydrocarbons and coal) rose from 300 MF in 1971 to 469 MF in 1976 (in current francs). During this period, the share for the national territory (metropolitan France and overseas departments) was about 50 % (1,032 MF) and has been increasing slightly.

The share for copper prospection has oscillated between 11.7 % and 23 %, and from 1970 to 1976 amounted to a total of 304 MF (current francs) with most of this for prospection outside of Europe.

Variations in the figures published for the EEC countries make it practically impossible to know the amount used for copper exploration in the ECC. Expenditure for exploration by the 14 main mining companies of the EEC between 1961 and 1975 were estimated at 1,378 M$ (constant 1976 $) of which 356 M$ in Europe. However this sum is for all of Western Europe and not only for the EEC countries. In addition, money spent for exploration in the EEC by foreign companies are often unknown. Total expenditures for exploration in the EEC is thus very uncertain and the share for copper prospection alone is unknown and may even be impossible to determine.

However it can be supposed, on the basis of some partial data for certain companies and through extrapolations from total figures, the total for copper prospection in the EEC countries since 1960 represents a figure of between 20 and 40 M$ (in 1976 dollars). As a result of the impact of discoveries and the allocation of funds by the governments of some countries (for example the French copper plan) the annual amount is no doubt increasing but it is not possible to give an exact figure. Lastly, by km^2 the mining prospection investment in the EEC was approximately the same as in Canada but in Europe the relative share of copper is definitely lower. Thus in spite of recent encouragement in this direction, expenditure for copper prospection withing the EEC has not been especially large during the last 25 years.

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2. MINING

2. MINING

2.1 Mining Methods

2.1.1 General Remarks

Before the beginning of the 20th century all copper mine production came from underground mines with an average ore grade which was between 18 and 20 %.

The concept of large scale mining with the resulting economies of scale first appeared in 1899 when D.C.Jackling and R.C.Gemmell planned to extract and process 2,000 tpd of 2 % grade ore from the Bingham Canyon porphyry deposit in Utah (US).

A capacity of 500 tpd seemed quite large at the time. The starting up of the mine in 1906 showed that the open pit mining of low grade ores was economically sound.

Today copper porphyries account for 50 % and open pit mines for 60 % of the Western World's copper production. In the US, which is the world's largest producer, open pit mines have in recent years covered about 90 % of the ore produced and 80 % of the recovered metal. In a very short time mines with larger capacities than the 2,000 tpd of Jackling and Gemmell were opened and the Bingham mine owned by Kennecott Copper Corp. has a capacity of more than 100,000 tpd of ore. This is the largest mine in the world and in 1974 up to 500 kt of ore and waste were moved in a single day.

Thus, the development of open pit mines marks a considerable change in mining techniques because of the modern high capacity equipment. However, recent prospection for deep deposits and the comparatively unfavorable trends for some cost factors such as energy could increase the relative share of underground mines in copper production. This trend is quite marked in the US.

TABLE N° 6

DISTRIBUTION OF NUMBER OF COPPER MINES IN THE WESTERN WORLD

ACCORDING TO ORE PRODUCTION CAPACITY (1977)

Annual Capacity	3 Mt	1 to 3 Mt	0.5 to 1 Mt	0.3 to 0.5 Mt	0.15 to 0.3 Mt	TOTAL
OPEN PIT						
North America	29	11		1	2	43
Latin America	5	1		1	3	10
Europe	2	1	1			4
Africa (CIPEC)	3	2	1	1		7
Africa (non CIPEC)	1	2			1	4
Asia	1	2		2	3	8
Australisia/Philippines	6	3	3	1	1	14
Sub-Total Open Pit	47	22	5	6	10	90
UNDERGROUND						
North America	5	5	8	2	6	26
Latin America	3	2	3	2	4	14
Europe			2	5	11	18
Africa (CIPEC)	2	3	4			9
Africa (non CIPEC)		2	4	4	1	11
Asia	1		3	10	7	21
Australisia/Philippines	3	2	2	1	1	9
Sub-Total Underground	14	14	26	24	30	108
OPEN PIT AND UNDERGROUND						
North America			1	1	1	3
Latin America					1	1
Europe		1	3	2	3	9
Africa (CIPEC)	3					3
Africa (non CIPEC)		2				2
Asia		1			1	2
Australisia/Philippines						
Sub-Total Open Pit and Underground	3	4	4	3	6	20
TOTAL	64	40	35	33	46	218

Table N° 6 shows the distribution of the number of copper mines in the Western World according to mining method and ore production capacity. The histogram (Fig. 9) can then be plotted with five classes depending upon the size of the mine. Although small mines with less than 150 ktpy were not included, there are large differences depending upon the mining method. For all mines considered, the relative distribution for open pit, underground and mixed mines is 41 %, 50 % and 9 % respectively. For the class of mines with the highest capacity (> 3 Mtpy) this distribution becomes 73 %,22 % and 5 %. For the three lowest classes (< 1 Mtpy) the distribution is on the average 18.5 %, 70 % and 11.5 %.[1]

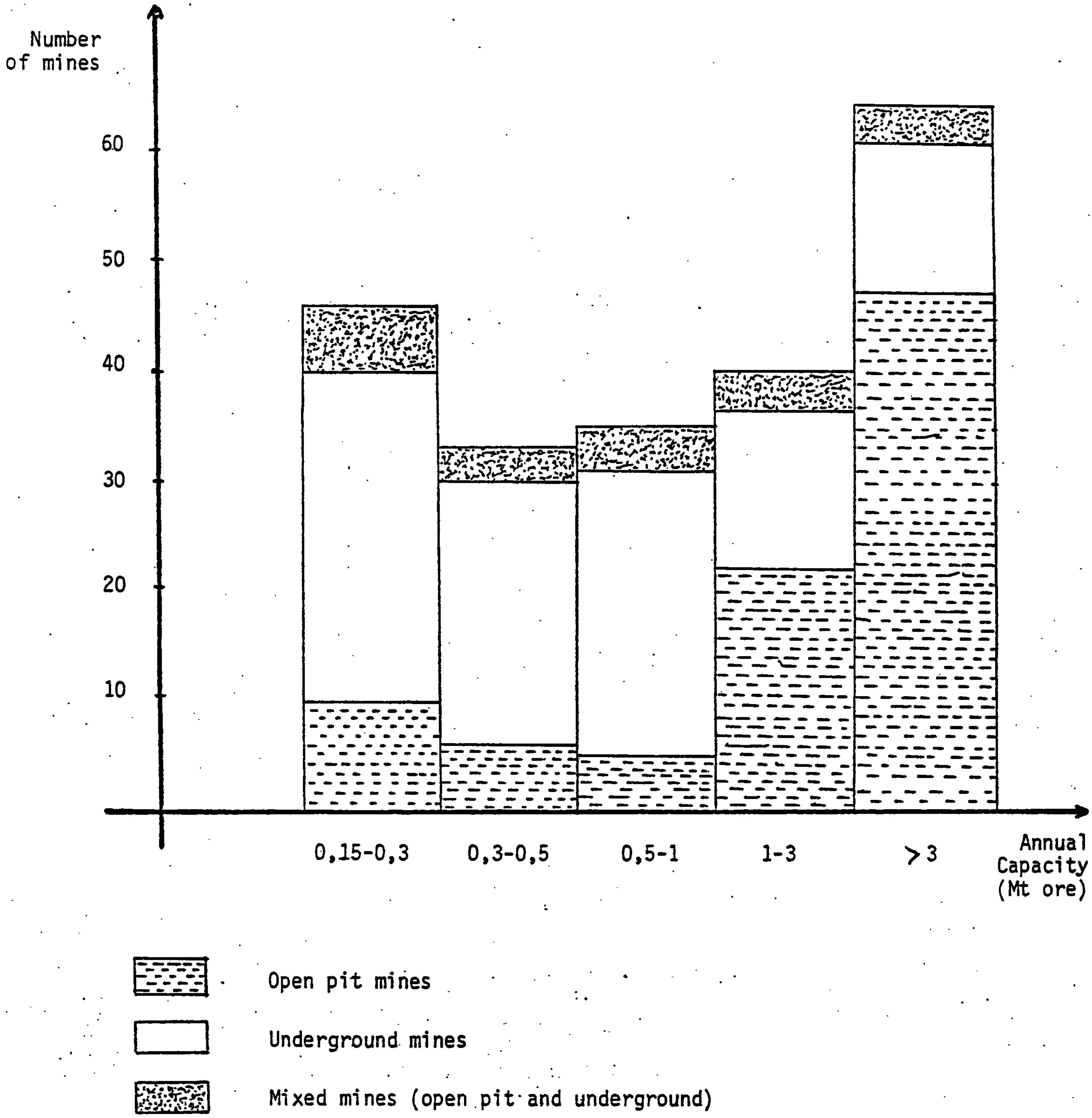
The average grade of the ores now being worked is on the order of 1.5 % but there are large differences between the various countries. In 1974, mines in North America had an average grade of 0.65 % while this figure was 0.97 % in Latin America, 1.11 % in Europe, 2.92 % in Africa, 0.59 % in Asia and 2.39 % in Australia.[2]

In recent decades the large increase in the demand leads to the mining of deposits with increasingly low grades.

Technical progress especially for the open pit mines has made it possible to produce the necessary amounts up to the present time. The problem of the limit has now appeared and the question is how low can the grade of a deposit be and still be profitable. Along with the economic problem there is also the question of the physical resources. While it is true that lowering the grade increases reserve levels, there is nevertheless a lower limit. The decrease in workable grades along with the exploitation of new resource such as seabed nodules could push even further back these limits which are trying to define. There might, however, be problems involved in the exploitation of these new resources. Thus the growing energy cost in mining could turn out to be the basic obstacle before the physical resources are exhausted.

Fig. 9

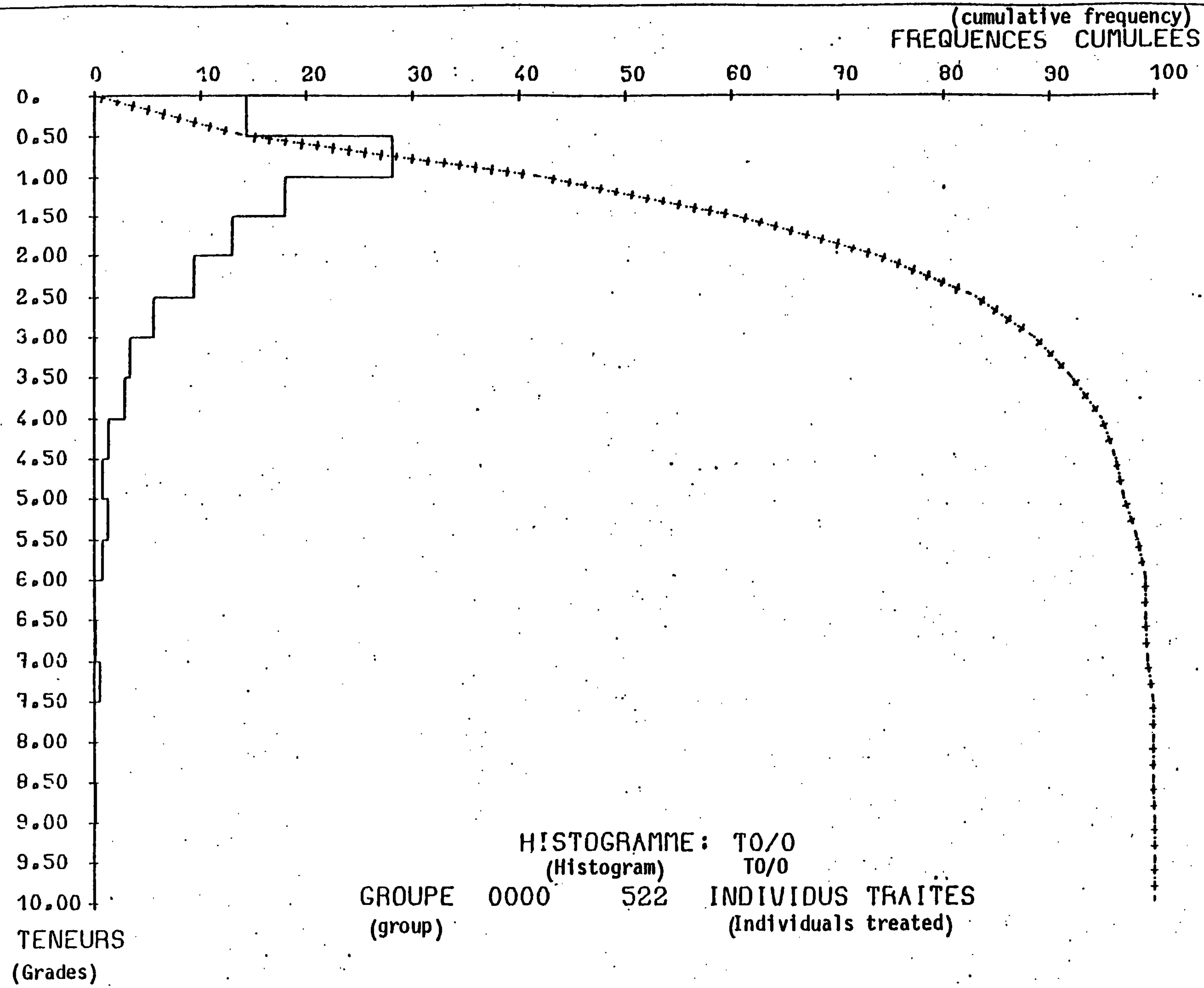
HISTOGRAM OF DISTRIBUTION OF COPPER MINES IN WESTERN WORLD



In the US, the average grade of the ore worked was about 1.6 % around 1920 and then fell to 1 % in 1943 reaching 0.6 % in 1970. Experts forecast that in the year 2000 the ore grade worked in the US could be an average of 0.25 %. The lowest grade worked today is close to 0.20 %. At this level the economic importance of the byproducts is essential for the operation to be profitable.[3]

Histogram (Fig.10), based upon the grades of more than 500 mines now being worked or projected, shows the distribution of average ore grades.[4]

There is a great deal of variety in the copper ores, deposit configurations, country rock, copper grades and aptitude for concentration upon which market value depends. The result is that most of the different underground mining methods are used.



* HISTOGRAMME DES TENEURS *
(HISTOGRAM OF GRADES)

Fig. 10

2.1.2 Open Pit Mines

While 60 % of the copper production of the Western World comes from open pit mines, this proportion is even higher in countries like the US, Canada, Chile and Peru where the deposits are essentially of the porphyry copper type.

For open pit, deposits are required which are relatively shallow and which have contours which can be determined easily at the exploration stage.

The development of high capacity equipment has made it possible to work these relatively poor deposits which are profitable only because of economies of scale. The essential phases of the exploitation are stripping of the overlying terrain, blasting and the loading and transport of the ore and waste mainly with electrical shovels and large trucks. Work can be carried on simultaneously on several benches of the pit.

The essential point in the study and operation of an open pit mine is often the transport problems. The viability and profitability of the exploitation are often conditioned by factors such as the location of the processing units, roads and means of access and the choice of equipment.

Table 7 shows the distribution of open pit copper mines by capacity and country or group of countries in the Western World in 1974 [5]

It appears that about 50 % of the production is accounted for by mines with an annual capacity of more than 75 kt of metal content. Europe holds a kind of record since 40 % of the production comes from mines with less than 25 ktpy and this is explained by relatively small size of the European deposits.

TABLE N° 7

DISTRIBUTION OF OPEN PIT COPPER MINES IN THE WESTERN WORLD BY CAPACITY

	< 25 ktpy Cu cont.		25-75 ktpy Cu cont.		> 75 ktpy Cu cont.	
	Number of mines	% Production of country	Number of mines	% Production of country	Number of mines	% Production of country
US	17	14	9	34	6	52
CANADA	9	27	6	73		
LATIN AMERICA	7	10	4	27	2	63
EUROPE	9	40	2	60		
AFRICA	17	21	3	13	4	66
ASIA	13	35	4	38	1	27
AUSTRALIA and P.N.G.	6	15			1	85
TOTAL	78	18	28	32	14	50

2.1.3 Underground mines

The following are the principal mining methods for underground copper mines :

- Open Stopes :
  . Room and pillars
  . Sublevel stoping
- Filled Stopes :
  . Cut and fill stopes
- Shrinkage Stoping
- Caving Methods :
  . Sublevel caving
  . Block caving

The choice of methods depends upon many parameters including deposit configuration (geometry, dip, thickness), ore grade, reserves, stability of vein walls, manpower costs and quality and the desired level of mechanization.

We will briefly review the field of application of each of these methods.

2.1.3.1 Open Stopes

In open stope methods the space left by the blasting and extraction of ore remains and is not filled either by filling or by caving.

Two open stope methods are especially used for copper ore :

- Room and Pillar Method : This method is used for very thick tabular deposits. It consists of leaving behind pillars which hold up the roof.

It makes possible intensive mechanization and efficiency is higher than in the other methods except for the caving methods. Productivity is on the order of 10 to 15 t/man-shift for all services together.

- Sublevel Stoping Methods : The ore is mined in vertical slices from sublevels cut into the mineralized body. The ore is then withdrawn through a network of chutes and draw cones to the principal levels.

This method is used for deposits whose average thickness is low and whose dip is high (> 50°) in strong country rock. Productivity is less than 10 t/man-shift for all services together.

2.1.3.2 Cut and Fill Stoping

If it is necessary to support the ore, the gap left by extraction is immediately filled in with waste with a suitable granulometric composition. The fill serves as a mark platform for mining the next slice. The capacity depends upon the rate of fill and productivity for this method is lower than for the other methods described here. It is about 5 t/man-shift for all services together.

2.1.3.3 Shrinkage Stoping

This method resembles the preceding one except that it is the ore itself which is used for fill. During mining only the swell is cut and removed through the main levels. All the ore is removed only after the whole panel has been blasted.

This method is used for narrow deposits with a high dip angle (> 50°). Productivity is approximately 5-10 t/man-shift, for all services together.

2.1.3.4 Caving Methods

- Sublevel Caving : This method is related to the sublevel stoping method above but here the mining of the ore leads to the collapse of the roof. Precautions must therefore be taken in order to avoid dilution..

Sublevel caving allows a high degree of mechanization and is the most productive along with the block caving method. Productivity is about 50 to 60 t/man-shift.

- Block Caving : The block caving method is used for thick masses or layers (at least 20 m) of tender ore. It is used for large capacity operations and makes it possible to reach high levels of productivity (50-60 t/man-shift).

A network of chutes and cones is cut into the lower part of the deposit for bringing the ore to the main level. A space is excavated in the entire surface above this network and then the upper part of the ore is weakened by cutting galleries and chutes which lead to the collapse of the ore under its own weight.

2.1.4 New Methods

New mining methods are being developed in addition to the conventional open pit or underground methods such as in situ leaching which is an extension of ore dressing processes. This method will be discussed in the following chapter.

2.2 Economic Data for Mines

2.2.1 General Remarks

Production costs and investments for mines are due to two series of factors. The first series consists of factors inherent in the deposit such as its location and specific characteristics which largely determine the mining method as the ore grade and its physical and chemical structure. Other factors are local and depend upon the economic and social conditions in the country involved such as manpower costs and quality, transportation costs for supplies, availability of energy sources, taxation, etc. All these elements affect in one way or another what can be considered the standard costs for a given mining method.

We will indicate some of the most important factors for production or investment costs.

2.2.1.1 Influence of the Mining Method

Obviously the essential factor for copper mining costs is the mining method used.

We saw above that productivity in underground mines varied a great deal depending upon the method used and that the average for the most efficient methods was 50 t/man-shift. This yield is over 100 t in open pit mines. The type of equipment used and its age and degree of wear along with the experience and relative productivity of the labour force also lead to significant variations around the indicated averages for the same methods. Production costs also follow these variations especially since consumption of supplies, energy and spare parts depends upon the mining method selected. In terms of a pound of copper recovered it is thought that the production costs for an underground mine are about twice those of an open-pit mine if the underground method is selective.

The ratio can be as high as four or six for mass mining methods.

Open pit mines are also less expensive in terms of investments per tonne of ore capacity.

While the open pit mines seem to be much more attractive it should be noted that the ore is in general of a much lower grade. While comparisons are useful, choice of the mining method depends much more on the characteristics of the deposit than economic considerations which are only a criterium for deciding the start of mining once the method has been determined.

2.2.1.2 Effect of Size of Mine

As in most industries, the unit costs for copper mining generally fall as the size increases. Thus in 1973 the Bureau of Mines[6] estimated that production costs (in 1970 value) for an open pit mine ranged from 39 ¢ per tonne of ore for a 30 ktpd mine to 30 ¢ for a 100 ktpd mine. This is also the case for underground mines although here the essential factor is the mining method.

Scale seems to be of lesser importance for investments. A comparison of total investments for current prospects shows that the large prospects support important infrastructure costs which counterbalance economies of scale for the mine and the ore dressing unit.

Whatever the economies of scale may be, however, the factors affecting the capacity of the exploitation will be deposit size or market conditions rather than simply an optimum unit cost.

2.2.1.3 Influence of the local factors

For several items of expenditure,costs can be determined on a technical basis but these figures are affected by a local factor. Thus energy consumption expressed in kWh per tonne of ore will generally be independent of local conditions but linked to the mining method. The cost of this energy will be the product of the technically determined consumption figure and the unit cost of energy in the country or more precisely at the site of the mine. Many items are thus affected in this way by local cost conditions.

As concerns direct production costs, the principal factors to be considered are labor costs, which may be affected by local productivity conditions, and energy and supply costs. In a 1976 study for the USBM, the Stanford Institute estimated comparative cost elements for several countries and the US [7].Table 8 shows these estimates for several countries. Without giving special emphasis to the figures themselves which are subject to frequent variations, large differences can be observed between the various countries especially for energy and manpower. As concerns manpower the jobs held by expatriates should also be taken into account in the case of the Less Developed Countries and this cost can be four or five times that of local personnel with the same qualifications. The training of local personnel should also be taken into account in this case.

In the same way, taxation and tariffs depend entirely on the country considered or are linked to special agreements in the case of particular mines.

Local factors also have an effect as concerns investments since various amounts may be added to total cost of the project for the construction of infrastructures or social equipment which may not exist or are insufficient at the mine site.

TABLE N°8

COMPARATIVE COSTS FOR LABOR, ENERGY AND SUPPLIES IN SEVERAL COUNTRIES

COUNTRIES	Labor	Energy, utilities	Supplies, equipment
US (base)	100	100	100
LATIN AMERICA			
MEXICO	36	60	153
PERU	30	120	135
EUROPE			
FRANCE	59	75	106
W. GERMANY	72	110	95
ITALY	62	220	120
IRELAND	41	120	132
SPAIN	40	100	124
AFRICA			
ZAIRE-ZAMBIA	60	40	150
SOUTH AFRICA	45	40	145
ASIA			
JAPAN	46	260	120
AUSTRALASIA			
AUSTRALIA	56	60	160

While all the local factors do not affect mining costs to the same degree, these costs like all the production costs or investments are linked to the unit costs for manpower, energy and materials as well as the transport costs specifically bound to the location of the mine or project.

2.2.2 Characteristics of Open Pit Mining Costs

2.2.2.1 Production Costs [5, 6, 7]

Open pit mining costs are generally expressed per removed tonne (ore or waste). These costs can vary from $ 0.2 to 0.8 per tonne removed depending upon the size of the mine and its location. In North America the range is reduced to $ 0.3-0.4. 25 to 30 % of the direct costs are for drilling and blasting, 40 to 50 % for loading and transport and the rest is divided amongst maintenance, supervision and administration.

These direct costs usually break down as follows : 40 % for labor, 15 % for energy and 45 % for supplies and spare parts. The overheads which are in addition to the latter items usually amount to 20 to 25 % of the latter.

It is estimated that mine production costs for an open pit mine represent 10 to 12 % of the production costs of the entire operation up to the production of the metal.

2.2.2.2 Investments [6]

It is difficult to determine the exact share of mining expenses in the investments for a project. Infrastructures and construction work, engineering expenses and general preproduction expenses are not always separated. It is estimated, however, that in the US in recent years investments, in 1970 value, for open pit mine varied from $ 119 to 172 per removed tonne of daily capacity. This amount covers only equipment and associated plant. This figure can be estimated today at $ 200 to 250. However this in most cases represents only about 5 to 10 % of the total investments of a mine project for the production of concentrates. For the mine itself, some additional work must be added for preparation, infrastructure and stripping and these are highly dependent upon the deposit and its location.

2.2.3 Characteristics of Underground Mine Costs

2.2.3.1 Production Costs [5, 6, 7]

Production costs for an underground mine are more closely linked to the characteristics of the deposit and the method used than in the case of open pit mines. The stability of the soil for example determines how much support will be necessary and special work in the main galleries may be necessary. Water may be present so that a great deal of pumping may be needed while ventilation may be difficult, etc... The essential factor will however be the mining method chosen. It should also be noted that in the same mine several methods can be used simultaneously in different parts of the deposit.

The specific characteristics of each mine and of each method makes it difficult fo give a distribution for the various operations i.e. development, blasting, loading, transport, maintenance, etc...

Table 9 shows the cost distribution for operating costs between labor, supplies and energy for four mining methods. The costs shown should be considered as orders of magnitude in 1976 values per tonne of ore produced for average size mines. The size of the mine

TABLE N° 9

STANDARD COST STRUCTURE FOR UNDERGROUND MINES

(in $/t of ore produced - 1976 value)

METHOD	Cut and Fill	%	Room and Pillar	%	Shrinkage Stoping	%	Block Caving	%
Direct Costs :								
Labor	6,31	56	1.56	46	4,64	62	1,60	58
Supplies and Spare Parts	4,38	39	1,68	49	2,45	33	1,11	40
Energy and Utilities	0,57	5	0.16	5	0,35	5	0,05	2
Total Direct Costs	11,26	100	3,40	100	7.44	100	2,76	100
Overheads (as percentage of direct costs)	18.2 %		28.2 %		19.0 %		15.9 %	

is a very important factor for production costs and the costs for cut and fill stoping can be divided by 2 or 2.5 when capacity rises from 10,000 tpd to 25,000 tpd.

Analysis of the table shows that there is considerable diversity in costs and that this is linked to a considerable extent to the average size of the mine in the case of the various mining methods. Block caving and, to some degree, the room and pillar method, show higher daily capacity with more mechanization and with equipment with greater capacity.

In relation to open pit mines the share of labor in the production costs is greater (50 to 60 %) while energy accounts for only 5 % of these costs.

The high costs for the cut and fill method explain the fact that it is reserved for deposits or parts of deposits where it is the only possible method and where the ore value is high. Block caving methods are justified in large mines with massive low grade deposits such as deep porphyry copper deposits.

2.2.3.2 Investments [6]

As is the case of open pit mines, aggregate investments figures make it difficult to estimate specific costs for each mine. However the USBM estimated the following costs for 1973 in 1970 values : $ 2,160 per tonne of daily capacity for a cut and fill mine, $ 630 for a room and pillar mine and $ 840 for a block caving mine.

While these figures are closer to the real costs of the mine, preparatory work for infrastructure and associated construction work etc., should be added. In a current project in a less developed country the mine investments are close to $ 4,500/t/day (1977 value) for a sublevel stoping mine. This amount represents 21 % of the total investments but if specific infrastructure and equipment which are necessary because of the local context, are excluded, this share rises to about 40 % of the industrial investment up to the copper concentrate production stage.

2.3 Recent Developments and Future Perspectives

2.3.1 Open Pit and Underground Mines

Since the Bingham mine started operations in 1906, open pit mining has developed at a very rapid rate but recent developments affecting several factors have tended to reduce the advantages of this method in rélation to underground mining where there seemed to be fewer possibilities for progress.

As a result of the development of prospection techniques such as geophysical methods there are now increased possibilities for the discovery of deeper deposits. Under present economic conditions it does not seem possible to exploit low grade deposits if they are very deep since open pit methods cannot be used while the stripping ratio rises rapidly with increasing depth. Progress in the field of selectivity for underground mines favors the working of deposits which are narrow but have high grade ore. In general terms it is true that once the most accessible deposits have been worked it will be necessary to prospect for resources at greater depth.

The sharp increase in energy costs made underground methods a little more competitive. In a 1974 study, the CRU [8] estimated energy consumption for open pit mines at 12 to 25 kWh/t of ore. Consumption was 20 to 60 kWh for underground mines. The ratio of the respective ore grades gives a lower consumption for the underground mines. The difference may seem to be relatively small but it has a tendancy to increase since open pit mines have increasingly low grades.

Environmental factors are being taken into account especially in the developed countries and this has affected the copper industry. The greatest effects were encountered in the field of the processing of copper concentrates but problems have already appeared at the mining level such as the reconditionning of the soil after open pit mining. The environmental factor alone can affect the choice of the mining method when no one method is necessarily indicated and the nature of the site is such that there will be problems and additioning costs for environmental reasons.

This analysis which increases the importance of the factors favoring the underground mining of rich deposits, should be taken into account as a possible trend in the middle and long term but it remains true that open pit mines alone account for 80 % of new project capacity and 67 % of the projected extensions. This is due to the fact that 70 % of planned new capacity concerns porphyry copper type deposits. [9]

The discovery of increasingly deep deposits and the need to exploit them makes it hard to forecast the long term trend for conventional mining techniques. New technological developments will be necessary in these methods in order to make the indispensable copper resources accessible at a reasonable cost especially in view of energy prices.

2.3.2 Conventional Methods, New Techniques and New Resources

The increased costs involved in working deep deposits with conventional methods is encouraging further research for new mining techniques combined with new processing methods. The whole production sector is therefore being reconsidered. In situ leaching of ores is an extension of conventional leaching techniques and might satisfy these new conditions.

Another approach to these problems is the prospection for new resources and development of processes for their treatment. Thus the exploitation of seabed nodules is faced with the special mining problem of collecting the nodules which is a completely new one and for which the mining industry and especially the copper industry have no particular competence. The solution for these new problems must therefore come from close cooperation between companies in different sectors. The mining industry will be an indispensable element but will not be the only.

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3. MINERAL PROCESSING

3. MINERAL PROCESSING

3.1 Types of Ores and Selection of a Processing Method

3.1.1 Types of Ores

Copper occurs in nature in the form of at least 160 different mineral species (only a few of which are of economic importance) with specific physical properties and chemical compositions and which are closely mixed with gangue minerals with little or no economic importance (quartz, feldspar, calcite, barite, etc.).

From the mineral processing point of view, two main groups of mineralisations are distinguished :

3.1.1.1 Sulphide Ores

Ores of this type, in which copper occurs in the form of sulphides in a silicate or carbonate gangue, are the most frequent in nature and constitute the primary mode of formation for copper deposits because of copper's great affinity for sulphur.

The most important minerals in this family are copper sulphides or mixed sulphides of copper and other metals (such as iron, silver, lead, etc.) i.e. chalcopyrite ($CuFeS_2$), chalcocite (Cu_2S), bornite (Cu_2FeS_4) and covellite (CuS).

Antimony and arsenic can also occurs in the sulphide lattice.

These ores are very often accompanied by sulphides of other metals (pyrite, sphalerite, galena, pentlandite, etc.) which may be by-products of the mineral separation processes. Copper can sometimes be considered as a by-product depending upon the relative importance of the various metals.

3.1.1.2 Oxide ores

Alteration through contact with air or surface water in the upper part of the deposit leads to the formation of oxidised copper minerals (oxides, carbonates, silicates or even sulphates and chlorides). Because of dissolution and reprecipitation phenomena the oxidised parts often have copper grades higher than those of the sulphide deposits from which they originate.

Mineralisations in which copper is present as native copper occur either in oxidation parts like the other ores of the oxide type, or in special formations (volcano-sedimentary formations). Their processing properties are intermediate between the two principal families mentioned above. Like the sulphide ore they are easily dissolved in sulphuric acid solutions.

Many deposits contain mineralisations of the sulphide type and of the oxide type and for this reason if they cannot be selectively extracted they will lead to processing problems since, in general, the same separation techniques are not used for the two types.

The composition, grade and density of the most economically important copper bearing minerals are shown in the following table:

	Formula	Copper grade %	Density
Chalcopyrite	$CuFeS_2$	34.6	4.1 - 4.3
Bornite	Cu_5FeS_4	63.3	4.9 - 5.4
Chalcocite	Cu_2S	79.8	5.5 - 5.8
Covellite	CuS	66.4	4.6
Cuprite	Cu_2O	88.8	5.9 - 6.2
Atacamite	$Cu_2(OH)_3Cl$	59.4	3.8
Malachite	$Cu_2(OH)_2CO_3$	57.3	4.0
Azurite	$Cu_3(OH)_2(CO_3)_2$	55.3	3.8 - 3.9
Chrysocolla	$CuSiO_3$, 2 H_2O	36.1	2.0 - 2.2
Enargite	3 Cu_2S, As_2S_5	48.3	4.5
Tetrahedrite	3 Cu_2S, Sb_2S_3	52.1	4.6 - 5.1
Tennantite	Cu_8 As_2S_7	57.0	4.6 - 5.1
Native copper	Cu	100.0	8.9

3.1.2 Types of Processes

Because of their grade or chemical composition, copper ores can only be used directly for metallurgy in special cases and must therefore be concentrated or dissolved.

The choice of the processing method depends upon several mineralogical, chemical and physical parameters such as the type of mineralisation (sulphide and/or oxide), the grade in copper of the carrier minerals and ore, the quantities of possible by-products, the copper content of unwanted minerals (such as arsenic bearing minerals) and the size of the minerals. Previous chemical and mineralogical study is therefore essential for determining the possible methods for treating the ore.

Naturally economic parameters will also play a primary role in the choice of the type of processing to be chosen (investments, operating costs, existence of a protected internal market, etc.).

It is estimated that at the present time more than 85 % of the world's copper is concentrated by physical methods (essentially flotation) and the rest is obtained by leaching.

3.1.2.1 Physical Concentration Processes

Concentration processes by physical means make use of differences in physical or physico-chemical properties existing between the minerals in order to separate them. Those used for copper ores are essentailly gravity methods (especially dense medium separation) and most important of all flotation.

Gravity methods, based upon the differences in density between fragments, can hardly be used for processing particles smaller than any tenth of mm. At this particle size, however, the copper-bearing minerals are rarely liberated and remain associated with gangue particles. It is therefore not possible in this way to prepare a final concentrate composed solely of copper-bearing minerals but only to eliminate relatively coarse gangue particles which will then not be treated further in the process. The enriched product obtained is called a preconcentrate.

Flotation, consisting of the separation of minerals in aqueous suspensions by means of air bubbles and on the basis of hydrophobic or hydrophilic properties of the surfaces of the minerals, is the usual process used for the concentration of sulphide ores since metallic sulphides (as well as native copper) are easily made hydrophobic. This makes it easy to float them from a non-sulphide gangue which is generally hydrophilic. Copper oxide ores are much harder or even impossible to float since their surface properties differentiate them very little from the gangue.

Flotation is a process for separating species which were previously part of a mixture ; this separation can take different forms. In bulk separation, for example, all the components of the mixture are divided into two-groups, in one of which the valuable product or concentrate contains at least two constituents, while in selective flotation (also called differential flotation) separation leads to concentrates containing only one product.

3.1.2.2 Leaching or Hydrometallurgical Processes

Leaching processes extract the metallic elements by dissolving the carrier mineral in a liquid phase ; this operation is followed by reprecipitation in a concentrated form which is often the metallic form.

They sometimes require a preliminary activation phase making the carrier mineral soluble.

These processes are essentially used for oxide ores since the copper present in this form reacts easily with acid solutions to produce copper solutions.

Application to sulphide ores involves previous oxidation and this can be done more or less easily depending upon the sulphide.

3.1.3 Selection of a Process

In general it can be said that a copper sulphide ore will be enriched by flotation except if the grade is so low (0.2-0.4 %) that grinding to the size required for liberation of the copper-bearing minerals is not economically justified. In that case, the ore can be acid leached after coarse crushing (heap leaching) or even sometimes by a method which requires neither crushing nor mining (in situ leaching). The investments and operating costs will then be relatively low but copper recovery will take a long time since reaction kinetics are very slow.

For the polymetallic sulphide concentrates which are most of the time difficult to separate by flotation,hydrometallurgical processes, still in development seem to be attractive

For the treatment of an oxide ore the most obvious method is to produce an acid solution of copper except when there is a carbonate gangue which would lead to prohibitive reagent consumption. In this case the appropriate methods would be either an ammonia leaching process, or flotation after sulphidization, or even a segregation process.

Processing mixed ores in which sulphides and oxides are present at the same time should be considered as special cases in terms of reactions to flotation and leaching of the various copper-bearing elements and gangue minerals.

In any case no single process can be theoretically indicated only from a mineralogical examination of samples : all development of industrial processes must go through a stage of laboratory tests and then, in general, through a stage of pilot plant tests.

3.2 Mineral Processing

3.2.1 Physical Processing Methods

Essentially there are two physical processes which can be applied to copper ores. These are separation by gravity and separation by flotation.

Both of them will be studied in this chapter, with particular emphasis on the most common, the flotation process.

3.2.1.1 Gravity Separation

Gravity separation is based upon the difference in density between :

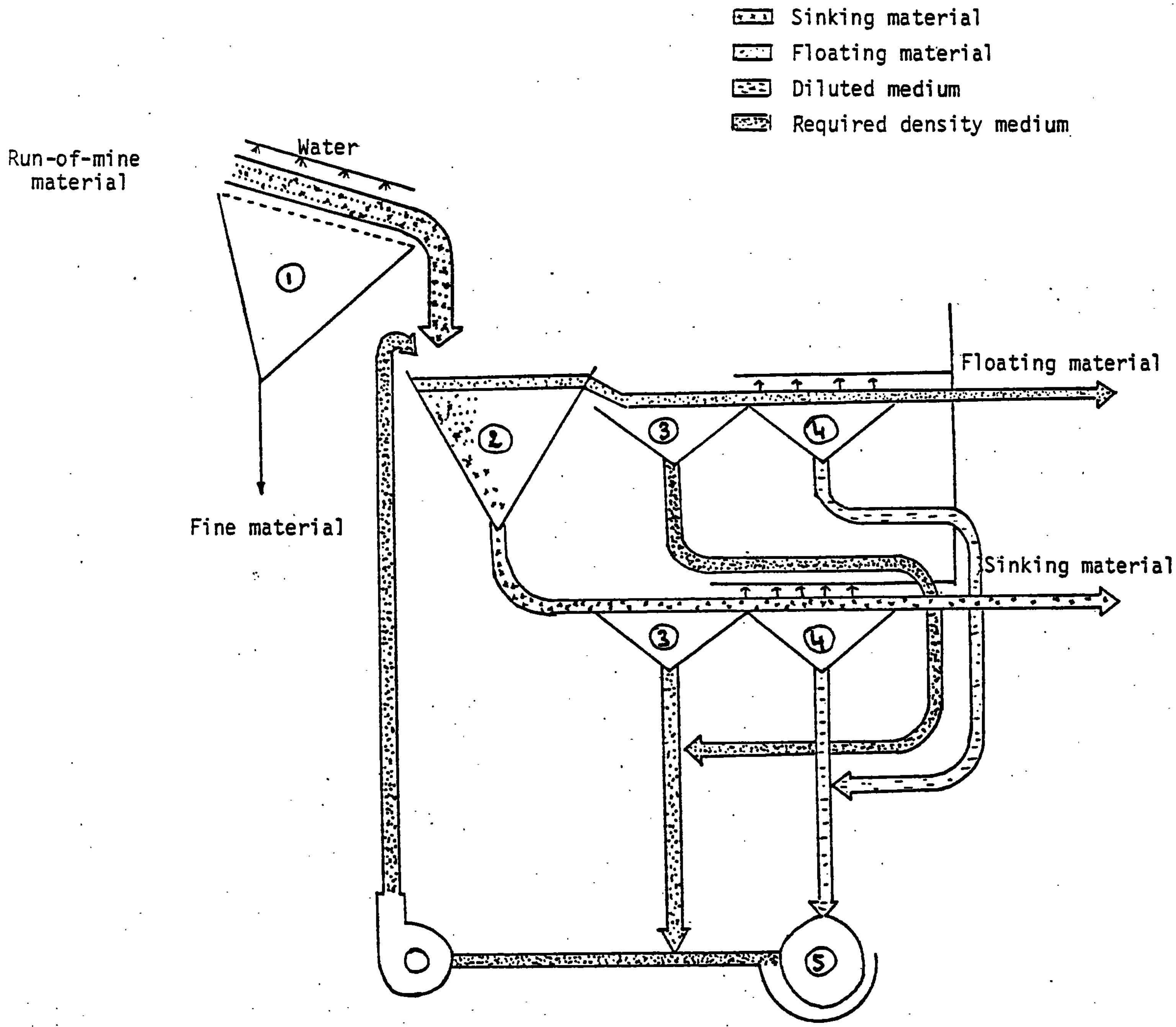
- Firstly, the copper sulphides and oxides (the density of which varies approximately from 3.8 to 6.0, except for chrysocolla which has a density of only about 2) ;

- Secondly, the gangue minerals (the density of which is near 2.7 for silicates and carbonates).

Dense medium separation seems to be the best gravity process for processing copper ores since it makes possible a precise division for a wide range of particle sizes at a relatively low cost.

The process consists of placing the ore particles to be separated into a bath having a density intermediate between that of the waste particles and that of the valuable particles. The dense medium bath is generally obtained from a suspension of a magnetic product (generally magnetite and/or ferrosilicon).

The denser particles fall to the bottom of the bath while the lighter ones float. Each fraction is collected and then cleaned and drained so as to recover the particles of the medium which will be reused after magnetic concentration.



1. Wet screening
2. Cone separator
3. Draining screen
4. Cleaning screen
5. Medium recovery

FLOWSHEET OF DENSE MEDIUM SEPARATION PROCESS

(Fig 11)

This separation can also be carried out by using centrifugal forces (dense medium cycloning) and can then be applied to particles which can be as small as 0.5 to 1 millimeter. There is no maximum particle size limit for the equipment but because of the mineralogy and texture of the copper ores particles larger than 1 or 2 decimeters are rarely processed.

This process cannot be used for finely and regularly disseminated mineralisations but can be applied when the separation between the gangue and the copper-bearing ore is clear cut. It is then generally possible to eliminate a relatively large fraction of waste (between 30 % and 80 % in weight of ore as an indication of the order of magnitude) and therefore its grinding and flotation is not necessary.

This operation also facilitates waste disposal.

Although from a theoretical point of view this technique is applicable to oxide ores as well as sulphide ores, hardly any industrial applications for the former case are known, no doubt because in this case the mineralisation is generally disseminated.

Dense medium separation consumes about 1 m^3 of water per tonne of ore processed (up to 95 % of this water can be recycled) and a few hundred grammes of magnetic medium material per tonne of ore processed.

3.2.1.2 Flotation Processes

A requirement to obtain a good flotation product is that the maximum of fragments in the mixture of particles to be separated should contain only a single type of mineral, either one or several copper-bearing minerals (or possibly by-products) or one or several gangue minerals.

In addition, the particles should be of a size which is compatible with flotation (between several microns and several tens or a few hundreds of microns).

Because of these two factors ore fragments from the mine (generally of several decimeters in size) are reduced by means of crushing and grinding operations.

The conventional flow sheet for the various stages of the flotation process is shown in the figure below based upon Tyrone (New Mexico). [1]

3.2.1.2.1 Crushing and Grinding

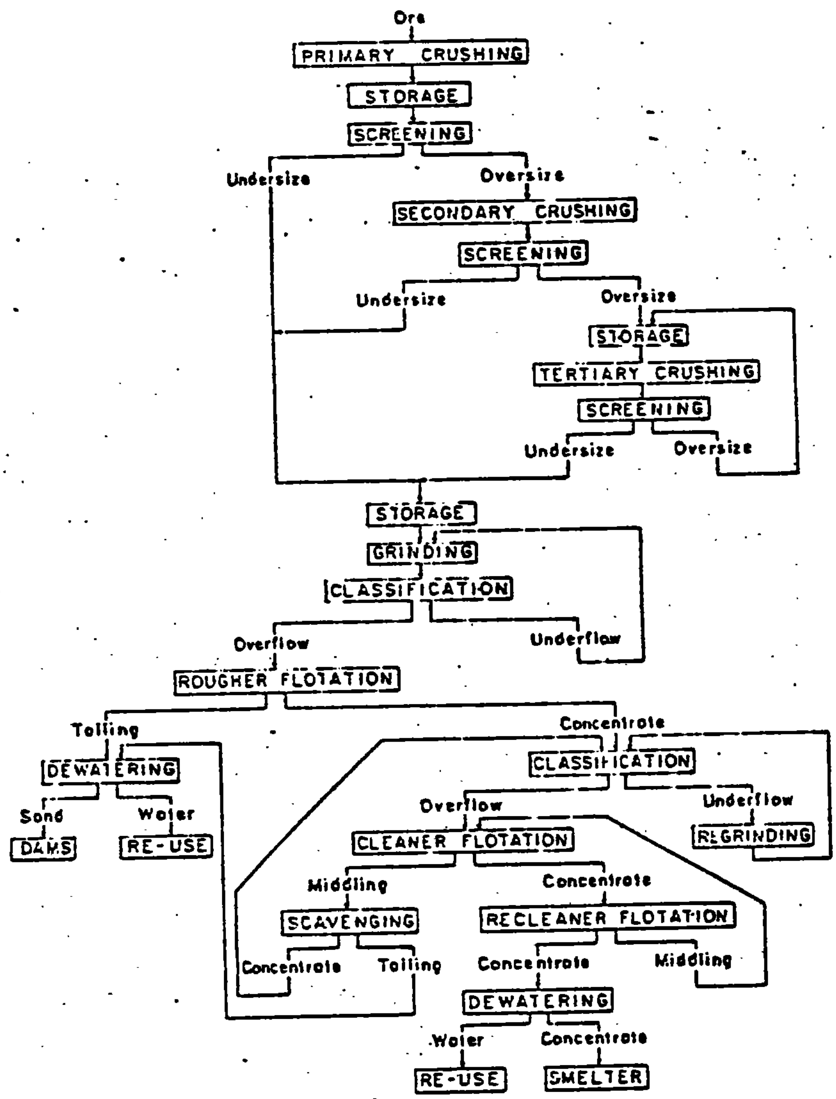
Crushing is an operation consisting of reducing the sizes of ore blocks from the mine to centimeter size. It is generally performed dry in two or three stages.

The essential equipment for the first stage is either a jaw crusher (a mechanism for fragmenting the material by compression between two jaws i.e. a fixed vertical jaw and a suspended mobile jaw with an alternating movement) or, in large plants, a gyratory crusher in which fragmentation results from compression between a fixed truncated shell and a truncated conical head mounted on a spindle.

Secondary and tertiary crushing (the latter is often eliminated in small plants) are generally carried out by gyratory crushing.

Fig. 12

TYRONE CONCENTRATOR FLOWSHEET



from F.J. WITTHAVER in Flotation [1]

Between the various crushing stages there is particle classification by screening and this makes it possible to retreat particles which have not been reduced to the required size. The transfer of the ore from a machine to another is carried out either by gravity or by conveyor belts.

The size of the primary crushers is almost always fixed by the size of the run of mine ore and not by the hourly yield of the flotation equipment. The capacity of crushing equipment makes it possible to operate them only for one or two shifts per day while the flotation equipment operates continuously. This organisation also makes it possible to spend more time to the maintenance of the crushing equipment which is subject to severe mechanical stresses ; buffer storage is required for the continuous feeding of the flotation system. Storing also contributes to the homogenisation of the ore.

Grinding takes place in wet medium in cylindrical or cylindroconical tubes revolving around a horizontal axis and filled with grinding bodies which grind the ore during their rising and falling motion within the mills. Ore and water are added at one end, and the ground pulp comes out at the other end.

In medium sized and large plants several lines of grinding mills operate simultaneously : grinding mills operate in series of two, the first is generally a mill in which grinding elements are rods, while the second is a ball mill which is better adapted for the production of fine particle sizes (sizes such that 80 % of the ore pass through a 75 micron mesh screen are very frequent).

One pass through the mill is not enough to produce the desired fineness so that it is necessary to classify the pulp processed by the mills and to recycle the insufficiently ground particles. This classification is carried out essentially by hydraulic equipment using the centrifuge principle. In relation to the screw and rake classifiers used previously the cyclones have the advantage that the necessary investment is small, little place is required and the equilibrium of operational conditions is quickly reached.

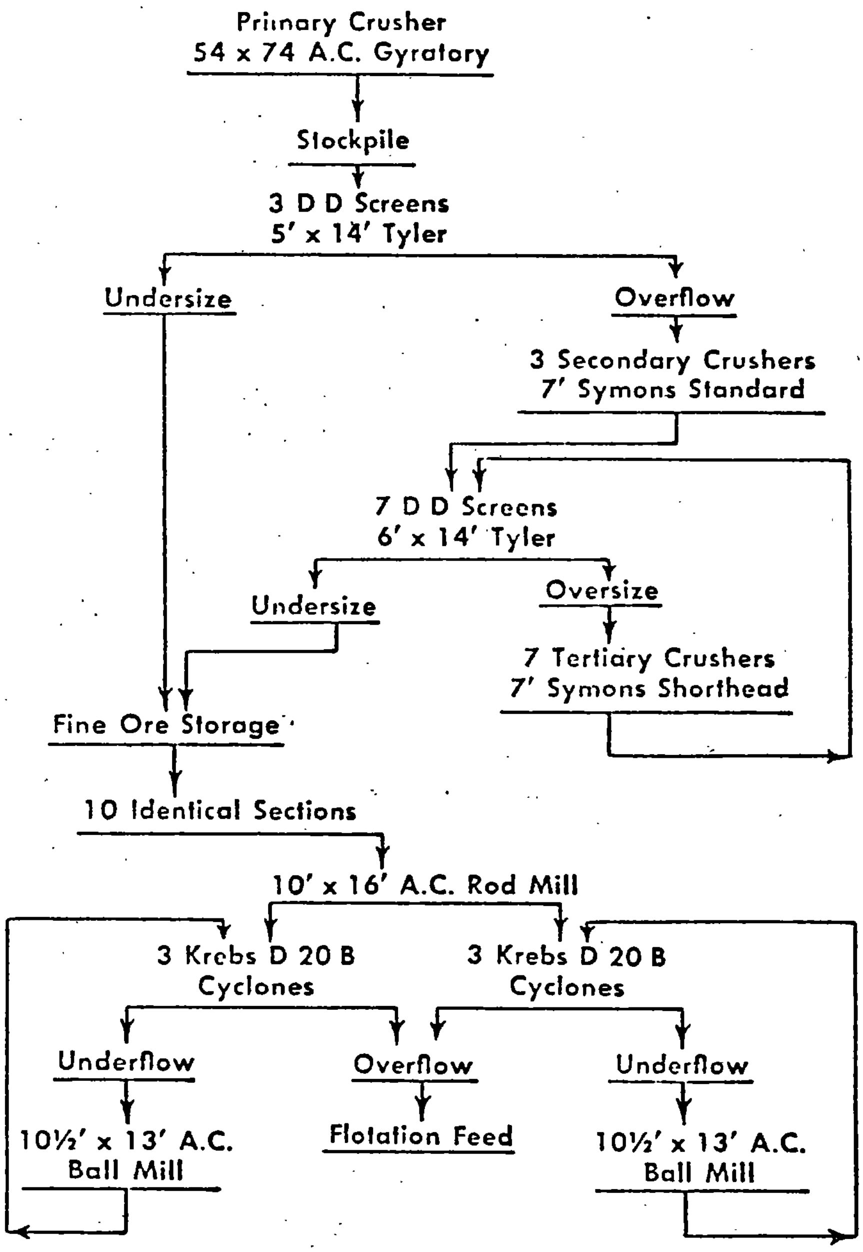
Crushing and grinding are expensive items in an ore dressing plants especially as concerns energy. It can be estimated that between 40 and 60 % of the operating expenses of a flotation plant are used in crushing and grinding. It is therefore desirable to limit the size reduction stage as far as possible. This explains why a grinding stage often alternates with a flotation stage producing a mixed fraction, which is then reground, and a waste fraction which is disposed of. The grinding expenses are lower compared to fine grinding of the complete ore and this explains one of the attractive features of preconcentration by gravity or flotation. In other hand, because of the high cost of grinding, the most economical grinding size is often bigger than the actual physical liberation size.

Amongst the recent improvements in crushing and grinding, one should note the development of automated plants allowing reductions in personnel, improved utilisation of existing plants and improved regularity in flotation feed. Autogenous or semi-autogenous grinding has also been developed making it possible to replace crushing and sometimes primary grinding by processing in a single unit (the grinding bodies consist in part at least of the ore itself). For this process it is necessary that the ore be dense and there should be a high degree of continuity in time for its physical characteristics (particle size, hardness).

Below will be found flow sheets for two crushing systems used at Pima. [2] One has the conventional crushers and grinding mills while the other has only an autogenous mill followed by a ball mill.

Fig. 13

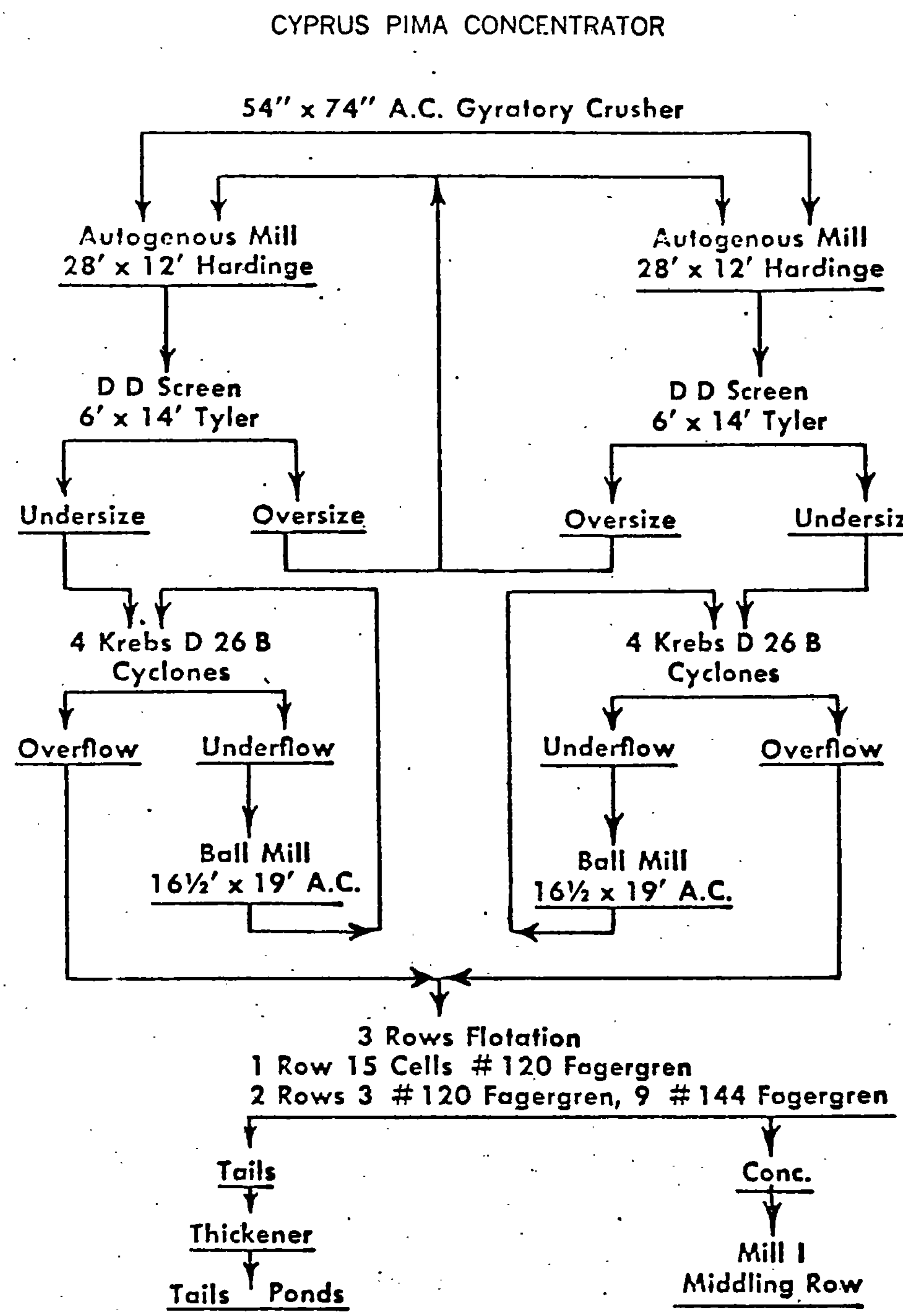
CYPRUS PIMA CONCENTRATOR



Mill I Crushing and Grinding Flowsheet [2]

from T. RAMSEY - Flotation 1976

Fig. 14



Mill II Flowsheet [2]

from T. RAMSEY - Flotation 1976

3.2.1.2.2 <u>The Flotation Process</u>

Flotation takes place by the addition of reagents to the ground pulp (the quantities used vary from about ten grammes to a few kg per tonne of ore) :

- <u>Collecting agents</u> which give hydrophobic properties to the minerals (especially products of the xanthate and dithiophosphate families) ;

- <u>Frothing agents</u> such as pine oil for making a froth which will be sufficiently stable to allow recovery of particles fixed at the bubble-liquid interface ;

- <u>Depressing agents</u> which are intended to keep the collecting agent from fixing itself on the surface of some minerals and <u>activating agents</u> which have the opposite effect. Sphalerite is activated by copper salts but depressed by SO_2, cyanide, sulphites. In small doses sodium sulphide can activate some oxides by giving them the properties of sulphides, while in large doses it becomes a depressant.

PH has an important role as a flotation regulator.

These reagents are added at several points :

- at the grinding stage ;
- after grinding in conditionning tanks (large tanks with agitation) ;
- during flotation.

Flotation takes place in large tanks (the volume can vary from several hundred liters to approximately ten m^3) equipped with agitators for aerating the pulp and dispersing the solids. These tanks are grouped so as to form circuits which are adapted to each type of ore (flotation plants can have capacities varying from several hundred tonnes to one hundred thousand tonnes per day).

" The flotation circuit for a single product is generally organised in several stages. The product is first preconcentrated in rougher cells and then enriched in cleaner cells. It is often necessary to supplement the rougher cells with a series of recleaning cells whose object is to exhaust the ore with valuable types of minerals. In this case the concentrate obtained is fed back into the rougher cells (Figure 15).

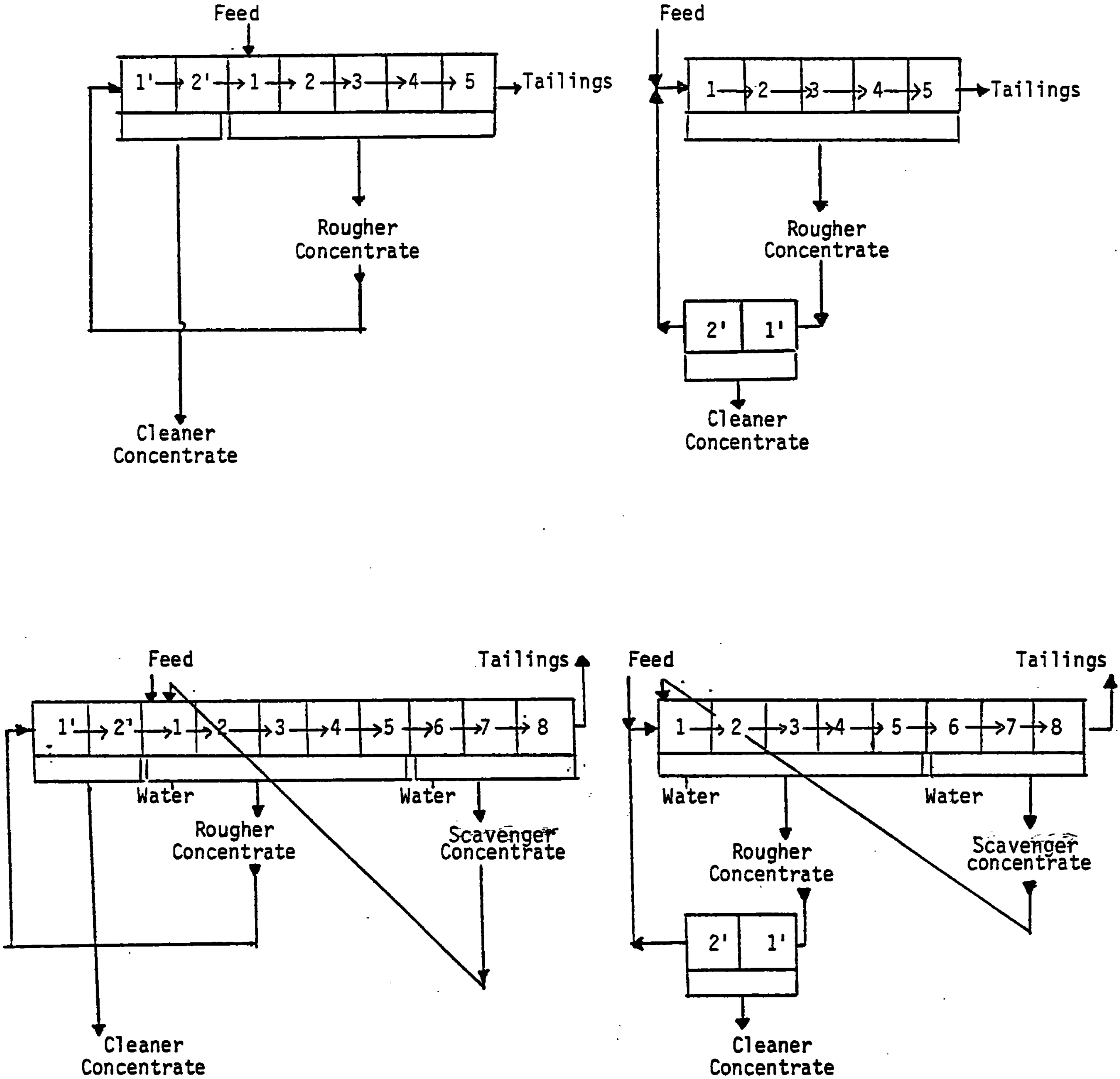
These combinations of circuits are necessary since the products floated in the rougher cells generally contain gangue particles which have been carried along mechanically. After passing through the rougher the waste is generally too rich and therefore require further treatment before being completely exhausted". [3]

Several recleanings are often necessary.

Selective flotation for several products can be carried out either by selectively floating the various mineral species one after the other or by floating a bulk concentrate which is then reprocessed in order to separate the various valuable types of minerals.

Improvements in flotation processes in recent years have principaly concerned increasing the sizes of the cells and automation of the circuits (pulp particle size and density, analysis of products by X-ray fluorescence, pulp level in cells).

Figure n° 15



FLOTATION CIRCUIT WITH ROUGHER CELLS (1,2,3,4,5)

THICKENER CELLS((6,7,8) AND CLEANER CELLS (1', 2') [3]

Sulphide Ores

In the simplest case in which copper sulphide is the only product to be recovered the following reagent consumptions and processing results can be expected. [4]

. Collector (Xanthate or Aerofloat)	25-300 g/t
. Frother	25-250 g/t
. pH regulator (lime)	1,000-4,000 g/t
. Flotation time	8-15 minutes
. pH of circuit	9-12
. Ore grade	0.5-5 % Cu
. Concentrate grade	20-50 % Cu
. Tailings grade	0.06-0.25 % Cu
. Recovery	90-95 %

It should be noted that the concentrate grade cannot exceed the grade of the copper-bearing mineral (see table at the beginning of 3.1 chapter). The most important mineral chalcopyrite, has a grade of 34.6 % Cu while average commercial concentrates have a grade of about 25 % Cu.

Below is the flowsheet for the Pima (Arizona) ore dressing unit which floats an ore containing 1 % Cu (in the form of chalcopyrite and chacosite) and produces a concentrate with a grade of 25 % Cu with a recovery of 91 %. [5]

If the ore is slightly oxidised the addition of sodium sulphide to the pulp leads to the formation of a thin layer of copper sulphide over the oxides which in this way can be floated with good recovery.

Copper flotation becomes more difficult when there are other valuable minerals which must be separated (molybdenite, pentlandite, sphalerite, galena, cobalt sulphide, etc.).

We will now consider some cases involving typical ores which have economic importance.

Porphyry Copper Ore

The term porphyry copper originally had a geological

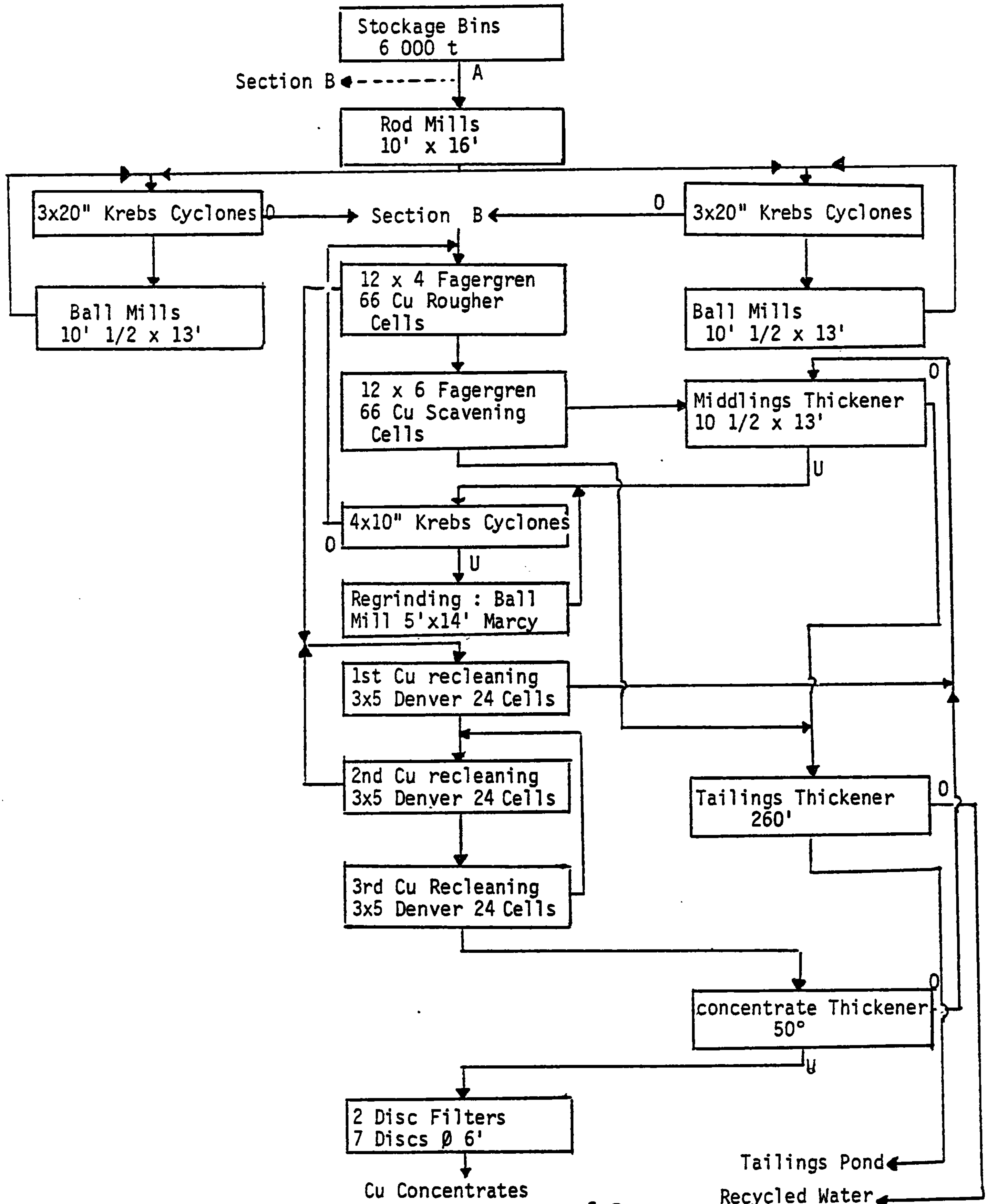


Fig. 16 Pima Mining Co. Dressing Flowsheet (1964) [3]

meaning and was used for a copper mineralisation disseminated in veinlets through granitic masses (monzonite). Today economic and technical considerations (mining and ore processing) also have a part in the definition of the term, and a prophyry copper deposit defines a low grade deposit which is worked on a large scale and in which copper-bearing species (chalcopyrite, chalcocite, bornite, etc.) are found in the form of veinlets disseminated in rock which is generally sericitized. In this type of deposit with a great vertical extension, there is often an iron cap and secondary supergene enrichment. The copper-bearing minerals (Cu grades varying from 0.4 to 1.8 %) are often accompanied by molybdenite (0.01-0.05 % MoS_2) and pyrite (1-5 % FeS_2). There are sometimes small quantities of gold and silver.

About half the world's copper production comes from deposits of this type which are especially common in the US (Arizona, Utah, New Mexico), South America (Chile, Peru), the USSR (Kazakhstan, Uzbekistan, Armenia, Siberia), Canada (British Columbia, Quebec), Bulgaria, Yugoslavia and the Philippines.

The processing objectives for this type of ore are to obtain a copper concentrate free of molybdenum since this metal cannot be recovered by copper pyrometallurgy (a typical concentrate contains 24-40 % Cu, 20-25 % iron, 5-8 % silica and 0.1-0.2 % MoS_2) and also a separate molybdenite concentrate (90-96 % MoS_2 and less than 0.5 % Cu). A pyrite concentrate may also be floated.

If the ore is only very slightly oxidised (less than 5-8 % copper present in oxide form), the copper and molybdenum sulphides are floated together by xanthates in an alkali medium (pH between 8.5 and 11.5) which depresses the pyrite (possibly with the help of cyanide). If there is more oxidation (between 8 and 25 %), flotation takes place in an acid medium (pH approximately 4) which allows surface dissolution of the copper oxides since xanthates decompose by hydrolysis in an acid medium, they are replaced by dixanthogens or dithiophosphoric acids. A sulphidizing agent may be used. Since flotation in an acid medium does not depress the pyrite another separation must be made in an alkali medium.

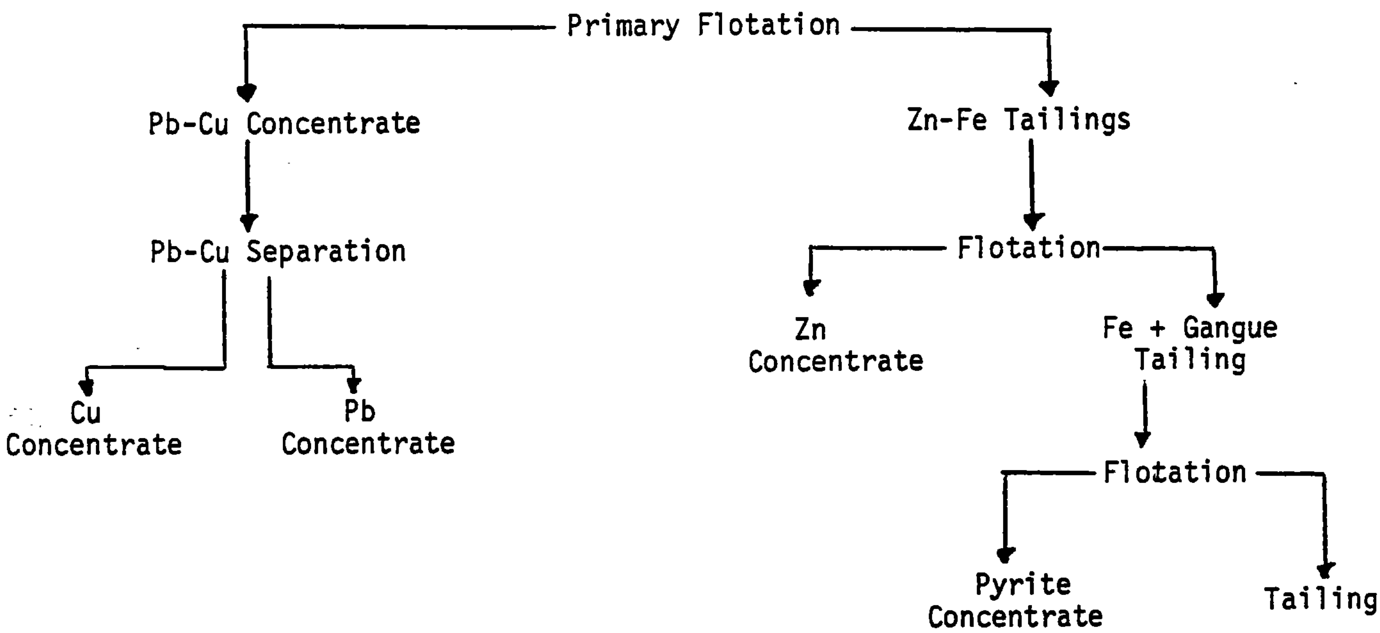
After several cleaning and regrinding stages, and after elimination of the reagents by thickening and heating of the concentrate pulp, copper and molybdenum sulphides are generally separated by depressing copper and selectively floating molybdenum (Morenci process with addition of cyanide, or Kennecott process using Nokes reagent with polysulphides or the San Manuel process using oxidizing agents. In Soviet Union, a fourth process consisting of depression by sodium sulphide is used extensively). It is also possible to depress molybdenite with starch and float the copper sulphides.

Complex sulphides ore (sphalerite, pyrite, galena, chalcopyrite)

This is the type of ore which is most generally found in the European Community (Sardinia, Tuscany, the Harz, etc.). It is generally found in large masses (and is thus very often called massive sulphide ore) and the various sulphides are very closely imbricated. The grades are often high but the deposits are often relatively small.

Flotation of a sphalerite, galena, chalcopyrite ore is relatively frequent but is difficult to carry out, since the valuable minerals are often closely associated, the flotation reaction properties of the zinc and copper sulphides are not very differerent and the presence of copper ions in the pulp favors sphalerite flotation.

The most common conventional method is semi-bulk flotation with the following flowsheet :



Lead and copper can be separated by several methods i.e. depression of copper by addition of sodium sulphide or cyanide, or depression of galena by heating the pulp to about 60° C or by addition of SO_2 and dichromate.

Sometimes, for example, at Rammelsberg, the various sulphides are floated one after the other (chalcopyrite, then galena, sphalerite and pyrite) by means of slowly increasing additions of xanthates.

In the following pages will be found the flowsheets for the New Brunswick plant (50 % Cu recovery in a concentrate having a grade of 22.6 % Cu from an ore assaying 0.33 % Cu) and the Idaraho unit (a 30 % concentrate is obtained from a 1 % ore with 77 % recovery). [5]

The metallurgical results in the case of a complex ore are naturally much worse than for simple ores and attempts are often made to market mixed concentrates, which are unfortunately harder to sell. Many hydrometallurgical methods are being studied for these mixed concentrates but at the present time their effectiveness has not been confirmed (see section on hydrometallurgical processing).

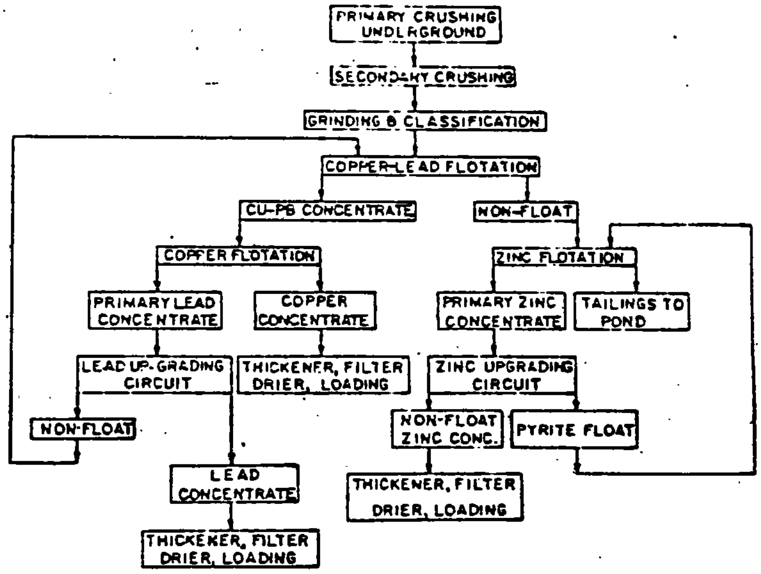
Oxide Ores

Oxide ores are much harder to float. Two methods do, however, exist [6] :

- Sulfidizing with Na_2S and NaHS leading to the surface formation of a sulphide layer making the mineral floatable with conventional sulphide ore collectors (used at Tsumeb in South Africa and Zaire) ;

- Flotation by fatty acids, the characteristic oxide collectors, which are unfortunately not very specific for carbonate gangues (used at Nchanga in Zambia and Kolwezi in Zaire).

The latter method seems to be losing ground while the former is increasingly used. In both cases recoveries are not very good (in general 70-85 %). The



Summary Flowsheet

Fig. 17 Brunswick n° 12 Mine [5]
from G. Nevmann et J. Schnarr

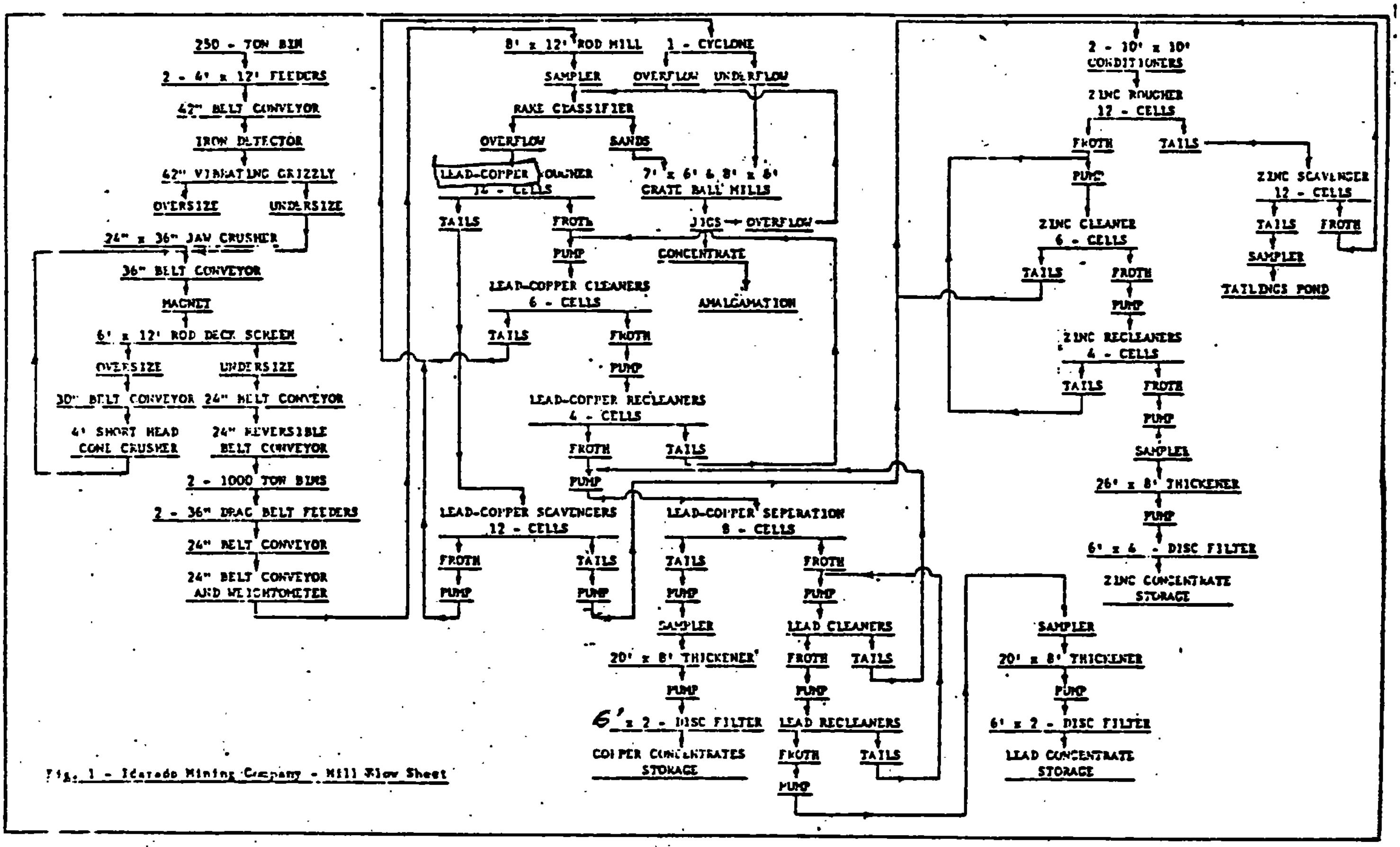


Fig. 18 Milling Practice at Idaraho Mining Company [6]
from R.C. Stevens et G.G. Granger

concentrates are hard to clean, and their grade is acceptable only because of the high copper content in the floated minerals (malachite).

3.2.1.2.3. Thickening and Filtration

Concentrate obtained by flotation contain approximately one part solid for two parts water (by weight) which must be eliminated before being transported to a metallurgical plant.

Water is eliminated by means of two types of operations used in series, thickening and filtration.

Thickeners are cylindrical steel or concrete tanks in which concentrates settle by gravity and produce a pulp containing 50-70 % solids, which slowly revolving rakes bring to the center of the thickener bottom, from which it is pumped (by diaphragm or centrifugal pump depending upon the size of the plant) to disc or drum vacuum filters. There bring the water content of flotation concentrates to about 10 % and the concentrate can then be shipped to metallurgical plants.

As an order of magnitude for the necessary surfaces for thickening and filtration the figures of 0.2 m^2 per tonne treated in 24 hours and 5 m^2 per tonne, treated in 24 hours can be given respectively as indications. It is clear that these surfaces depend upon the ore and they must be experimentally determined in each case.

Thickening and floculating agents may be used.

These equipment can be automated and the operation of filters can be controled by the density of the pulp coming from thickeners.

3.2.1.2.4 Tailings disposal and Water Recycling

Tailings produced by a copper ore flotation unit represent a very large fraction of the run-of-mine ore and it is sometimes difficult to dispose of them. There is often a choice between two methods i.e. construction of a tailings pond behind an artificial dam which can be made out of part of the waste, or use as fill for mine excavations (especially for underground mines). The possibiliites depend largely upon the particle size of the waste (excessively fine products cannot be used for fill or for building a dam).

The waste contains between 4 and 5 tonnes of water per tonne of solid and in most cases for simple flotation it is possible to recycle 75 % of this water in the flotation unit which consumes 4 m^3 of water per tonne of ore.

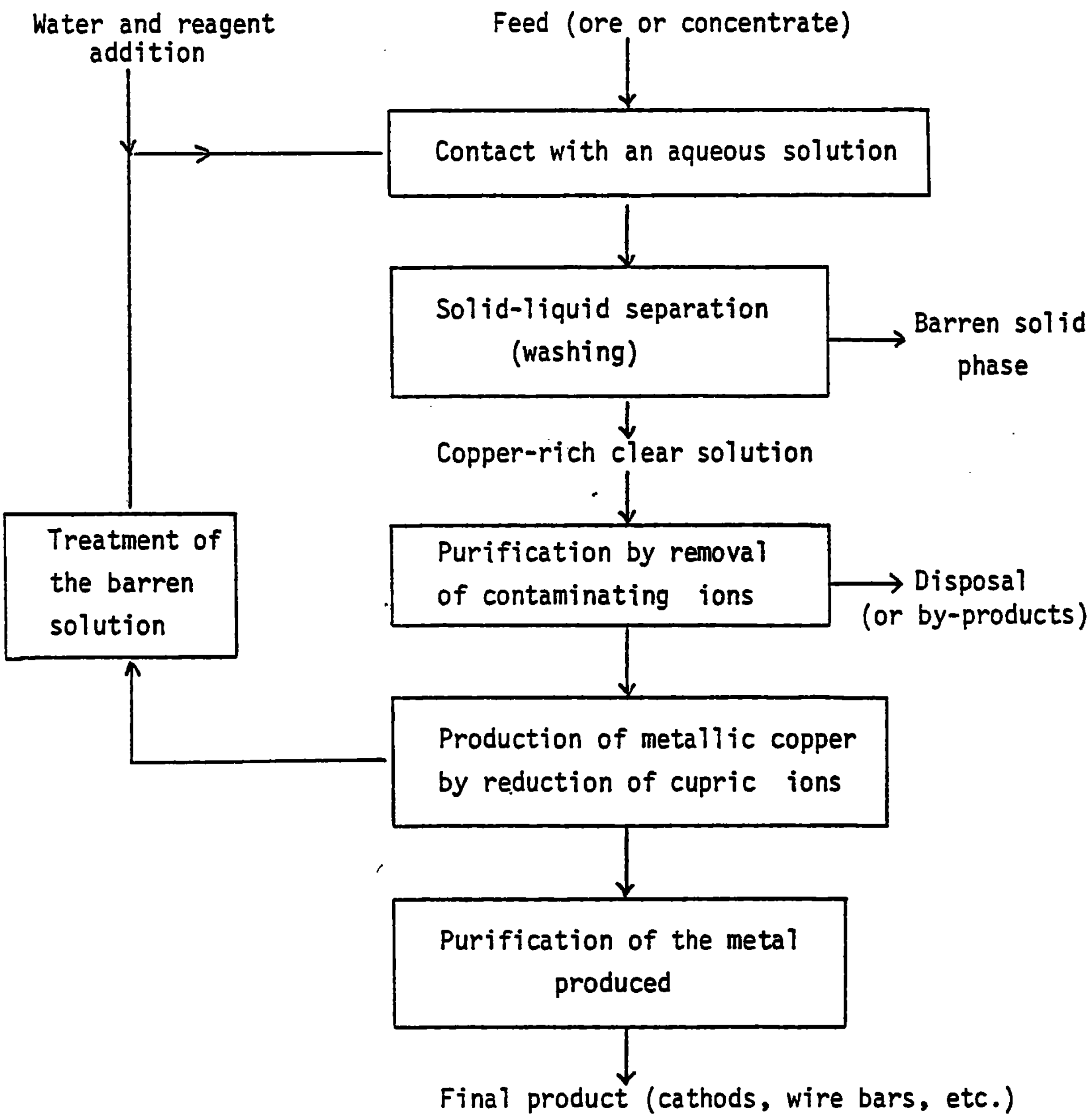
Water recovery can sometimes be increased by thickening the waste.

It is difficult to recycle water in complex sulphide flotation since there is then a risk of reintroducing water containing depressing reagents at a point of the circuit where flotation is required. Recycling allows savings in water and reagents, and is compulsory when environmentally harmful products are used. When abundant water is available it may, however, not be applied.

3.2.2 Hydrometallurgy in the production of copper

Traditionally, the production of metallic copper from its ores or concentrates through a dissolution of the copper-bearing minerals into an aqueous solution, is the field of process application to oxide ores and concentrates, or low grade sulphide ores.

The various steps involved in a hydrometallurgical process can be summarized in the following block diagram :



For the past ten years, a very strong emphasis has been given to the processing of sulphide concentrates by hydrometallurgical methods. The main reasons which can be given for this interest can be found in the environmental regulations which have to be progressively met by smelters using conventional pyrometallurgical processes (see the Pyrometallurgy section). A fairly obvious way to prevent sulphur dioxyde (SO_2) release into the atmosphere, which is the major polluant for these processes, is to use a method in which there would be no gaseous effluent, and hydrometallurgical processes have this great advantage. Research in this field has developed along various routes, and it is felt that it is too early yet to set rules and choices for defining technical, economical and ecological advantages. This is the reason why a very detailed presentation is given of the various processes on which research and development programs are currently active.

However, progress in the development of new pyrometallurgical processes - interestingly coming from countries outside the U.S.A., the major world producer - indicate that it is possible to produce copper by pyrometallurgy at acceptable SO_2 level in effluent gases, whilst at the same time reducing the overall energy consumption in the production of copper. The energy crisis seems to have considerably altered the impetus given to hydrometallurgical processing of copper from sulphide concentrates. Only one plant (the Arbiter process installed in Montana by the Anaconda Company), seems to be operating successfully presently.

Thus, for quite a number of years to come, it is felt that hydrometallurgy will be maintained in its minor but still very important role, contributing an estimated 15 % of the world production of copper, through the processing of low grade oxide and sulphide ores, and oxide concentrates.

The presentation will therefore be made in three points :

1. Hydrometallurgy of sulphide copper concentrates.
2. Hydrometallurgy of low grade sulphide ores.
3. Hydrometallurgy of oxide ores and concentrates, to which will be attached a fourth point, which is included here for reason of convenience, or
4. Mixed processes.

3.2.2.1 Hydrometallurgy of sulphide concentrates

The conventional pyrometallurgical processes for the recovery of copper from sulphide concentrates obtained by flotation lead to the release of large quantities of sulphur dioxide into the atmosphere, and therefore, to certain limitations on the development of copper concentrate processing, especially in the U.S.A., but also in most developed countries having high population densities. This has led metallurgical companies, as well as research centers, to undertake many studies in the field of hydrometallurgy of sulphide concentrates, which is generally considered to be less polluting.

While environmental problems are the most obvious reason for the increased interest which has been shown for about ten years in leaching processes, this is far from being the only reason :

- Increased copper needs and the progressive disappearance of conventional deposits are leading, and will continue to lead, to an increased exploitation of complex sulphide ores (for example Cu-Zn), for which differential flotation rarely gives satisfactory concentrates. Hydrometallurgical processes are capable of treating complex or low grade concentrates (for example as low as 10 % copper), which, in most cases, leads to considerable savings at the concentration level (lower investments for plants and better recovery);

- Most of the reagents necessary for leaching are produced in large quantities and often at moderate prices (oxygen, ammonia, chlorine, hydrochloric acid, etc.). Sulphuric acid in particular is relatively cheap;

- The development of selective liquid-liquid extraction reagents for copper makes it possible to produce directly marketable copper from relatively impure and dilute solutions;

- The appearance on the market of special materials at moderate prices would provide solutions to engineering problems (abrasion, corrosion);

- Increasing transport costs and the tendancy to treat concentrates at mine site, make the conventional smelters, which are economical only for very large throughputs, less competitive in relation to hydrometallurgical plants for low tonnages (for example 5 to 30 kt of copper per year).

At the same time, however, pyrometallurgical techniques have developed very rapidly :

- Specific anti-pollution devices have been added to existing plants for the reduction of SO_2 emissions (Cat-Ox, U.S.B.M., Wellman-Power Gas, I.F.P. processes, etc.);

- New processes have been developed in which the smelting and conversion stages are integrated into a continuous operation (or continuous sequence of operations) leading to more concentrated SO_2 emissions, thus facilitating sulphuric acid production from waste gases, for example, Mitsubishi, Noranda, Worcra, Outokumpu processes.

Hydrometallurgical processes for sulphide concentrates can be grouped according to the nature of the leaching medium, the most important of which are chloride, ammonia, sulphuric, and to a lesser extent, nitric and cyanide.

3.2.2.1.1 Leaching Processes in Chloride Medium

Leaching in chloride medium has several interesting aspects :

- High solubility of most metal chlorides ;
- Use of controled oxidation methods in order to oxidise sulphide to elemental sulfur only and in most cases, to leave iron undissolved ;
- It is easier -compared to sulphate medium- to achieve a separation of various metals and there is therefore a greater potential for treating complex sulphide concentrates (for example, Cu-Zn, Cu-Ni-Co).

There are however considerable drawbacks :

- Corrosion problems (especially with ferric chloride) involving the use of expensive materials;
- Dissolution of As and Sb.
- The high iron content of the leaching solutions makes it difficult to obtain pure copper.

The various processes briefly described in Table 10 include a leaching stage (with $FeCl_3$ or $CuCl_2$ + NaCl), a copper recovery stage and a leaching solution regeneration phase. It should be noted that all these processes except U.S.B.M. lead directly to marketable copper. The low quality of the copper obtained with the U.S.B.M. process is due to the presence of large quantities of ferric iron during electrowinning.

All the processes produce elemental sulphur which in view of the present price of this substance does not appear to be an economic by-product. The only reasons for recovering it in this state are the energy savings and the fact

TABLE N° 10

PRINCIPAL PROCESSES FOR TREATMENT OF CONCENTRATES

Processes	Leaching Medium	Temperature	Copper Recovery and Type of Metal	Regeneration of leaching medium	Final sulphur compound	Final iron compound	Possibilities for Recovery of other substances	Energy consumption kWh/kg of Cu	State of Advancement	Ref.
Cymet	$FeCl_3$	80° C	Very pure copper	Electrolysis	Agglomeration in autoclave at 130° C and hot filtration	Fe	Precious Metals, Pb, Zn, Ni, Co	2	Pilot 25 t concentrate/day	[7]
USBM	$FeCl_3$	105° C	Electro-deposition in (300 A/m^2) cell diaphragm of copper powder to be refined	Aeration at 180°C	Extraction with Ammonium sulphide which can be recycled		Idem	1.5 + medium regeneration + refining	Labo.	[8]
NIM	$FeCl_3$		Liquid-liquid extraction - elution-electrowinning	non precised	S recovery possible	Fe	Idem	?	Pilot	[9]
Minemet Recherche	$CuCl_2+NaCl$	?	Idem	Aeration	S left in residue	FeOOH	Idem	Limited	Pilot	[10]
Duval	$CuCl_2+NaCl$	105° C	Direct electrowinning	Aeration at 140° C	Idem	Insoluble oxides and sulphates	Idem	?	Industrial 36,000 t Cu/year	[11]

that it may be less polluting in this form than in some other.

3.2.2.1.2 Leaching Processes in Ammonia Medium

These processes are derived from the process which has been used for about twenty years for nickel concentrates at the Fort Saskatchewan plant (Canada) of Sherritt-Gordon Mines.

Leaching takes place in an ammonia medium in the presence of oxygen with sulphur being oxidized into sulphate and copper going into solution in the form of a complex copper tetrammine ion.

In the case of chalcopyrite, the rate of dissolution is limited by the stage during which oxygen diffuses through the layer of reaction products (ferric hydroxides), which form on the surface of the mineral. In this case, an increase in the rate of dissolution can be obtained by considerably increasing partial oxygen pressure (80° C ; 10 kg/cm^2 - Sherritt-Gordon process) or by eliminating the oxide layer by vigorous agitation (60 to 90° C ; 1.4 kg/cm^2 - Arbiter process [12]).

The Anaconda Co. uses the Arbiter process in a plant in Montana producing 36 kt of copper per year where the problems involved in agitation, copper recovery (liquid-liquid extraction) and ammonia recycling seem to have been solved.

The advantages of the Arbiter process at the economic as well as technical level will no doubt give it a leading place amongst methods for future use as concerns the recovery of non ferrous metals from complex sulphide concentrates.

The main disadvantage of the process is the need either to recycle ammonia (with the production of 1.5 t of gypsum per t of copper) or to sell ammonium sulphate. Profitability depends to a critical extent upon oxygen, lime and ammonia costs.

3.2.2.1.3 Leaching Processes in Sulphate Medium

Sulphuric acid combined with various oxidizing agents is being given a great deal of attention in research on hydrometallurgical techniques for copper sulphide concentrate processing. The principal reasons for this are the low cost of the acid, fewer corrosion problems and the possibility of acid regeneration during copper electrowinning.

The leaching systems can be divided into three main groups :

- Sulphatation (by roasting or concentrated H_2SO_4) and water leaching ;
- H_2SO_4 leaching under oxygen pressure ;
- H_2SO_4 leaching in the presence of ferric ions (slow reactions rarely considered for concentrate processing).

To these three main groups could be added the processes using oxidizing agents such as manganese dioxide or potassium dichromate. These methods have not yet however shown sufficient advantages to justify a special study.

Sulphatation and Leaching

Practically total sulphatation of concentrate can be obtained through concentrated sulphuric acid attack at about 240° C. The sulphur obtained in the elemental state is eliminated by distillation ; the copper and iron sulphates are dissolved by water leaching. Copper is recovered by reduction (SO_2), precipitation of copper cyanide (HCN) and reduction of the latter (H_2 at 300° C) and the reagents are regenerated during the various stages.

The main disadvantages of the process are the production of iron sulphate as an effluent and the blocking of lines and valves by sulphur vapor.

This process (ANATRAED) was originally presented as very competitive but attempts to develop it were unsuccessful (1970 : Anaconda pilot for 6 tpd of concentrates) [13].

Sulphatation can also be brought about by roasting. The choice of temperature is a critical factor, for it is essential to work at a temperature which is high enough to transform iron sulphides into oxides but not high enough to lead to the formation of insoluble copper ferrite. The optimum temperature is around 680-700° C.

Many variants of the process involving sulphatation roasting have been proposed which differ basically as to the types of processing of the solutions (electrowinning, cementation, precipitation, liquid-liquid extraction).

The limits and disadvantages of sulphating-leaching roasting are well known :

- Copper recovery is generally lower than in conventional metallurgical processes ;

- Precious metals are often lost in the residues.

- It is necessary to remove large volumes of solution containing copper (spent electrolyte) and electrolysis produces more sulphuric acid than is necessary for leaching (which could sometimes be an advantage) ;

- Although the roasting gases have a high enough SO_2 concentration to allow autothermal conversion into sulphuric acid, the concentration is quite close to the lower limit of possibility for this operation. From this point of view the processes have only a slight advantage over pyrometallurgy.

TABLE N° 11

COMPARISON OF VARIOUS SOLUTION PROCESSES WITH SULPHATE MEDIUM

LEACHING PROCESSES		DISADVANTAGES	ADVANTAGES
Sulfatation and water washing	By concentrated H_2SO_4	Production of iron sulphate Difficult handling of sulphur in liquid or vapor form	Possible recycling of reagents
	By roasting	Poor copper recovery. Rejection of the residual solutions. SO_2 content in the exhaust gases. Solubility of iron.	Low energy requirement. Production of H_2SO_4. Easy to run unit operations. Low investments.
Sulphuric Acid Leaching	High temperature (140-200° C)	High investment and Operating costs Sulphur oxidised to sulphate, use of lime necessary	Fast and practically complete copper dissolution
	Medium temperature ($\sim$100° C)	Poor extraction. Residue recycling after flotation	Moderate grinding Acceptable temperature
	Medium temperature after ultra-fine grinding	Cost of ultra-fine grinding	Satisfactory yield and speed of copper solution Acceptable temperature

- The dissolved iron presents various problems especially during electrowinning.

- In the case of complex sulphides the metal separations are more difficult in a sulphate medium than in an ammonia or chloride medium.

On the other hand, the method has various advantages :

- Sulphatation roasting can be carried out autothermally, for example in a fluidized bed reactor ;

- The various unit operations are relatively well tested and do not need particularly elaborate or expensive equipment. The investment is estimated at 80 % of that for a pyrometallurgical plant with the same capacity.

- The excess sulphuric acid production can be an advantage if oxide ores are also present. Leaching with H_2SO_4 can then be carried out profitably with this surplus acid (Lakeshore Hecla process).

H_2SO_4 Leaching under Oxygen Pressure

Complete oxidation of sulphide concentrates can be obtained rapidly at a temperature of 140° to 200° C under high oxygen pressure. The sulphur is oxidized into sulphate and the costs of the oxidizing agent and lime (calcium sulphate precipitation) are the same as for the ammonia process.

Investments and operating costs are high and the process involving moderate temperature of the order of 100° (slightly below the sulphur fusion point of 112.8° C) are preferable (fewer engineering problems, lower heating and compression costs and recovery of elemental sulphur so that lime is not necessary).

Two principal schemes for leaching at moderate temperatures have been proposed :

- Sherritt-Gordon process [14] : leaching at 110° C, 14 to 35 Atm. maintained by O_2. Copper dissolution is only about 65 % and the residue must be concentrated by flotation and recycled.

- Lurgi process at Mitterberg (pilot plant in Austria, 3T concentrate/D) : dry grinding (10 to 20 microns) with a vibrating mill, leaching at 110° C, 3 to 15 Atm. kept up by O_2 ; very good Cu dissolution (98 %). [15 - 16]

These two processes also make it possible to recover Ni and Zn. They also seem to be particularly attractive for concentrates rich in arsenic and antimony. In order to compare them, it is necessary to compare the costs of flotation of residues with recycling on one hand to the costs of concentrate drying followed by ultra-fine grinding (grinding energy : 0.33 to 0.83 kWh/kg of copper for a 30 % concentrate) on the other hand. It is clear that the first method is more economical but the question should be reexamined for the case in which wet grinding would be possible.

The interest of sulphide concentrate leaching in sulphate medium lies in the fact that no other elements than those already present in the concentrate are added.

Sulphuric solution also makes possible the recovery of metals such as zinc and nickel which could be interesting in the case of complex sulphides.

It does however have the disadvantage that precious metals are not recovered and that there are considerable heating, compression and sometimes oxidation costs.

3.2.2.1.4 Other leaching Media

Solution in cyanide or nitrate media are sometimes proposed. These techniques use reagents which are expensive or difficult to handle (cyanide) or which have both disadvantages at the same time and are hard to recycle, without any decisive advantage.

3.2.2.1.5 Discussion

The only really convincing and realistic basis for a comparison of the various hydrometallurgical processes for copper sulphide concentrate processing is the investment and operating costs. However there are so many processes and variants and the concentrates to be processed are so different that a general discussion is impossible. It therefore seemed appropriate to present a brief comparison of the main characteristics of the available processes.

Iron Recovery and Sulphur Oxidation State

These two points, rather than the other factors, distinguish the various processes and also have an important effect on operating costs.

It should be noted in particular that the processes which leave sulphur in the elemental state require only about 30 % of the oxidising agent required by processes producing sulphate. An additional advantage is sometimes indicated for the former consisting of the fact that elemental sulphur may be recovered and marketed. Separating sulphur from the residues is not however a simple and cheap operation and it is not certain that sulphur recovery is economical. On the other hand, the other processes have the disadvantage that the sulphate must be precipitated with lime, producing calcium sulphate which must be removed and dumped.

Fortunately, in certain processes iron is precipitated during leaching. In others, it is made insoluble during roasting and, lastly, in chloride processes the iron is dissolved and may be recovered as a marketable product, which may be one of the advantages of these processes.

Recovery of Dissolved Copper

All the techniques which can be applied

industrially at the present time use a conventional electrowinning unit representing 20 to 40 % of the total investment.

Some methods use a liquid-liquid extraction stage which has the definite advantage that a purer copper can be obtained than by pyrometallurgical techniques or by traditional hydrometallurgical techniques. The investment cost of a typical liquid-liquid extraction and electro winning circuit breaks down as follows : 20 % for the extraction unit, 20 % for the solvent and 60 % for the electrowinning unit. The costs for the last item can be reduced by a third through the use of high current densities.

Reagent Costs

The cost of leaching reagents can be considered negligible for processes in a sulphate medium.

For the Arbiter process with ammonia the recycling of ammonia can be estimated at over 95 % (provided that ammonium sulphate is not produced).

For processes in chloride medium, data are not available as to chloride losses. The latter are, however, considered low.

Nature of Concentrates which can be processed

When a concentrate contains a gangue which is acid consuming (for example limestone) the only applicable processes are ammonia processes and sulphatation roasting processes. Sulphatation roasting in the presence of lime seems to be especially attractive in this case.

The capacity of the various processes for treating complex concentrates (for example Cu-Zn, Cu-Zn-Pb, Cu-Ni-Co) is very important since hydrometallurgical processes have precisely the greatest advantages over pyrometallurgy in this field. Separations are probably easier to make in chloride media than in ammonia media but each case should be examined separately.

The principal harmful impurities, arsenic and antimony, remain in the residues during ammonia leaching provided that there is sufficient iron to form insoluble ferric arsenates and antimonates. This is also the case in a sulphuric acid medium, whereas As and Sb are dissolved in chloride medium.

Technological Problems

The main technological problems concern corrosion, especially for chloride processes, since ferric chloride is very corrosive and requires the use of expensive products leading to large investments. The processes with sulphuric acid under pressure also involve problems from this point of view.

Processes with an ammonia medium, and leaching after sulphating roasting, present the fewest problems.

Recovery of Precious Metals

Sulphide concentrates always contain precious metals. The degree to which the latter are dissolved during the leaching phases depends to a great extent on the mineralogical form in which they are expressed but also on the intensity of leaching and the fineness of grind of the leached concentrate.

Palladium, platinum and silver are soluble in ammonia solutions. Palladium and silver are soluble in sulphate solutions and all the precious metals are soluble in chloride media. However industrial practice does not always show results in accordance with the above remarks. At

Mitterberg (Austria), for example, gold and silver remain in the sulphuric leaching residue and must be extracted by cyanidation.

Nature of Effluents

While pyrometallurgical processes present the problem of gaseous effluents (SO_2), they are however excellent since they leave the iron in an inert and stable state without water soluble salts. On the other hand, while leaching processes eliminate the disadvantage of SO_2 air pollution they often produce harmful liquid and solid effluents

Energy Consumption

Leaching processes for sulphide concentrates are relatively new and no precise data are available on their energy consumption ; but, according to H.H. Kellogg, this consumption is definitely higher than for pyrometallurgical processes. (Table N°12)

PROCESS	PRODUCT	CONSUMPTION 10^9 Joule/t
Flash smelting, converting electrorefining	Cu cathode	23
Leaching electrowinning (ammonia + oxygen or ferric chloride	Cu cathode	~ 100

TABLE N° 12

Comparison of energy consumption (process fuel equivalent) for pyro- or hydrometallurgy (according to H.H. Kellog [17]) including energy for production of sulphuric acid and not including energy for ore extraction and benefication.

3.2.2.2 Hydrometallurgy of Low Grade Sulphide Ores

Because of the progressive drop in deposit grades and the increase in copper needs, large open pit mines are taking an increasingly important role. This type of exploitation involves the extraction of considerable volumes of materials which would be considered sterile for conventional processing but which nevertheless contain considerable copper contents (up to about 0.8 %).

Dump leaching of these ores makes it possible to recover a large part of the copper content with moderate investments and operating costs.

The general principle of the method can be summarised as follows :

Ore heaps, prepared with various techniques depending upon the exploitation, are soaked with aqueous solutions of sulphuric acid (pH 2 to 3).

As the solutions pass through the heaps and soak the ore several chemical and biochemical reactions take place resulting in the dissolution of copper and iron (in the form of sulphates).

The solutions charged with copper are collected at the base of the heap.

They are then passed through a precipitation unit where copper is generally recovered by cementation on iron (various kinds of scrap, detinned cans and sometimes iron sponge). Thus copper of varying purety (50 to 90 %) depending upon the cementation technique is obtained, and is then refined by conventional metallurgical processes.

The barren solutions are recycled at the top of the heap.

It is possible to regulate pH (by H_2SO_4) at several points as well as to add water and carry out purging (so as to maintain constant total salt concentration). Some precautions are also taken in order to maintain temperatures within optimum limits and favorise the activity of certain bacteria playing a determining role in copper leaching processes.

In some deposits, rich ore is extracted from underground mines. It is then possible to remove and heap up large quantities of low grade rock which can be processed by leaching.

Some mining companies use in situ leaching.

The basic principles of this method are similar to those of dump leaching i.e. the solution moves from the top down through the leached ore and dissolves copper. Adequate arrangements are necessary for the collection of the copper rich solutions without excessive losses. These solutions are then pumped to the surface and recycled after recovery of the copper. Techniques utilising acid solutions containing ferric iron, oxygen and bacteria produce a relatively rapid dissolution of copper minerals as in dump leaching.

Leaching in a deposit which has not been worked in any other way constitutes a special case of leaching called solution mining.

In both cases the basic problem is rock fracturing (conventional or nuclear) and the containment of solutions (the deposit boundaries must be made impermeable).

There are different problems in dump leaching depending upon whether oxidized or sulphide ores are being processed.

The chemistry of the process is well known and controlable in the case of oxidized ores. The main difficulties involved concern the engineering of the heaps upon which solution flows and extraction speeds and yields depend. The process is complex in the case of sulphide ores since bacterial phenomena are involved and it seems that at the present time the main objective is to understand how to construct heaps so as to control bacterial activity.

Dump leaching has many advantages :

- The possibility of processing ores which would not be economically processed by any other method ;
- Relatively low investments and operating costs ;
- Relatively low energy consumption (basically breaking, handling, pumping).

The disadvantages, however, are not negligible :

- Environmental problems (pollution of groundwater, landscape, etc.) ;
- Long periods of time necessary for application ;
- Limited extraction yields.

In relation to dump leaching the advantages of in situ leaching are :

- Small investment and lower operating costs ;
- Minimal (visible) disturbance of the environment ;
- Reduction in time necessary for application.

3.2.2.3 Hydrometallurgy of Oxide Ores and Concentrates

Under present technical conditions it is difficult to enrich oxide ores by flotation.

In the case of oxide ores, hydrometallurgy is not a possible replacement method for pyrometallurgy or a supplementary method, but is the main processing method.

Many leaching techniques have been studied. The main reagent used is sulphuric acid but some attempts have been made to use other media (chloride, cyanide, ammonia).

3.2.2.3.1 Leaching in Sulphuric Medium

This is the medium the most usually used because it is the cheapest. It dissolves copper carbonates and tenorite well, but it is less successfull with cuprite. During attack on silicates the silica which is liberated forms gels which lead to inhibiting effects and clogging. It is mainly used in dilute solutions (less than 50 g/l) so that selective dissolution is possible which is more effective because copper dissolves faster than elements such as Fe or As or parts of the matrix. This reduced activity has the advantage that acid consumption and purging are limited but the contact time required is longer (reduction in the specific capacity of the tanks).

In situ leaching, heap leaching, vat leaching or agitation leaching are used depending upon the composition of the ore, and its grade and physical characteristics.

On this point the following information is available (average data from US workings) :

Leaching	Ore	Average solution	Length of leaching cycles
In situ	Very low grade (run-of-mine ore)	50-60 %	Measured in years
In heaps	Crushed and relatively porous	~ 60 %	Measured in months
In tanks	Crushed and not very porous	~ 80 %	Measured in days
Agitation	Varies	> 90 %	Measured in hours

TABLE N°13

LEACHING METHODS

A comparison between the various techniques cannot be made adequately except for a given case of application since the choice of a method is the result of a compromise between advantages and disadvantages and there can be considerable variations between the different cases. We will therefore limit ourselves to general remarks :

Ores which can be Processed

In situ and dump leaching are most often used for very low grade ores for which only inexpensive methods are possible. In view of the low grade of the ore, the increased efficiency which could result from the two other methods would not be sufficent to cover higher cost.

These two methods are, however, only technically applicable if ore porosity and permeability are sufficiently great. The method involving agitation is applied specifically to fine ores or fractions while non porous ores require extensive crushing.

Time required for Application

The periods necessary are relatively short for vat leaching (with or without agitation) and are longer for the other methods.

Final Concentration of Solutions and Copper Recovery

The average final concentrations of enriched solutions are given in the following table :

	Reference N°	Average Cu Contents
In situ leaching	1	2 to 5 g/l
Dump Leaching	2	2 to 5 g/l
Vat Leaching	3	About 30 g/l
Reactor Leaching with Agitation	4	Maximum of several g/l

TABLE N°14

FINAL COPPER CONTENTS OF SOLUTIONS

Only vat leaching leads to copper contents which are high enough so that simple electrowinning is sufficient to recover this copper (20 to 25 g/l of H_2SO_4 for a current efficiency of 70 to 80 at 100 A/m^2, approximately 2 kWh per kilogramme of copper, using Pb anodes with 3.5 % Sb).

The copper content of solutions from reactors with agitation is limited by various factors which are relatively uncontrolable (ore grade, pulp density, liquid-solid separation difficulties, dilution water from grinding) and in practice make counter current circulation or recycling, which could increase the copper content in solution, impossible.

Utilisation of methods 1, 2 or 4 involves either copper cementation (low investment but high operating costs : consumption of iron and sulphuric acid and need for thermal refining) or concentration by liquid-liquid extraction (lower operating costs but higher investments) before electro winning.

Reactor leaching with agitation is certainly the solution method consuming the most energy per tonne processed.

The following table gives approximate idea of energy costs (kWh/t) for copper solutions by means of the four methods.

Item \ Leaching	In situ	Dump	Tank	Reactors with Agitation
Extraction	1	5	5	5
Crushing	0	0.5	1	1
Grinding	0	0	0	15
Leaching	0.05	0.05	0.05	1
Total	1.05	5.55	6.05	22

TABLE N°15

APPROXIMATE ENERGY CONSUMPTION FOR THE VARIOUS LEACHING

Investment and Operating Costs

The relative investments and operating costs for the various methods depend upon many factors (especially nature and grade of ore). For example, the following table can be given for a 1 % copper ore :

Leaching	Copper Recovery	Costs (indices)	
		Investment	Operating costs
In situ	Liquid-liquid extraction and electrowinning	1.3	0.3
Tank	"	2.7	0.6
Reactor with agitation	"	3.4	0.7
Dump	"	2.0	0.5
	Cementation	1.0	1.0

TABLE N°16

INVESTMENTS AND OPERATING COSTS (INDICES)

(according to Lewis and al.) [18]

This short summary is intended only as an indication but it does however show that in situ leaching is very economical for low grade ore and that there is very high energy consumption for agitation leaching (this technique could only be justified on technical basis in the case of high grade and easily crushed ores). Dump leaching with cementation may seem interesting but it must be kept in mind that the sales price of the copper produced will be definitely lower than that for the other methods.

3.2.2.3.2 Other Leaching Methods

Other leaching media are sometimes used :

- Chloride medium :

The main interest of this method is the possibility of the existence of monovalent copper in chloride medium. There are many disadvantages (especially the undesirable dissolution of iron and calcium) and the replacement of H_2SO_4 can be considered only in certain special cases. The example of the Timna (Israel) mine can be mentioned which studied this possibility. Israel has large potential hydrochloric acid resources (Dead Sea) while the country is forced to import most of the sulphur it consumes.

- Ammonia Medium :

The main interest of this medium lies in its selectivity (case of carbonate gangue) and the relative facility with which the ammonia can be recycled. However, the reagent used is relatively expensive. [19]

- Cyanide Medium :

Outside of its selectivity with respect to iron, this solution medium seems to have only disadvantages (cyanide consumption, cost of reagent, toxicity, pollution). [20] [21] [22]

3.2.2.3.3 Discussion

None of the various solution media proposed has a decisive advantage over the sulphuric medium using a reagent which is particularly economical and can be regenerated during copper recovery operations (electrowinning and liquid-liquid extraction).

In addition to its selectivity, the aqueous ammonia solution also has the advantage of being easily recycled but evaporation losses generally make the process uneconomical.

The ores presenting the most problems are also ores with a carbonate gangue (acid consumption) and ores containing copper silicates (difficult dissolution). In addition to the considerable efforts now being made concerning effective flotation (for example research on specific chelating collectors), research on chemical processing of silicate and carbonate ores deserves to be continued on a large scale.

3.2.3 Mixed and Roasting Processes

3.2.3.1 The LPF Process (Leaching-Precipitation-Flotation) is especially well adapted to the processing of mixed sulphide-oxide ores which cannot be separated. [23]

The process consists in leaching by dilute sulphuric acid so as to dissolve most of the copper oxides followed by precipitation of the copper in the pulp by iron (cementation) and flotation of the cement. It is not essential that the copper sulphides be prefloated (they are then concentrated with the cemented copper) but this is sometimes necessary when other minerals are to be recovered (molybdenite for example) and when pyrite is to be separated for the preparation of iron for cementation.

It seems to be difficult to give general indications as to the recovery of other elements of value by the LPF process since their behaviour obviously depends on the mineral combinations in which they are found and the flow scheme utilised. Molybdenite, and precious metals are recovered to the extent to which they are floated with the sulphides.

In most cases commercial application of the process requires some modifications of the conventional flow schemes i.e. addition of units for leaching, copper precipitation and magnetic separation (iron recovery in tailings) and utilisation of acid resistant materials in parts of the installation.

The higher investment and operating costs for an LPF unit in relation to a direct sulphide flotation unit should be compared with increased profitability due to higher copper (oxide) recovery. Clearly the result of this comparison depends upon copper prices, the nature of the ore, rate of oxide copper recoverable by LPF, nature of the gangue, etc.) and the flow scheme adapted. The additional profitability is often very low.

A study by the U.S.B.M. (1970) estimated that for an ore with 2.1 % Cu (0.43 % oxide of copper) an increase in the recovery rate from 76.8 % to 91.8 % leading to additional copper recovery of 3.2 kg/t of unsorted ore would produce only a marginal profit (it should be noted that the ore then consumed much sulphuric acid : 40 kg/t of ore).

3.2.3.2 Chloridation Roasting

Many studies have been made on chloridation in extractive copper metallurgy principally for oxide ores but for sulphide ores as well. The only chloridation roasting technique which is now being developed to some extent is the segregation process.

Simple chloridation can be carried on either at a relatively high temperature (for example 800° C) so as to vaporise copper chlorides or at a lower temperature so as to leave copper chlorides in calcine from which they are then extracted by leaching.

The main chloridizing agents now in use or proposed are sodium chloride (especially for segregation), hydrochloric gas, chlorine and iron chlorides.

Chloridation roasting processes seem to be of interest only for copper silicate ores which are difficult to leach and to a lesser extent for carbonate ores although the latter can consume much chloridation reagent (which is then hard to recycle). It is not certain that chloridation roasting is of interest in relation to leaching in the case of copper oxides.

It has sometimes been considered for the processing of sulphide ores but there is then the serious problem of sulphur which is found mainly in the form of H_2S (chloridation by HCl) or in the form of sulphur mixed in varying proportions in chlorine-sulphur compounds (chloridation by Cl_2).

Many other elements can give volatile chlorides (Ag, Au, As, Sb, Sn, Pb, Zn etc) at the same time as copper. The main gangue constituants are generally inert (SiO_2, Al_2O_3, etc.). Iron can be kept in the calcine by maintaining sufficient oxygen or steam partial pressure in the roasting gases.

Aside from separation, chloridation roasting for copper ore has not yet been used for industrial development and there have been very few studies at the pilot scale. Data concerning the operation of particular types of units is therefore estimated and fragmentary and there are many problems :

. Important corrosion problems ;

. Problems in the processing of chloride complex mixtures ;

. Energy consumption (roasting, regeneration of chloridation reagent) ;

. Environmental problems.

A special case of the chloridation roasting method is the segregation process and in particular the variant called TORCO (Treatment of Refractory Copper Ore). The ore is heated in a fluidised bed at about 750° C and is then placed in a chamber with carbon reducing agent (about 2 %) and sodium chloride (about 0.2 %). After various chemical reactions (generation of HCl from steam ore silicates and sodium chloride, chloridation of copper by HCl and reduction of copper chloride by carbon), copper is found in the metallic state on the reducing agent. Copper is recovered by flotation, sometimes after grinding.

In existing plants (Akjoujt 90 kt/m of ore) copper recovery is low (around 70 %). Various trace elements can be relatively well recovered with the copper (for example Au and Ag).

Various analyses indicate that the Akjoujt unit is not profitable. It should however be noted that the context is very unfavorable.

It should not therefore be concluded that the segregation process is not profitable because of the results of the Akjoujt plant. It remains true however that at the technical level this promising process for treating so called refractory ores presents complex problems (separation, flotation, recovery of dusts, general scheme) so that a complete analysis and a search for a specific solution would be required in each particular case.

3.3 Installations in the Community and Processing Methods Used (Present Situation and Projects)

At the present time no copper ores are processed by hydrometallurgical methods in the EEC except for the negligibly small production from drainage water from old mining operations at Avoca in Ireland.

In general, the European mines are characterised by complex mineralisations (except for Avoca and Fragne in Italy), by low processing capacities and by the fact that the ores are mined underground.

The main points concerning operations in the Community will be presented in the following paragraphs.

The geographical location of these installations is shown on the map in Figure n° 8 below.

3.3.1 Italy

There are four plants producing copper concentrates by flotation in Italy i.e.

- Fragne (Piedmont)
- Fenice Campana (Tuscany)
- Funtana Raminosa (Sardinia)
- Rosas (Sardinia).

3.3.1.1 The Miniera di Fragnè-Chialamberto is a private mining company which owns a copper mine at Chialamberto in Piedmont with estimated reserves of 1.5 Mt of ore containing 1.5 % copper in the form of chalcopyrite.

There is an ore dressing plant with a capacity of 700 t/d and it now treats an ore mined during the preparatory work.

The mine should reach full operating capacity in 1979. Processing consits of grinding to 80 % passing 75 microns followed by flotation of the chalcopyrite by xanthates in an alkali medium and then flotation of the pyrite in an acid medium. A total of 25 kt will be processed in 1977.

The metallurgical balance is as follows (in %) :

		Grade %		Recovery %
	Weights %	Cu	S	Cu
Run-of-mine	100	1.51	7.89	100
Chalcopyrite	6.12	23.55	33.97	95.5
Tailings	83.91	0.04	0.81	2.2

3.3.1.2 The mining installations of Fenice Capanne as well as those of Funtana Raminosa and Rosas belong to the government owned company EGAM.

The ore being mined contains sphalerite, marmatite, pyrite, galena and chalcopyrite in a siliceous gangue.

Its chemical composition is as follows :

- . S 4-5 %
- . Fe 7 %
- . Cu 0.3 %
- . Zn 3-4 %
- . Pb 1 %

Reserves in ore of this type make it still possible for mining to continue for a period of 18 months at the rate of 120 kt/y. Mining at greater depths would make it possible to reach reserves of 4 Mt of a copper-bearing ore with a grade of 1.7 % copper and 2 Mt of an ore similar to the ore now being mined.

Four types of concentrates with the following compositions are produced :

	Sphalerite Concentr.	Galena Concentr.	Copper* Concentr.	Pyrite Concentr.
S %	33.2	10.9	38.3	42-44
Fe %	8.3	5.6	25.9	36
Cu %	1.5	4.9	8.2	0.5
Pb %	0.9	65.1	5.5	0.4
Zn %	50.8	4.4	16.0	4
	ρ Zn:68%	ρ Pb:74%	ρ Cu:37%	

Flotation is of the conventional semi-bulk type. A mixed galena-chalcopyrite preconcentrate is first

* The copper concentrate is now (Feb. 1978) richer and the recovery higher.

floated by means of ethylxanthate at a pH of 8. The chalcopyrite of this preconcentrate is then depressed by adding cyanide while the sphalerite in the tailings of this first separation is reactivated by copper sulphate with massive lime additions. After desliming and change of water a marketable pyrite concentrate is floated.

The copper-bearing residue is sold to Bolinden in Sweden.

3.3.1.3 At Funtana-Raminosa 60 kt of run-of-mine with a grade of 0.75 % copper, 0.65 % lead and 2.0 % zinc are processed annually. Three concentrates and tailings with the following grades are produced :

	Chalcopyrite Concentr.	Galena Concentr.	Sphalerite Concentr.	Tailings
Cu %	29.09	3.16	1.99	0.05
Pb %	2.63	47.50	0.53	0.07
Zn %	3.58	10.07	47.15	0.25
Ag/g/t	800	1 300	140	
Cd %			0.9	
Metal Recovery%	73.7	76.5	79.6	

After grinding of 65 % of the particles to less than 75 microns, the process consists of a semi-bulk Cu-Pb flotation and then depression of galena with dichromate followed by reactivation of the sphalerite in a basic medium with copper sulphate.

The copper concentrate is sold to Rio Tinto in Spain while the lead and zinc concentrates are processed in Sardinia.

A project exists for the extension of the mine and ore dressing unit, the capacity of which will be raised to 100 or 120 kt/t with reserves of 2 Mt of ore.

3.3.1.4 The Rosas ore dressing unit processes two types of ore in separate batches at a rate of 200 t/d.

Processing of the first type assaying 2.56 % Zn, 0.33 % Cu and 0.22 % Pb, gave the following copper concentrate grades in recent months :

- Cu.................................... 27.5 %
- Pb .... 2.3 %
- Zn .... 9.5 %
- Ag ....956 g/t

at a recovery of 45-50 %.

This type of ore will be processed so as to produce only a mixed Pb-Zn-Cu concentrate which will be treated by the Imperial Smelting plant at Gavino in Sardinia.

The second type of ore is only lead and zinc-bearing (Zn : 2.69 %, Pb : 2.8 %).

The Rosas reserves are very low (300 kt) and it is planned to close this installation in the near future.

3.3.2 The Irish Republic

There are two installations in the Irish Republic producing marketable copper concentrate :

At Avoca an ore is floated which contains chalcopyrite and pyrite in a siliceous gangue and has a 0.5-0.6 % copper grade.

The process used is conventional. The chalcopyrite is floated in a basic medium by dithiophosphate and the pyrite is then floated with xanthate after acidizing of the pulp.

The processing unit has a capacity of 4 kt/d and in actual fact treats between 2.2 and 2.5 kt/d and produces 40 t/d of a concentrate containing 21-24 % Cu, 1 % Zn and 100 g/t of silver, as well as 150 t/d of a pyrite concentrate with a 50 % sulphur grade.

Depending upon the origin of the ore in the deposit copper recovery can be 93 % (ore mined underground) or 75 % (open pit). This difference is explained by the partial oxidation of the open-pit ore and by the presence of clay which makes flotation more difficult.

The copper concentrate is sold either to Boliden in Sweden or to Rio Tinto in Spain. The pyrite concentrate is used locally for fertilizer production.

Reserves amount to 8 Mt of ore with a grade of 0.7-0.8 % Cu but the company is now having economic problems because the combined low copper prices, low grade of ore, which is mainly mined underground.

Projects for in situ leaching are being studied.

It should also be noted that a very small amount of copper is produced by cementation from the drainage water of old mines nearby.

At Tynagh Irish Base Metals treats a complex sulphide ore (galena, sphalerite, pyrite, copper arsenious-sulphides) in a barite and quartz gangue with an average grade of 4 % Pb, 3 % Zn, 0.10 % Cu and 53 g/t of Ag. Copper is therefore a by-product of lead and zinc.

Plant capacity is about 2 kt/d. Dense medium separation is included in the processing method. After crushing it makes it possible to eliminate 30 % in weight as tailings from the 10-40 millimeters particle size fraction.

The flotation method is conventional for this type of ore. There is a bulk Pb-Cu flotation followed by depression of lead by means of dichromate, and then flotation of zinc after regrinding. The fines contained in the ore before grinding are treated separately and a mixed lead-zinc concentrate is obtained.

The preconcentration tailings are used for underground fill and the flotation tailings are impounded on the surface after barites flotation.

The characteristics of the sulphide concentrates are as follows :

- Lead concentrate : 26,850 t in 1976
  - . 76.6 % Pb
  - . 4.5 % Zn
  - . 0.7 % Cu
  - 450g/t Ag

Pb recovery : 80.9 %
Ag recovery : 38.6 %

- Zinc concentrate : 25,800 t in 1976
  . 51.3 % Zn
  . 2.3 % Pb
  . 0.4 % Cu
  .450 g/t Ag
  Zn recovery : 73.7 %
  Ag recovery : 10.8 %

- Copper concentrate : 2,960 t in 1976
  . 23.1 % Cu
  . 17.1 % Pb
  . 8.7 % Zn
  .3,600 g/t Ag
  . 600 g/t Hg
  . 5 % As
  . 5 % Sb
  Cu recovery : 60.3 %
  Ag recovery : 34.0 %

- Mixed lead-zinc concentrate : 3,450 t in 1976
  . 33.9 % Pb
  . 18.1 % Zn
  . 1.4 % Cu
  . 360 g/t Ag
  Pb, Zn Cu and Ag recovery is about 4 %

It is difficult to market the copper concentrate firstly because of the presence of antimony, arsenic and mercury and secondly because of processing problems due to the high lead and zinc contents. On the other hand its silver grade is an attractive factor.

3.3.3 West Germany

The only copper mine which is now being worked is at Rammelsberg in the Harz Mountains in Lower Saxony.

Two types of ore are treated. The first is called "rich" and has a grade of 6 % Pb, 1.4 % Cu, 16 % Zn and 10 % Fe. The second called "banded" has a grade of 4 % Pb, 1 % Cu and 11 % Zn.

Both are characterised by the high degree of intergrowth of the minerals.

Two plants treat these ores using the same method. One unit has a capacity of 720 t/d and the other a capacity of 600 t/d. The flotation principles are as follows. After grinding to 60 % passing 40 microns, a differential flotation process is applied by careful addition of collectors so as to produce a copper concentrate, a lead concentrate and a mixed concentrate which is reground at 95 % passing 40 microns and selectively floated.

The metallurgical balance is as follows :

"Rich" Ore

	Pb %	Cu %	Zn %	Fe %	Production t/d
Run-of-Mine Ore	6	1,4	16	10	720
Cu concentrate	6	22	12	24	46
Pb concentrate	39	1.5	21	9	111
Zn concentrate	5	1	44	10	262
Recovery	66	55	80		

"Banded" Ore

	Pb %	Cu %	Zn %	Fe %	Production t/d
Run-of-Mine Ore	4	1	11	9	600
Cu concentrate	8	22	11	23	27
Pb concentrate	38	2.5	21	9	63
Zn concentrate	6	1,0	44	9	150
Tailings	0.7	0,05	0.8	8	360
Recovery	60	56	77		

The copper concentrates are sold to Metallgesellschaft while the zinc concentrates are processed at the site. The Pb concentrates are sold to an Imperial Smelting plant.

The Pb-Zn and Cu flotation tailings are subjected to two additional flotations. One produces a pyrite concentrate which is not marketed at the present time while the second gives a barite concentrate which is used essentially as drilling mud.

The present economic reserves at Rammelsberg are such that it is expected that the mine will be closed towards 1985.

3.3.4 United Kingdom

In the United Kingdom there are several tin processing units in Cornwall which in the past recovered copper sulphides as by-products. At the present time it seems that the Geevor and South Crofty installations produce only tin and that only the Wheal Jane mine sells a copper concentrate.

The Wheal Jane mine which opened in 1971 processes by flotation and gravity concentration a tin ore assaying 1.25 % Sn, 1.9 % Zn, 0.3 % Cu, 9 % F and 1 % As.

After grinding the ore to 300 microns and desliming the sulphides are bulk-floated.

The rougher concentrate obtained is reground. Liberated cassiterite is removed and then a copper-zinc concentrate with a grade of about 5 % Cu, 3 % Zn and with 250 g/t of silver is produced by differential flotation from pyrite and arsenopyrite.

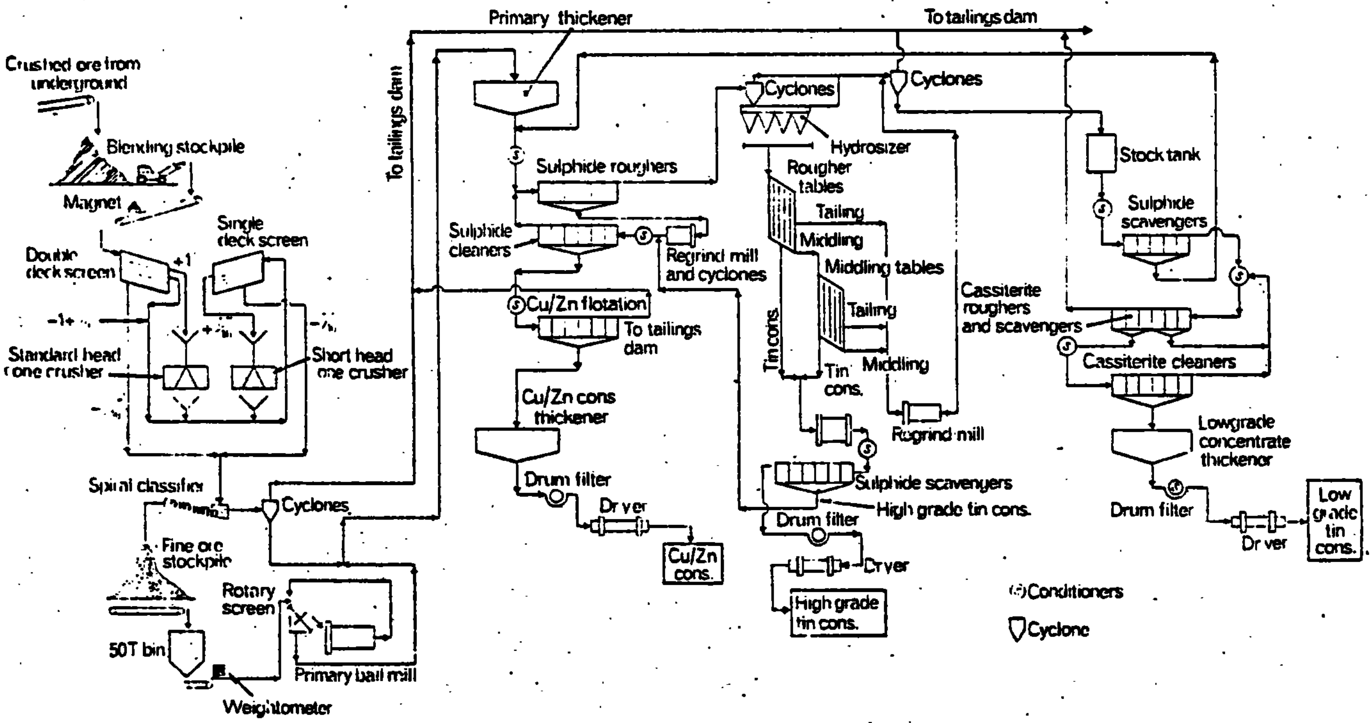
Cassiterite is concentrated partly by tabling, partly by flotation, after further sulphide removal.

Copper recovery is 63 % and the production of 390 tonnes in 1973 and 1974 was to be raised to 600 tonnes in 1976 corresponding to an increase in processing capacity from 206 kt/year to 320 kt/y.

Zinc and tin recovery are expected to be 51 and 62 % and tonnages in metal content should rise from 1,600 to 2,500 tonnes for Sn and from 2,000 to 3,100 tonnes for Zn.

The mixed zinc-copper concentrate is sold to Boliden in Sweden.

Another mine, the Mount Wellington is also about to start



Simplified flowsheet for Wheal Jane crushing plant and concentrator

Fig. n° 19

to operating in Cornwall. The ore dressing unit will treat 850 t/d of ore assaying 1.2 % Sn, and 25 % S. A mixed sulphide concentrate (copper-zinc with silver) will first be separated by flotation. After tabling, the flotation residue will be refloated in order to produce a concentrate with a grade of 35 % Sn (Sn recovery 70 %). We do not know the Cu grade of the ore, that of the concentrate nor the planned copper production.

3.3.5 France

At the present time in France the Salsigne plant is the only installation producing copper concentrates. There is also a project for opening a mine and an ore dressing unit at Bodennec in Brittany.

3.3.5.1 Salsigne

The Société des Mines et Produits Chimiques de Salsigne mines a vein deposit about fifteen kilometers north of Carcassonne from which sulphur, arsenic, gold, silver, copper and bismuth are extracted from arsenopyrite, pyrite and chalcopyrite. The copper grade of the ore reaches on the average 0.1 % but is constantly falling. 160 kt of ore are processed annually.

After conventional ore crushing and grinding to 80 % passing 75 microns, the sulphides are bulk-floated by amylxanthate, and the concentrate contains almost 50 % pyrite and nearly 50 % arsenopyrite along with some pyrrhotite, chalcopyrite and bismuthinite, at a copper grade of 0.2-0.5 %.

After agglomeration with lignosulfonates, the concentrates are smelted in a water-jacket furnace. Part of the sulphur, arsenic and bismuth escapes in gaseous form, while at the base a matte containing copper, silver and gold is separated from the slag.

Flux is added to the furnace as well as a cheap copper-bearing product (for example copper and arsenic-bearing sludge from copper electrolysis), the object of which is to aid in the collection of gold. This addition amounts to 30 t/month of copper. The copper content of the matte can vary from 6-20 % depending upon market conditions. The total yield is 80 % for the copper and annual Cu production is 450 tonnes.

3.3.5.2 Bodennec

A massive sulphide deposit with galena, sphalerite, pyrite and chalcopyrite in a schist gangue was recently discovered at Bodennec.

Exploitable underground reserves amount to 1.7 Mt with a grade of 1.8 % Cu, 2.7 % Pb, 4.5 % Zn and 74 g/t of silver.

The planned daily production is 700 t/d.

If the feasibility study is positive, exploitation could begin around 1981.

3.4 Economics of Copper Ore Processing Methods

In this chapter, we will briefly examine the sales formulas for processed concentrates and will then consider the investments and running costs of the various processes now used or under study.

There may be considerable differences between the various kinds of copper ore (grade, reserves, monometallic or polymetallic character). It is therefore necessary to be very careful when applying data taken from the litterature to ores which are not of the classical porphyry copper type with a Cu grade of about 0.8 % and with reserves amounting to several tens of million tonnes.

3.4.1 Marketing of Flotation Concentrates and Cement Copper

Copper concentrate produced by flotation is generally sold according to the following formula :

$$P = (x - a)\ C - (x - a)\ R - S$$

where P is the sales price of a tonne of concentrate having a grade x, a is a value of between 1 and 3 %, C is the copper wirebar price, R is refining costs and S is smelting costs. There are several bonuses or penalties which can be added to or substracted from this price P :

- 95 % of the silver is paid for at market prices for the contents over 30 or 35 g/t,
- 92 % of the gold is paid for at market prices for the contents over 1 g/t,
- There is a penalty for zinc and lead if their grade in the concentrate exceeds a figure varying from 3 to 10 %,
- There is a penalty for arsenic in the concentrate if it assays over approximately 0.5 or 1 % (penalty of about $ US 2.5 per point),
- There is a penalty for antimony in the concentrate when its grade is over 1 % (penalty of about $ US 1 per point).

Some other elements can also affect concentrate prices (lead, iron, silica).

It is also the practive at the present time to deduct anti-pollution expenses (about 2 to 3 US cents per kg of copper). A marketing commission may also be deducted from the concentrate price.

In the case of copper concentrates from Funtana Raminosa (Sardinia) smelting costs amount to £ 29.65 per tonne and refinery costs to £ 89.65 per tonne of copper content. This concentrate has a grade of 29 % Cu, with 800 g/t of silver and 1 g/t of gold. Since copper and silver prices are respectively £ 700 per tonne and £ 87 per kg, the price of a tonne of concentrate in pounds sterling will be :

. Copper (29-1) x 700 x 10^{-2}	= £ 196.00
. Silver (800-31) x 87 x 10^{-3}	= £ 66.00
. Deduction for refining 29 x 89.65 x 10^{-2}	= £ 26.00
. Deduction for smelting 29.65	= £ 29.65
	£ 207.25

If a lead and/or zinc concentrate also contain copper the payment for the copper varies in relation to the smelter where it is processed (Imperial Smelting process or conventional process) or according to the need which a smelter may have at a certain moment for a concentrate with specific chemical characteristics as a part of its supply mixture. In general no more than 25 % of the contained copper is paid for.

Copper cement prices are based upon copper prices with a deduction for smelting and refining expenses which are about $ US 200 per tonne of contained copper for a cement with a Cu grade of about 80-85 %.

For each ore treated in a flotation plant there is a relation between the grade of the concentrate and the corresponding metal recovery. The mineral technologist chooses a concentrate grade and a metal recovery rate optimal from the economical point of view. Operating costs have a negligible effect on the choice of this economic optimum unless there is an additional operation (for example regrinding). Concentrates produced by flotation generally have a Cu grade of 25 % and the corresponding recovery are 80 - 85 %.

3.4.2. Investments and Operating Costs for Physical Processing

3.4.2.1 Dense Medium Separation

The investments necessary for starting a dense medium cyclone plant are now about 4,000 kF for a capacity of 720 t/d (or US $ 1,100 per tonne in 24 hours). The corresponding operating expenses are about US $ 1.3 per tonne treated.

3.4.2.2 Flotation

On the basis of the Marshall and Stevens index for 3rd quarter 1977 (Mining and Ore Processing) we have brought up to date an estimate of the cost of a copper-zinc processing unit of 1.2 kt/d given in "Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations" of E.A. Parkinson and A.L. Mular (1972). [24] (See table 1 and figure 20)

Mineral Processing Equipment costs and Preliminary Capital cost Estimations [24]

Example

Total capital investment for a 1 200 tpd plant

(1)	Purchased equipment costs based on M&S index (3rd quarter 1977) of 524 for mining and milling :	1 640
	Crushing 305 000 $ Grinding 496 000 $ Copper flotation 296 000 $ Zinc flotation 543 000 $	
(2)	Installed equipments costs (1,43 times Item 1) :	2 345 000
(3)	Process Piping (10 % of Item 2) :	235 000
(4)	Instrumentation (3 % of Item 2) ;	70 000
(5)	Buildings and site development (35 % of Item 2)	821 000
	Mill Building Crusher Building Assay, Machine Shop, Office	
(6)	Auxiliaries (5 % of Item 2) :	117 000
	Water supply Diesel Standby Power	
(7)	Outside lines (8 % of Item 2) :	188 000
(8)	Total physical plant costs (2+3+4+5+6+7) :	3 776 000
(9)	Engineering and Construction (25 % of Item 8) :	944 000
(10)	Contingencies (10 % of Item 8) :	378 000
(11)	Size factor (5 % of Item 8) :	189 000
(12)	Fixed capital costs (8+9+10+11) :	5 287 000
(13)	Working capital cost (10 % of Item 12) :	529 000
(14)	TOTAL CAPITAL INVESTMENT (12+13) :	5 816 000

Cost per tonne worked in 24 hours.................. 4 850 US $

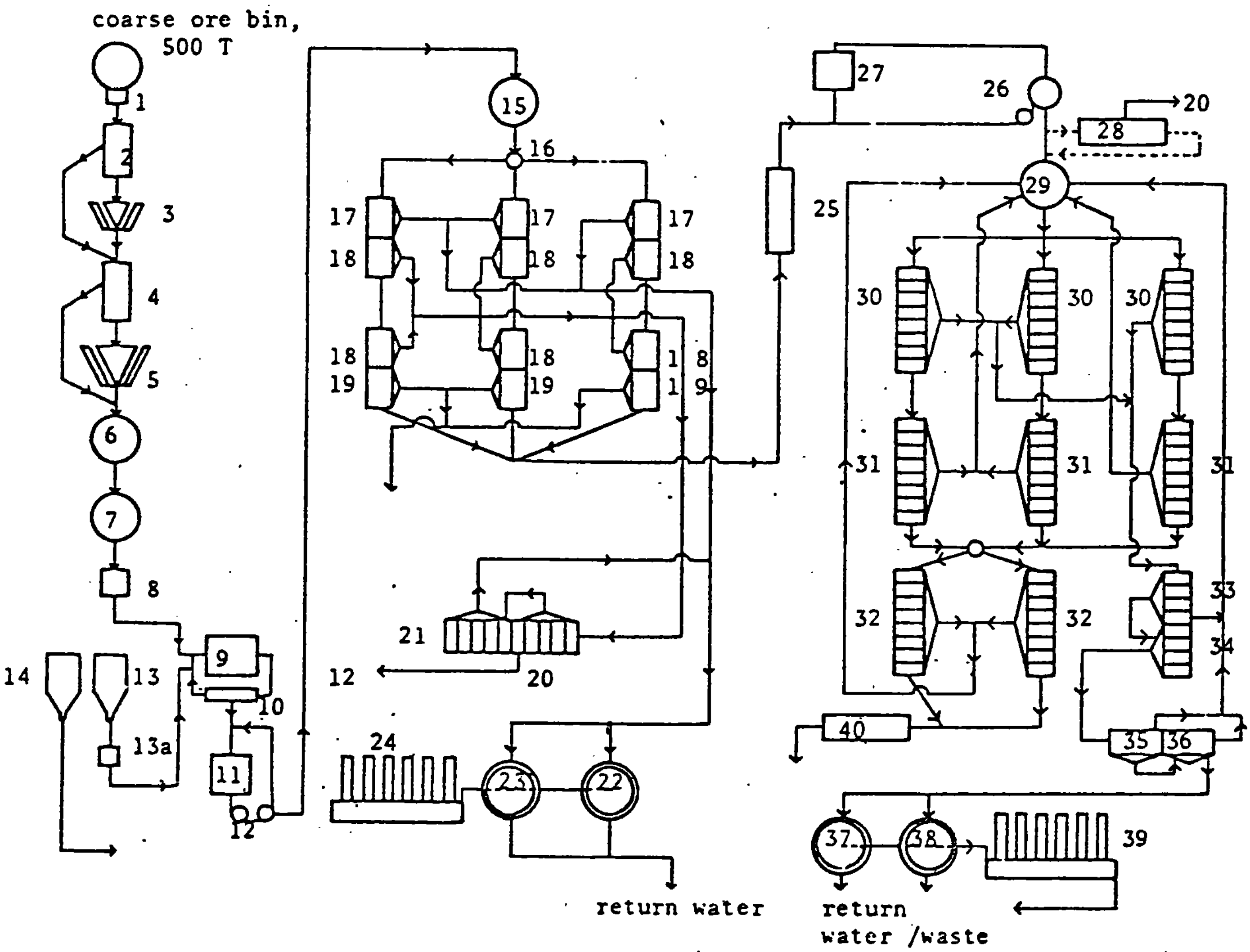
Table 1

Fig. 20

1. 1 - 36"x60" vibrating feeder
2. 1 - 4'x8' single rod deck screen
3. 1 - 4-1/4' standard cone crusher
4. 1 - 5'x8' single rod deck screen
5. 1 - 4' short head cone crusher
6. 1 - 32'x32' open steel ore bin
7. 1 - 32'x32' open steel ore bin
8. 1 - weightometer
9. 1 - 9'x12' rod mill
10. 1 - 54" classifier
11. 1 - 7'x11' ball mill
12. 2 - 20-B cyclones
13. 1 - 20'x18' elevated soda ash storage tank & pneumatic conveyor
14. 1 - bulk line storage
15. 1 - 8'x8' super conditioner
16. 1 - 5'x3-way distributor
17. 12- primary #24 roughers
18. 24- secondary #24 roughers
19. 12- #24 tertiary
20. 5 - primary #24 cleaners
21. 5 - secondary #24 cleaners
22. 1 - 30'x10' thickener
23. 1 - 30'x10' thickener
24. 1 - 9'x6' disc filter
25. 1 - 4'x30' blanket table
26. 2 - 20-B cyclones
27. 1 - 9'x10' ball mill
28. 6 - #24 scavengers
29. 1 - 8'x8' super conditioner
30. 24- primary #24 roughers
31. 24- secondary #24 roughers
32. 16- scavengers
33. 4 - primary #24 cleaners
34. 4 - secondary #24 cleaners
35. 4 - #24 tertiary cleaners
36. 4 - #24 final cleaners
37. 1 - 30'x10' thickener
38. 1 - 30'x10' thickener
39. 1 - 9'x6' disc filter
40. 1 - 4'x30' blanket table
41. 1 - tailings disposal pump

FLOW SHEET FOR ESTIMATING

MILL COSTS.



In 1974 Milton Lewis and Roshan Bhappu [25] estimated the capital necessary for constructing a flotation plant treating 40,000 tonnes per day of ore with a Cu grade of 0.60 %. The figures corrected on the basis of the M and S Mining and Ore Processing index for the 3rd quarter 1977 are as follows :

	10^3 US $
. Crushing	30,502
. Grinding	45,000
. Flotation	25,100
. Filtration	1,600
. Reagents (storage, preparation, distribution)	1,600
. Fallings disposal	7,700
. Buildings	2,700
. Miscellaneous and contingencies	16,300
. Operating expenses (3 months)	26,000
	156,500

The investment per tonne of run-of-mine ore treated in 24 hours thus amounts to US $ 3,900 and the operating costs are US $ 1.5 per tonne treated.

In a 1975 paper, Yaroll and Davis [26] gave estimates for flotation plants with small capacities (100 to 500 tonnes per day). The investments corrected for inflation and including the operating expenses are as follows :

Plant Capacity in t/d	Investments Readjusted for 3rd quarter 1977 (US $ 10^3)	Investments in US $ per tonne of capacity per day
100	1,535	15,000
200	2,500	12,500
300	3,300	11,000
400	4,050	10,100
500	4,700	9,500

The Yaroll and Davis estimate is based on the cost of the equipment necessary for a plant processing 200 tonnes per day. The extrapolation for the other capacities is based on the hypothesis that in the range in which plant capacity is increased by use of longer unit capacity equipment rather than duplication of units, the investments is a power function (with a 0.7 exponent) of capacity. In comparison with those used by other authors, Yaroll and Davis' method seems to exaggerate the ratio between total investments and equipment costs.

Operating Costs

Energy

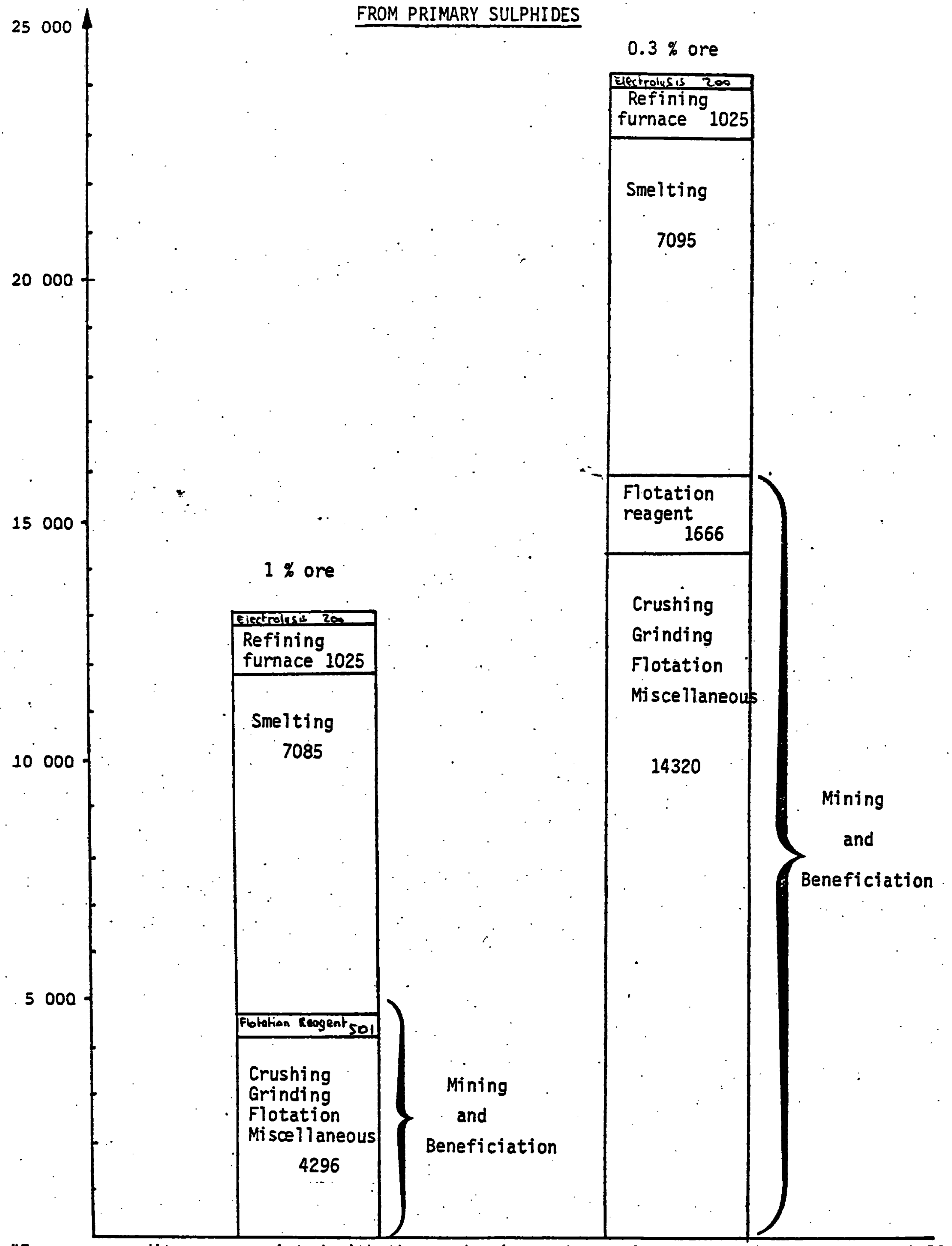
Energy consumption in a flotation plant varies in general from 20 to 50 kWh per tonne of ore. Crushing and grinding can account for about 10-20 kWh per tonne of ore.

The share of energy in the cost price of an ore processing plant is generally about 18 % in Europe for polymetallic ores.

Energy consumption for copper production from sulphide ores concentrated by flotation is indicated in the following figures (figures n° 21 and 22). [27]

ENERGY CONSUMPTION FOR PRODUCTION OF COPPER

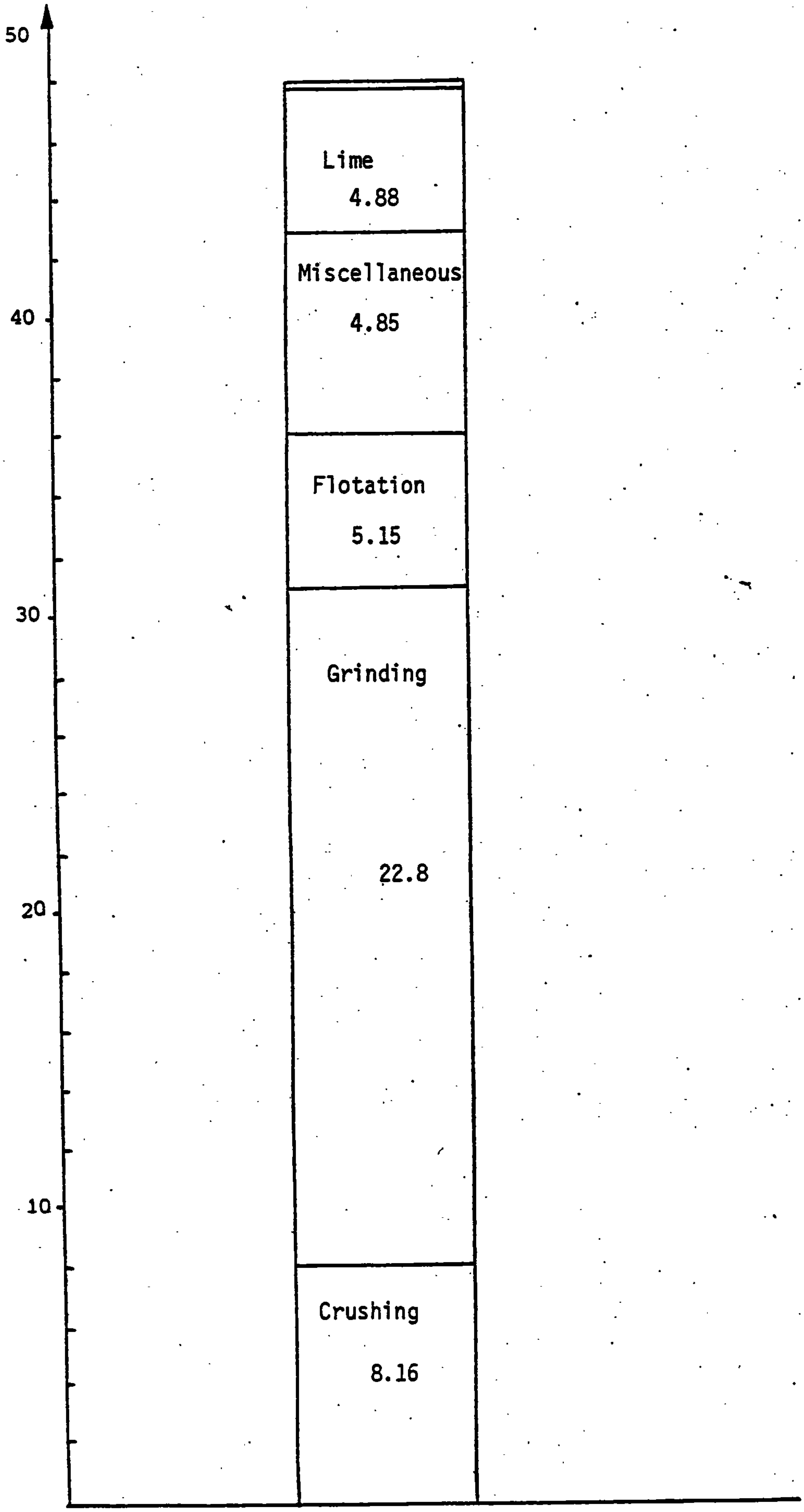
FROM PRIMARY SULPHIDES



Ref:"Energy expenditures associated with the production and recycle of metals" O.R.N.L. nov.1972

Figure n° 22

ENERGY CONSUMPTION FOR PROCESSING A 1 % Cu ORE



Ref :"Energy expenditures associated with the production and recycle of metals" O.R.N.L. nov. 1972

Labor

Manpower requirements per tonne treated in a flotation plant can vary to a considerable extent depending upon the size of the plant, its complexity (number of products to be separated, number of regrinding and thickening stages, ...) the layout of the plant and degree of automation (crushing, grinding, thickening and filtration can be run automatically without human intervention).

(continued page 3-76)

In a non-automated plant, having a small capacity, treating a simple monometallic ore the necessary manpower can be estimated as follows :

	Personnel on shifts	Personnel not on shifts	Total
Crushing	2 x 1	-	2
Grinding-Flotation	1 x 3	-	3
Thickening-Filtration	1 x 3	-	3
Tailings disposal	1 x 3	-	3
Analyses		1	1
Supervision	1 x 3	2	5
			17

Maintenance is generally carried out both by the personnel of the processing plant itself for the routine work and by the personnel of a central workshop for more complex operations. The annual cost (spares and labor) can be estimated at 6 % of the investment.

Supplies

Two categories of supplies can be distinguished :

In the first category are supplies which are consumed in proportion to the tonnage of ore treated (grinding materials, flotation agents, floculating agents) and their costs can be estimated on the basis of laboratory and pilot plant tests. A statistical study shows that in the United States in 1975 the cost per tonne treated for flotation and floculating agents was as follows :

. Simple copper ore 16 cents
. Copper-Molybdenum ore 11 cents
. Copper-lead-zinc ore 24.5 cents
. Copper-zinc-pyrite ore 60.4 cents.

In the second category are supplies which are estimated to be proportional to maintenance expenses but are not included in the latter (joints, pump fittings, oil and grease, etc.). Their cost is estimated to be 15 % of maintenance costs (Avies and Newton, 1955). [28]

In the large US plants the operating expenses are about US $ 1.5 to 2.0 per tonne of ore treated. They are distributed as follows :

% per Item		% of Direct Cost	
Crushing	15	Labor	34-38 %
Grinding	31	Grinding Materials and Crushers Spares	21-26 %
Transportation	3	Energy	18-22 %
Flotation	14	Reagents	11-14 %
Analyses	4	Miscellaneous	8 %
Water Removal	4		
Supervision	3		
Maintenance	20		
Miscellaneous	6		
	100		100

These expenses can be much higher if the ore is polymetallic and if the plant is small. At Funtana Raminosa in Sardinia the flotation cost in 1976 reached US $ 5.6 per tonne for an hourly capacity of 8 tonnes. The cost per tonne of copper content can then be calculated on the basis of a distribution of operating costs amongst the various metals.

Plant amortization which is added to these operating costs can have a great deal of influence if the reserves are small.

3.4.2.3 Economics of Mixed Processes

Mixed processes like the segregation process (thermal treatment + flotation) or the LPF process (hydrometallurgical phase + flotation) can only be used for relatively rich ores. This is true because the hydro-or pyrometallurgical stage costs are added to the physical separation investments and costs while producing a product requiring a refining stage. The segregation process for example consumes in its thermal stage 30 to 35 liters of fuel per tonne of ore, 20 kg of coal and 1 to 2 kg of sodium chloride per tonne of ore.

3.4.2.4 Investments and Operating Costs for Hydrometallurgical Methods

There are several recent studies on the economic evaluation of hydrometallurgical processes for the treatment of copper ores : Milton Lewis and Roshan Bhappu (1976) [18], Otto Sitnai and Paul Peeler (1976) [29] T.C. Argawal and al. (1974) [30].

We will present below the main conclusion of Milton Lewis and Roshan Bhappu concerning the application of hydrometallurgical processes, whether conventional (heap and vat leaching) or more recent (in situ leaching and agitation). In leaching, the case of heap leaching cementation and solvent extraction various modifications are distinguished.

In each case 4,500 tonnes per day of ore are assumed to be trated 330 days per year. The recoveries are as follows :

. Heap leaching .... 60 %
. Vat leaching .... 80 %
. Agitation leaching .... 90 %
. In situ leaching .... 60 %

Reserves amount to ten years of exploitation and two hypotheses are made as far as grades are concerned.

TABLES N°17 & 18

Estimated capital costs (in millions of dollars) for alternate processes. For the heap leaching situations, the ore is crushed through a nominal three-inch size using a two-stage crushing system.

Type of leaching / Recovery method / Copper percent	Heap Cementation		SX-EW		Vat SX-EW		Agitation SX-EW		In situ SX-EW	
	0.50	1.00	0.50	1.00	0.50	1.00	0.50	1.00	0.50	1.00
Mining	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	-	-
Crushing	1.6	2.6	2.6	2.6	3.6	3.6	3.6	3.6	-	-
Ore handling	0.6	0.6	0.6	0.6	0.8	0.8	0.3	0.3	-	-
Grinding	-	-	-	-	-	-	5.1	5.1	-	-
Leaching	0.6	0.6	0.6	0.6	3.3	3.3	3.0	3.0	0.8	0.8
Metal recovery	1.2	1.4	5.2	8.7	6.4	10.6	7.0	11.2	5.2	8.7
Supporting facilities	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2A	0.2	0.2
Total	7.5	7.7	11.5	15.0	16.6	20.8	21 5	25.7	6.2	9.7
$ per ton day*	1 660	1 710	2 560	3 330	3 690	4 620	4 780	5 710	1 380	2 160

Notes : (1) Cost of developing mine not included
(2) Cost of developing a tailings disposal area and equipment for handling tailing not included
* 45 kt/day mine capital is included

All direct operating costs per ton ore treated.**

Type of leaching / Recovery method / Copper percent	Heap Cementation		SX-EW		Vat SX-EW		Agitation SX-EW		In situ SX-EW	
	0.50	1.00	0.50	1.00	0.50	1.00	0.50	1.00	0.50	1.00
Mining *	1.050	1.050	1.050	1.050	1.050	1.050	1.050	1.050	0.175	0.175
Crushing	0.252	0.252	0.252	0.357	0.357	0.357	0.357	0.357	-	-
Ore handling	0.185	0.185	0.185	0.185	0.370	0.370	-	-	-	-
Grinding	-	-	-	-	-	--	1.063	1.063	-	-
Leaching	0.444	0.645	0.404	0.561	0.405	0.615	0.430	0.666	0.221	0.378
Metal recovery	2.393	4.731	0.571	0.898	0.675	1.086	0.731	1.176	0.530	0.856
Supervision	0.044	0.044	0.111	0.111	0.111	0.111	0.130	0.130	0.062	0.062
Administration	0.235	0.240	0.324	0.388	0.417	0.494	0.508	0.584	0.205	0.268
Total	4.603	7.147	2.897	3.445	3.385	4.083	4.269	5.026	1.193	1.739
Cost per kg of copper	1.534	1.191	0.966	0.574	0.846	0.510	0.949	0.558	0.398	0.290

Notes : * stripping ratio 1 : 1 - Ore and waste cost $ 0.45 per ton of material mined
** The M and S index is intended for the reevaluation of investments rather than operating costs. We have used it here because no other simple updating means is available.

Investments and operating costs were updated in 1977 by means of the M & S index (value 450 in the 2nd quarter 1975 and 524 in the 3rd quarter 1977).

The authors indicate the ore grades for which at a given copper price the various processes are economically interesting, in all cases for a tonnage treated daily of 5,000 tonnes. The results are as follows :

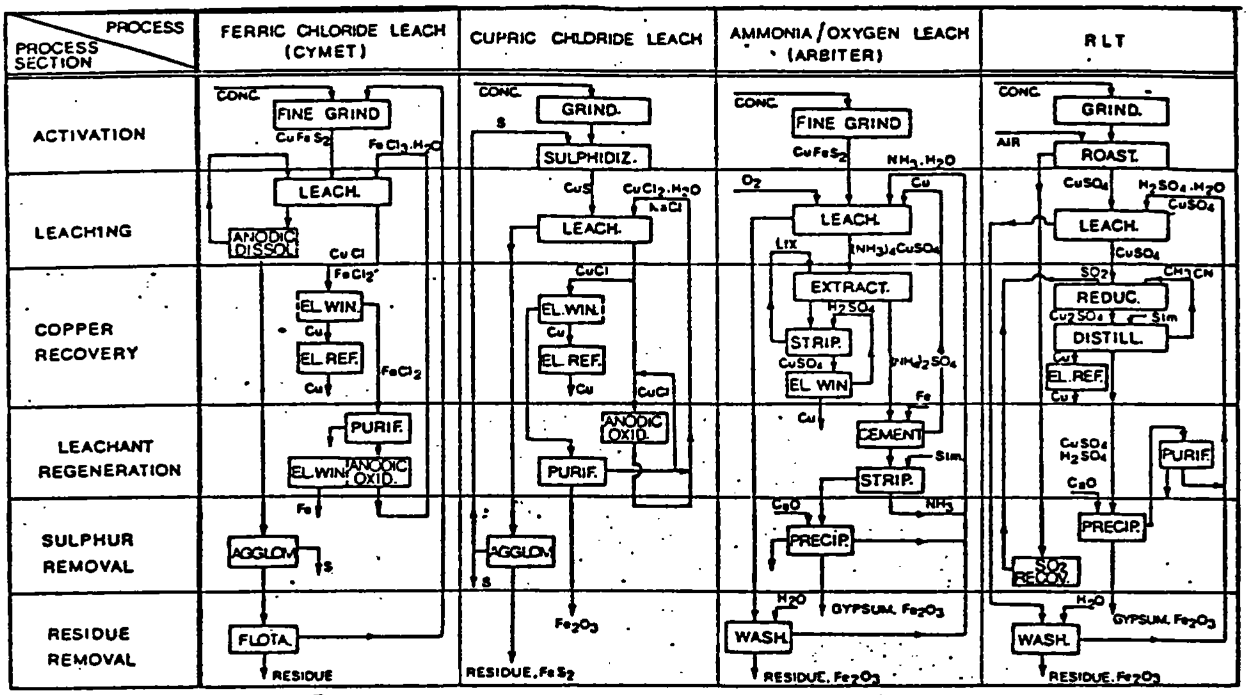
Acid-soluble Cu Ore Grade	Copper Prices $ 0.6 per pound	Copper Prices $ 0.8 per pound
0.3 - 0.5 %		(5)
0.5 - 0.6 %	(5)	(2) (3) (5)
0.6 - 0.8 %	(5)	(1) (2) (3) (4) (5)
0.8 - 0.9 %	(2) (3) (5)	(1) (2) (3) (4) (5)
0.9 %	(2) (3) (4) (5)	(1) (2) (3) (4) (5)

(1) : Heap leaching - cementation
(2) : Heap leaching SX - EW
(3) : Vat leaching SX - EW
(4) : Agitation leaching SX - EW
(5) : In situ leaching SX - EW

Sitnai and Peeler [29] compare four hydrometallurgical processes under development or already in operation for copper sulphide ores :

- Ferric chloride leaching (Cymet)
- Cupric chloride leaching (Duval, Imetal)
- The Arbiter process
- The RLT process (Roasting, Leaching Thermal disporportionation) developed by Parker in Australia which begins with sulphate roasting and solution in a sulphuric medium. After SO_2 reduction of the copper ions the latter are stabilised by addition of acetonitrile which is then distilled. This leads to the precipitation of copper metal.

Diagrams for the four processes are given below :



—Process block diagrams for the four selected hydrometallurgical processes.

Fig. 23

The investments updated in the 3rd quarter 1977 (M & S index) are as follows for the four processes (annual production of 50,000 tonnes of copper) :

Summary of the fixed capital costs (in 10^6 US $) for the four selected hydrometallurgical processes				
SECTION	Process			
	Ferric chloride leach	Cupric chloride leach	Ammonia oxygen leach	RLT
1. Activation	4.6	4.7	3.9	8.2
2. Leaching	25.7	5.1	4.7	4.7
3. Copper recovery		25.5	26.1	18.7
4. Leachant recovery	9.8		6.0	7.0
5. Sulphur recovery	3.9	3.0		
6. Residue treatment	2.1	1.2	1.9	1.9
7. Wirebar casting	3.1	3.1	3.1	3.1
Total fixed capital	49.2	42.6	45.7	43.6

TABLE N° 19

The total cost per kg of copper for each process is shown in the table below (operating costs were also updated in terms of the M & S index but penalties and bonuses based on raw materials prices have not been changed). For the sake of comparison, investments and mining costs for a modern pyrometallurgical plant at its optimal capacity of 100,000 tonnes/year are also given :

Calculated costs of the four hydrometallurgical processes in comparison with a modern smelter (Basis US $ 3rd Q. 1977)

	Ferric chloride leach (a)	Cupric chloride leach (a)	Ammonia and oxyden leach (a)	RLT (a)	Future pyromet. process (b)
Total fixed capital, (10^6 $)	49.2	42.6	45.7	43.6	
Specific fixed capital ($/tpy of Cu)	980	850	914	870	1 000
Processing costs (c/kg Cu)					
Process materials	0.5	2.0	14.2	7.9	1.2
Energy	8.4	5.2	4.7	3.8	4.4
Labour dependent costs	8.7	7.1	3.9	6.0	9.0
Capital dependent costs (3 %)	3.0	2.6	2.7	2.6	3.1
Operating cost	20.6	16.9	27.5	20.3	17.7
Depreciation (12 % cap.)	11.9	10.2	10.9	10.6	12.1
Indirect costs (30 % labour)	2.6	2.1	1.2	1.8	2.7
Manufacturing cost	35.1	29.2	39.6	32.7	32.5
Credits (c)	9.9				3.0
Penalties (d)	3.5	3.6	3.9	6.6	1.7
NNet manufacturing cost	28.7	32.8	43.5	39.3	31.2

Notes : (a) production capacity of 50 000 tonnes/year wirebar copper
(b) using improved technology at production capacity of 100 000 tonnes/year and sulphur dioxyde recovery greater than 98 %
(c) for electrolytic iron, sulphuric acid, and sulphur
(d) for copper losses and residue disposal

TABLE N° 20

The conclusion of the authors is that at the present time with the possible exception of the ferric chloride leaching process none of the four hydrometallurgical processes are competitive with pyrometallurgical processes especially if the concentrates contain appreciable quantities of precious metals. Under special circumstances, however, a specific process could be interesting. This is true in particular in the case of polymetallic concentrates where it is difficult by physical processing to separate copper from the other metals with good metallurgical results.

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4. METALLURGY

4. METALLURGY

4.1 Extractive Metallurgy by Pyrometallurgy

4.1.1 Introduction

At the present time pyrometallurgical methods are used essentially for sulphide concentrates with a copper grade of between 20 and 50 % depending upon the original mineral species.

In general, however, the average composition will be as follows :

. Cu	22-27
. S	28-32
. Fe	20-30
. Balance	gangue SiO_2, CaO, Al_2O_3, etc.

Three pyrometallurgical treatments are necessary for the passage from the sulphide concentrate to copper blister :

1 - Roasting
2 - Matte smelting
3 - Converting

We will briefly describe these operations

4.1.2 Roasting

4.1.2.1 General Remarks

This operation is made to obtain a product which when smelted will give a sufficiently high copper grade matte and to eliminate, if possible, certain volatile impurities. It eliminates part of the sulphur and transforms the pyritic part of the concentrate into iron oxides. The operation is not always necessary and depends both upon the kind of concentrate being treated and the metallurgical method used by plant. Some plants eliminate the operation or perform it at the same time as flash smelting. The essential object of roasting is to eliminate the excess sulphur which remains bound to the iron during smelting and can lead to the formation of low grade matte.

During smelting the following is eliminated on the average :

- . 50-60 % of the initial sulphur,
- . 20-30 % of the arsenic and antimony.

Working temperatures are between 300 and 800° C and the furnace atmosphere must be exactly adjusted to the impurities which are to be eliminated.

Roasting in a copper plant is always a partial operation since the copper must be kept in the sulphide state.

4.1.2.2 Roasting Methods

We will mention only the two furnace models which are still in operation : the multiple hearth furnace and the fluidized bed furnace.

4.1.2.2.1 The multiple hearth furnace is generally used in the older plants and has 10 to 14 superimposed hearths with the outside diameter varying from 5 to 8 m and total height of 20 to 25 m.

The tonnage loaded is about 300 to 350 tpd for average concentrate in a furnace of 7.6 m with 12 hearths. Thermally the operation is almost self sufficient.

It should be noted that the furnace is fed with a mixture of raw charge plus fluxes which has been homogenized and that the calcine goes directly to the matte smelter.

Few firms now construct this type of equipment (only 2 or 3 throughout the world).

4.1.2.2.2 Fluidized Bed Furnace

Modern roasting plants have in recent years used the fluidized system.

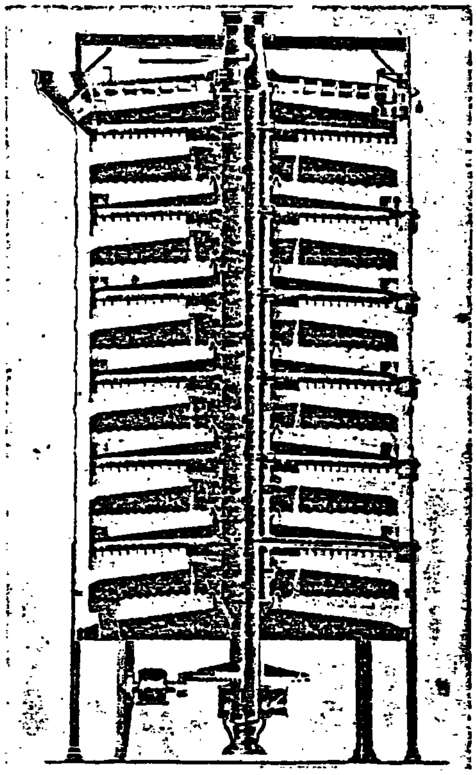


Fig. n° 24

MULTIPLE HEARTH ROASTING FURNACE

The furnace consists of two cylindrical refractory chambers connected by a conical element. It is fed with air by a hearth equiped with tuyeres whose design varies depending upon the manufacturer (a total of three or four in the world).

Figure n° 25 shows how the device operates.

A true solid-gas emulsion is created in the lower cylinder which is completely homogeneous in composition, temperature and specific weight.

The gases drive 30 to 35 % of the calcine while the rest overflows from the furnace.

The general processing conditions are :

. Specific capacity for dry raw concentrate : 50 to 60 tpd and m^2 ;

. Temperature : 500-800° C.

A hot cyclone and an electrical precipitation unit make it possible to recover the rest of the calcine and all the latter is sent for smelting.

A definite tendancy can be observed for abandoning roasting in the new plants because much dust is produced which is expensive to recover and most of which must be recycled. There are also smelting problems.

In addition, except in the case of pyritic concentrates, the copper grades of the charge are higher and there is a tendancy towards a sulphur deficit. In this case there is no longer any reason for roasting.

4.1.3 Smelting

4.1.3.1 Principles

The object is to separate the metallic sulphides of the concentrate or calcine from the gangue. This is

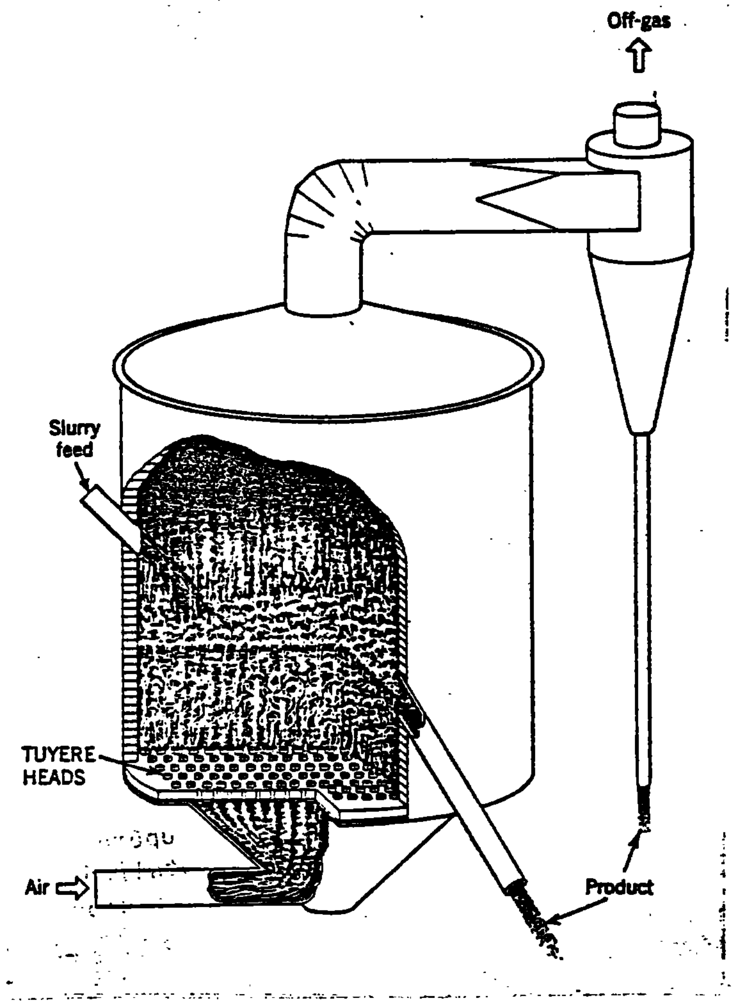


Fig. n° 25

done by smelting with fluxes adapted to the nature of the gangue. The temperature is approximately 1 250° C. Whatever kind of furnace may be used, the smelted material separates into two layers i.e. slag and matte.

The main reactions are as follows :

. $2CuFeS_2 + O_2 \longrightarrow Cu_2S + 2FeS + SO_2$

. $2Cu_5FeS_4 + O_2 \longrightarrow 5Cu_2S + 2FeS + SO_2$

. $2CuO + 2\ FeS_2 \longrightarrow Cu_2S + FeS + SO_2$

. $FeS_2 + 5Fe_2O_3 \longrightarrow 11FeO + 2SO_2$

. $SiO_2 + 2\ FeO \longrightarrow SiO_2\ ,\ 2FeO$

According to Ruddle the basic components of the matte are :

Cu_2S, FeS, FeO, Fe_3O_4.

Slag from copper smelting consits essentially of fayalite SiO_2, 2FeO and tridymite accompanied by small quantities of aluminium and calcium silicates.

The following tables show some typical compositions for mattes and slags :

Mattes	Cu %	S %	Fe %
Reverberatory furnace 1	35.8	24.4	31.9
Reverberatory furnace 2	41.0	24.9	30.9
Electric smelting	55.6	25.3	20.3
Flash smelting	50.5	22.3	22.5

Scories	Cu %	SiO_2 %	FeO%	CaO%	Al_2O_3%
Reverberatory furnace 1	0.38	35.7	46.9	7.9	3.6
Reverberatory furnace 2	0.41	38.3	47.4	6.2	4.8
Electric smelting	0.52	38.8	34.9	14.9	4.8
Flash smelting	1.65	27.1	56.7	0.51	6.8

The modern practice is to attempt to obtain mattes which are as rich as possible in copper so as to reduce converting time. In certain cases it is possible to reach a copper content of 65-70 %.

4.1.3.2 Smelting Methods

4.1.3.2.1 The Reverberatory Furnace

This furnace which consumes a great deal of fuel is still found in most plants but will be progressively replaced by more efficient furnaces. This change is a slow one because of the size of the investment (new units only) and the reverberatory furnace must still be taken into account.

These furnaces are described by many authors and we will give only the characteristics of recently constructed furnaces (about 5 to 6 years old).

Characteristics	Units	N° 1	N° 2
Inside Length	m	33.6	33.6
Inside Width	m	9.7	11.00
Hearth Side Height	m	3.7	4.00
Gas Outlet Side Height	m	3.4	3.4
Depth of Bath	m	0.8	1.1
Wall Thickness :			
. Side Walls	m	0.970	0.970
. Hearth (bottom)	m	1.750	1.750

The modern materials are chrome-magnesite brick for the side walls and arch while the floor consists of a prisé of chromite over a layer of poured slag.

Processing capacity varies depending upon whether the charge is roasted or raw and whether the heating methods has enriched air hearths or not.

Charge	Combustion	Dry Raw Capacity (tpd)	
		Minimum	Maximim
Raw	Normal Air Enriched Air	800 1 000	900 NA
Roasted	Normal Air Enriched Air	1 200 NA	1 300 NA

Note : An average concentrate was considered to have a grade of 25 % Cu for the large furnaces i.e. at least 33 m x 10 m.

In practice, an air enriched modern furnace fed with 25 % Cu raw concentrate can produce 95 to 100 kt/y of blister at a 98 % efficiency.

The main consumptions are :

. Fuel bunker C   450 to 550 kg/t of metal
. Oxygen (85 %)   75 to 80 m^3/t of metal.

The furnace also produces 2.8 to 3.8 t/h of steam with a waste heat boiler at 40 bars and 400° C.

As an example, we give below a simplified material balance for a modern furnace for a tonne of dry concentrate :

Input	Weight	% Cu	Content
Concentrate	1,000 kg	30,3	303 kg
Conv. Slag	458 kg	4.5	20.6 kg
Total			323.6 kg

Output	Weight	% Cu	Content
Matte	782.3	41.0	320.7 kg
Slag	521.3	0.41	2.6 kg
Smoke	3.4	15.5	0.5 kg
Total			323.8 kg

4.1.3.2.2 Flash Smelting

This process was first developed in Finland. After 1945, this country was in cheap electricity shortage for furnaces and the Outokumpu Co. first used this process at Harjavalta.

The principle is to feed a homogeneous mixture of fine and dry concentrate with fluxes at the top of a shaft by means of air preheated to 500-600° C (cf Figs. 26 and 27). The sulphides are roasted and smelted in one operation during their passage down the shaft in accordance with the reactions below, for example :

$$4\ CuFeS_2 + 5\ O_2 \longrightarrow 2\ Cu_2S,\ FeS + 4SO_2 + 2FeO$$

producing matte. Below the shaft is a furnace with a hearth like that of a reverberatory furnace which collects the smelted products and forms slag through the action of silica on wüstite :

$$SiO_2 + 2\ FeO \longrightarrow SiO_2,\ 2\ FeO$$

Settling is carried out and the matte separates from the slag. The bottom of the furnace thus acts as a settler.

The gases separate from the smelted material and dusts in another shaft which leads to the heat recovery and gas treatment equipment.

This process is almost thermally self-sufficient and is well adapted to mixed chalcopyrite and pyrite concentrates because of the great heat of reaction of the pyrite.

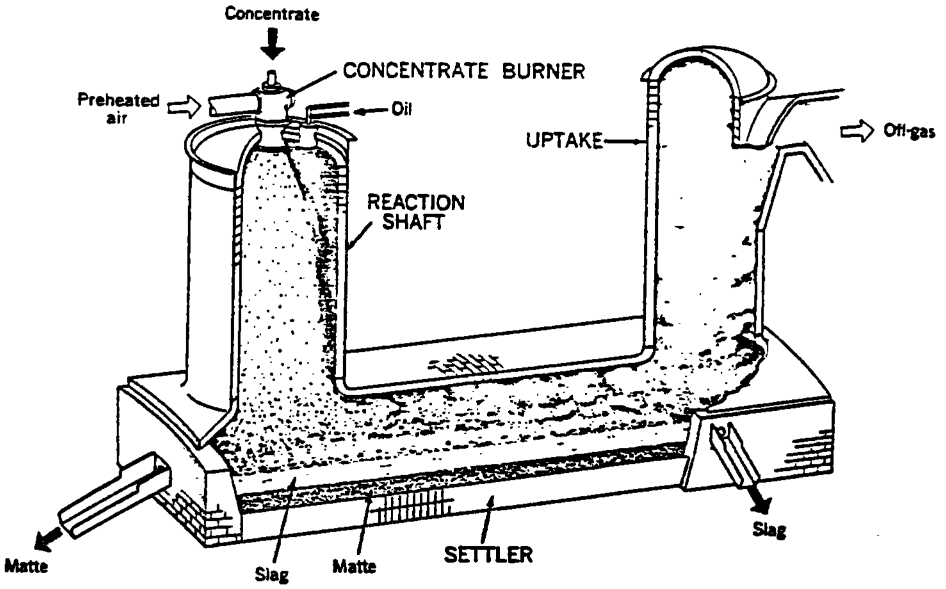


Fig. 34. Cutaway view of Outokumpu flash smelting furnace.

Fig. n° 26

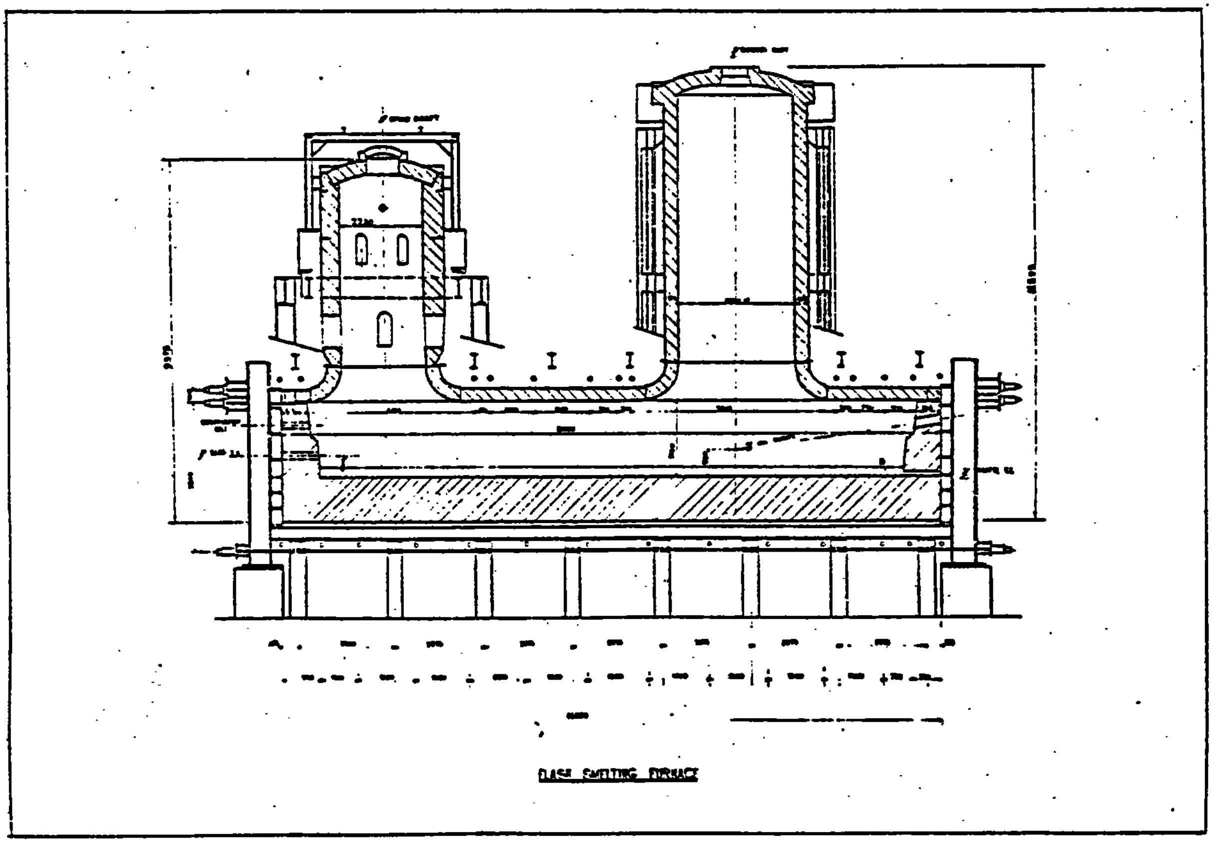


Fig. n° 27

By controlling the air/concentrate ratio it is possible to adjust for the copper grade of the matte and in certain runs it was even possible to obtain blister directly from the furnace. A matte with a 65 % copper grade is normally obtained from a 22 % concentrate.

Flash smelters have large processing capacities within the range of about 200 tpd to 2,000 tpd.

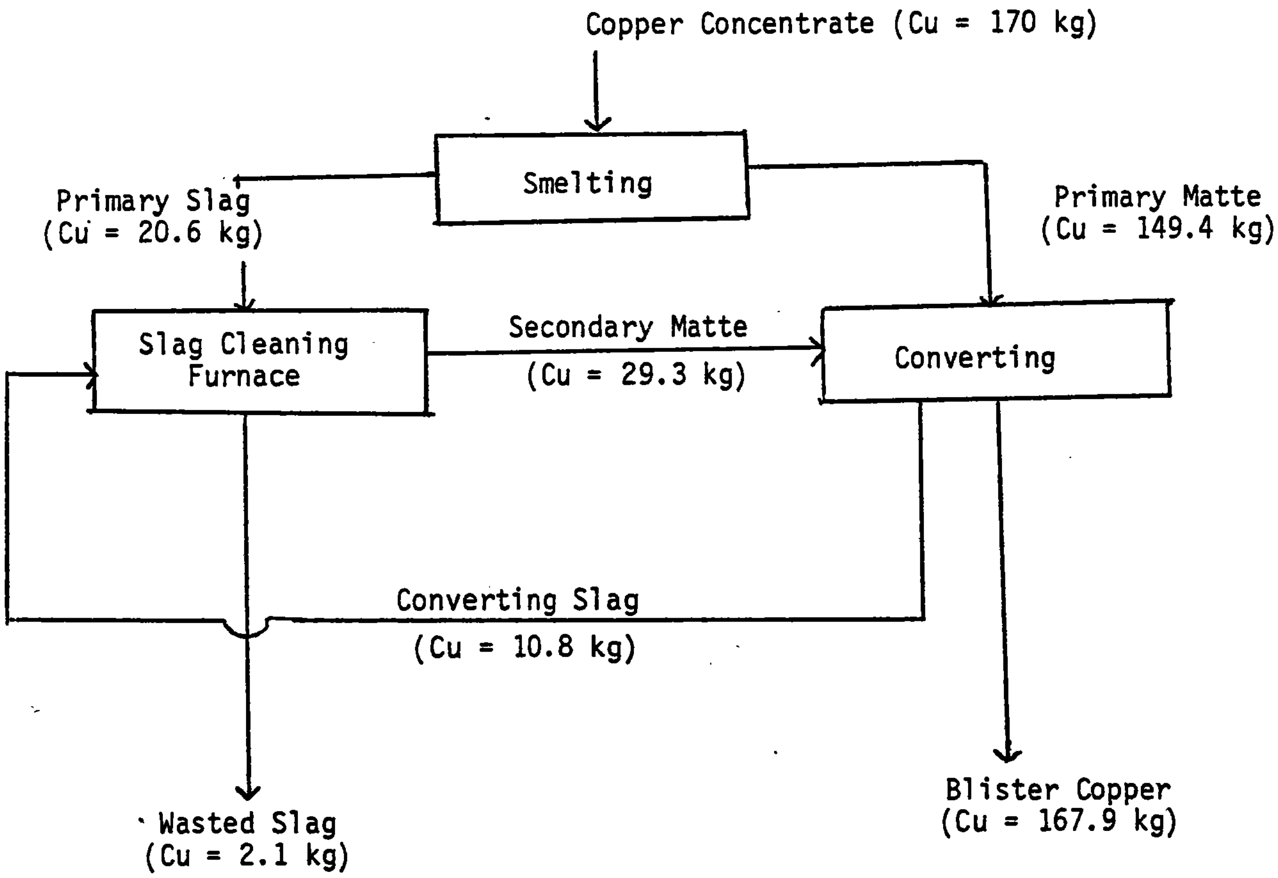
The building materials are the same as those used in reverberatory furnaces and the annual working time is about 340 days.

Thermal economy is much better for the flash smelter since for a furnace with a capacity of 40 t/h of concentrate (960 t/d) and with air preheated to 450° C with fuel oil, 1.2 t of excess steam (at 40 bars) are produced per tonne of concentrate. 39 kg of fuel oil must be added per tonne of concentrate. The advantage over the reverberatory furnace is very large (it should be recalled that the latter consumes 138-150 kg of fuel oil per tonne of concentrate with a Cu grade of 25 %.

At the present time flash smelting furnaces have been tested industrially and are the best solution for problems of energy economy, recovery of sulphur in acid form and high grade matte.

However, flash furnaces give a rich copper slag (2 to 3 %) which has to be reprocessed either by electric smelting or by flotation in order to give an acceptable material balance.

Hereunder is summarized a material balance with slag reprocessing by electric smelting.The concentrate is a poor one (17 %).



4.1.3.2.3 Electric Smelting for matte is of interest if hydro-electric energy is available. Only Sweden and Norway are in this position in Western Europe.

Hereunder are given the characteristics for the largest furnaces currently in operation (not necessarily in Europe) :

	1	2	3
Power (MVA)	51	36	12
Number of electrodes (Soderberg)	6	6	6
Length (Inside)	38.4 m	31.4 m	23.5 m
Width (Inside)	11.6 m	10.36 m	6.0 m
Height (Inside)	6.4 m	4.8 m	3.5 m
Annual Blister Capacity (t/y)	125,000	182,000 (1)	80,000

(1) Special case : The concentrate is very rich.

Electricity consumption is about 400-500 kWh/t of concentrate.

Cu content of matte are within 40-60 % range.

Another advantage of the electric furnace is the small volume of gas produced and therefore the gas cleaning operations are very easy.

4.1.4 Converting

4.1.4.1 Principle

So-called reverb matte essentially contains Cu_2S, FeS, FeO, Fe_3O_4.

Converting makes it possible to pass from this smelted mixture to the almost pure blister metal by means of selective oxidation of iron which is then transformed into silicate. The following reactions are therefore involved :

. $2FeS + 3O_2 \longrightarrow 2FeO + 2SO_2$

. $Cu_2S + O_2 \longrightarrow 2Cu + SO_2$

. $3Fe_3O_4 + FeS \longrightarrow 10FeO + SO_2$

. $2FeO + SiO_2 \longrightarrow SiO_2 + 2FeO$

The best solution would be to obtain the magnetite reduction, this compound being source of troubles in the smelting furnace.

The operation consists of two phases :

The first consists of passing from matte to copper sulphide Cu_2S. The iron is eliminated by oxidation and slagging.

The second phase consists of passing from Cu_2S to copper blister.

4.1.4.2 Equipment and Method

The Peirce-Smith type converter is the most widely used. It consists of a horizontal cylinder with a diameter of 3 to 4 m and with a length of 5 to 9 m (see Fig N°28).

The present standard is 13' x 30' (3.96 m x 9.14 m) with a tendancy to become 13' x 35' (3.96 m x 10.60 m).

A row of 48 to 52 tuyeres of 40 x 49 mm is fitted to the converter and fed with air at about 1 bar by means of a mainpipe fixed in parallel to the shell.

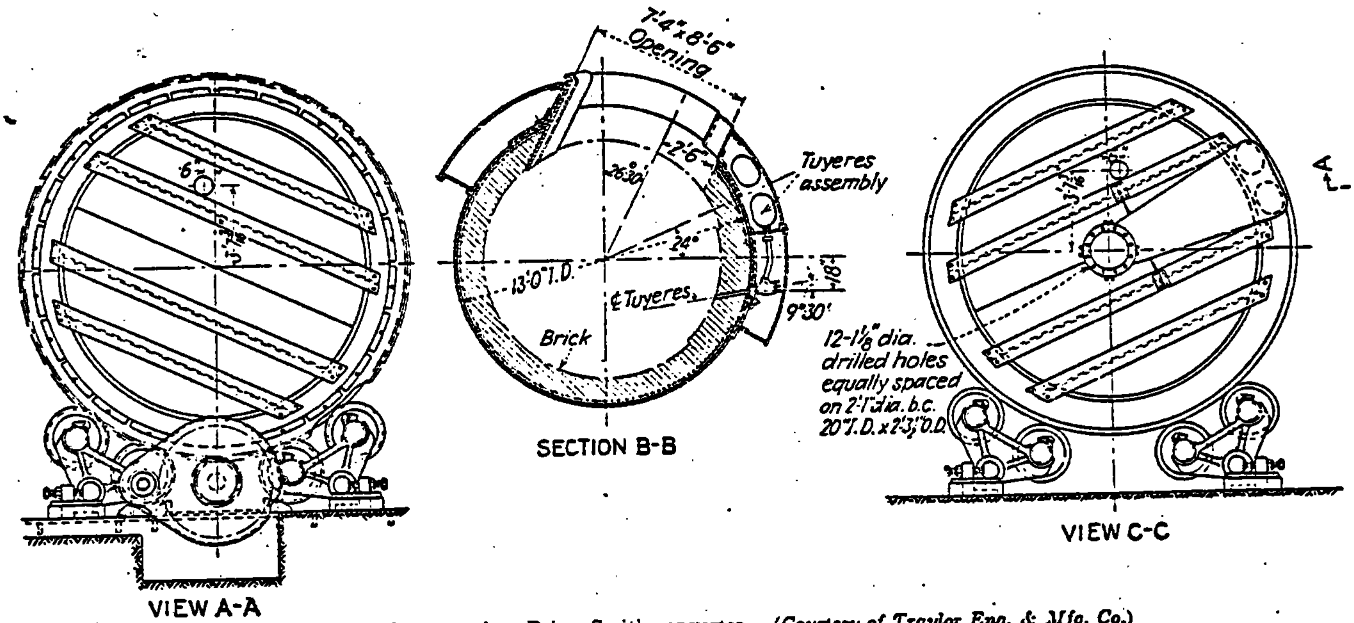


FIG. 3a.—Cross section, Peirce-Smith converter. (*Courtesy of Traylor Eng. & Mfg. Co.*)

Fig n° 28

The most part of the lining consists of chrome-magnesite bricks (thickness 230 mm on the arch, 470 mm at the tuyeres).

The lining life is very short and if the plant equipment consists of two or three converters of course one of those is in relining position.

Consumption of refractory material is from 2 to 4 kg per tonne of copper.

The operation takes place in two stages :

First , several ladles of black matte are loaded. These are successively blown with the addition of silica or siliceous copper ore until the quantity of copper corresponding to a tap (70 to 100 t depending upon the plant) is obtained in the form of Cu_2S (white metal). The furnace is deslagged before loading each matte ladle.

After this period copper blast is done for about 2 hours and blister is poured.

The blister characteristics depend upon the impurities in the concentrate.

The following table gives some analyses :

Blister

Cu %	Fe %	S %	Bi %	As %	Sb %	Pb %	O_2 %
98 to 99.6	0.05 to 0.2	0.05 to 0.2	0.002	0.001-0.2	0.001-0.2	0.01-0.2	0.05 to 0.1

Converter Slag

Cu %	FeO %	Fe_3O_4 %	SiO_2 %	MgO %	CaO %	Al_2O_3 %	S %
1.5 to 4.5	40 to 60	15 to 20	20 to 30	1	2 to 10	1 to 7	0.5 to 3

4.1.5 New Smelting Processes

We will consider these processes here since in certain cases they integrate smelting and converting and can go till to the metal. These processes have the following characteristics in common :

. The operations are as continuous as possible ;

. Oxygen enriched air, and sometimes pure oxygen, is used leading to more intense reactions with energy economy and the production of high SO_2 content gases. These ones can be used easily for acid production or sometimes even sulphur ;

. The products are generally higher in copper and it is sometimes possible to obtain blister directly ;

. The slag must be reprocessed in a separate unit (hot or cold) in order to have a copper balance as good as conventional plant balance ;

. Computer control on line is generally used ;

. The investment is smaller than for conventional processes ;

. Labor saving is important.

Some processes have been accepted industrially while there have been trouble with other ones.

The table below summarises the situation :

Process	Situation
WORCRA	Pilot plant in Australia with capacity of 72 tpd of 24 % Cu concentrate.
NORANDA	Pilot plant in Canada with capacity of 100 tpd of concentrate. Plant in 1973 for 800 tpd of concentrate. Project for Kennecott ar Garfield with 3 furnaces with 1,600 tpd of concentrate capacity.
MITSUBISHI	Pilot plant in 1961. Furnace for 4,000 tpm of blister at Naoshima. Project for Texas Gulf at Timmins (Canada).
T.B.R.C.	Pilot plant at Port Colborne (Canada). Project for Afton Mines (British Columbia). 28 ktpy of Cu.

As far as we know no smelting plant using these processes is projected for Europe.

4.1.6 Blister Refining

The electrical properties of copper (its major use) require careful refining and certain impurities must be eliminated. Hereunder is a typical analysis in ppm.

As	Sb	Bi	Pb	Ni	Te	Se	O_2	S
4-10	5-10	< 2	5-10	<15	4-8	2-10	<300	15

4.1.6.1 Fire Refining

This process is less and less used.

The blister smelted in a reverberatory furnace is subjected to air oxidation in order to volatilise the oxides of certain metals (As, Sn, Sb, Bi, Pb) or to produce a slag of the oxides of non volatil metals. The slag is removed.

In a second phase, the practice consists of a reduction by poling with green wool poles in order to transform the cuprous oxide Cu_2O to the Cu state. The operation takes place at 1,100° C.

Various shapes of cast are made :

1) Wire bars of 60 to 350 kg ;
2) Plates,
3) Billets for pipes (from 50 to 300 mm in diameter),
4) Anodes for electrolytic refining if this operation has to be made after fire refining.

4.1.6.2 Electrolytic Refining of Copper

An electrolytic copper refining is a soluble anode process. The selective chemical action of the current in this process gives the copper a higher purity than through fire refining. The process can start with a metal which has already been pyrometallurgically refined and has a high Cu content.

4.1.6.2.1 Behaviour of Copper Impurities in Electrolytic Selection

The electrolyte is an acid $CuSO_4$ solution (analysis : 35-45 g Cu and 140-200 g H_2SO_4 free per liter). The continuous current passing through this solution with a tension between anode and cathode of 0.15-0.3 V acts differentially on the following four groups of impurities :

a) Zn, Fe, Co, Ni, Pb, Sn are electropositive with respect to Cu and dissolve with it in the bath. Only Sn as an arseniate or sulphate and Pb as sulphate pass into the anode slimes. The others which are also attacked by H_2SO_4 remain in solution and can then be highly concentrated without impeding copper deposition the cathode.

b) Ag, Au, Pt are electronegative. Au and Pt are not dissolved and remain in the slimes. The same is true for Ag except if there is sufficient acidity. Slight mechanical driving is to be feared unless the circulation is slowed down.

c) Cu_2O attacked by H_2SO_4 gives half its copper to the solution and half to the slimes. Cu_2 (Se, Te), Cu_2S, Ag_2 (Se, Te) are neither electrolyzed nor attacked and go into the slimes.

d) As, Sb, Bi which are very undesirable are partially dissolved and partly pass into the slimes in the form of basic arseniates or sulphates and are partly deposited with the copper. For example, when the As content is under 0.3 % in the anode all As is precipitated by other impurities mainly Sn. When the As content is over 0.3 %, 60 to 70 % of the arsenic is dissolved. If As is allowed to concentrate in the bath to a content of more than 1.25 % too much acidity will be lost and the As will contaminate the cathode.

A partial remedy is to add 0.015 % glue (45-60 g/t Cu) and 0.01 % Cl to the bath in the form of hydrochloric acid. Sb is less directed towards the cathode while Bi on the other hand has a greater tendancy

4.1.6.2.2 Arrangement of Electrodes

The multiple arrangement is characterised by parallel feed to the electrodes in the tanks. Each current to an anode passes through the bath to the neighbouring cathode. This arrangement is frequently used mainly in the Walker system. The rows or blocks of joined tanks are fed in series.

4.1.6.2.3 Operating Conditions for the Process

Electrolyte

This is the principal resistance of the circuit. Its composition, temperature and circulation must be carefully controlled.

Composition : the conductivity of the bath improves as the percentage of H_2SO_4 rises (with limits due to risk of crystallisation), and as the sulphate contents fall (but 30 g Cu/l is the lower limit for a good deposition since below this level As is more intensely directed towards the cathode). Therefore during the operation it is necessary to compensate the acid content which slows copper solution and the impurities of group a) and to bleed and clean the electrolyte.

Raising the temperature improves conductivity and deposition but is not economical above 55-60° C. The additional heat over 30-35° C (temperature reached by Joule effect) must come from heating (steam at 1 kg) which is expensive and at 60° C evaporation reaches 110 kg per m^2 of bath per 24 h. A loss of 5-10° C in the flow from block to block of tanks should be noted.

Diagonal circulation from above (tank inlet) to the bottom (tank output) makes it possible to homogenise the bath since $CuSO_4$ which forms at the anode is heavier than the regenerated acid at the cathode. Circulation should increase with the current density, temperature and arsenic content but it sould fall when the percentage of gold and silver increases. If the circulation is too intense, it increases copper losses (bath) and especially silver and even gold losses towards the cathode since it then impedes the decantation of the slimes. All lead siphon tubes on the outlet side should be insulated (glass joints and supports). The entire circulation from the heated upper electrolyse reservoir usually passes through the tanks in one cascade.

4.1.6.2.4 Electrical Energy, Current Density, Resistance Voltage, Efficiency

The choice of current density in A/m^2 of cathode determines the number of tanks, their arrangement and the size of the tank house. It depends upon the cost of energy and the composition of the anode copper. A high arsenic or silver and gold content requires a low current density which is correlated with less intense circulation. In multiple connection the high current intensity leads to large investment in busbars. In addition, within the limits of 160-225 A/m^2 the 10 to 14 day life of the cathode is acceptable and the anode gives two cathodes.

The overall resistance of the circuit breaks down as follows : contacts 15 %, electrolyte and transfer 65 %, conductors 10 %, back electromotive force 5 %, slimes and miscellaneous 5 %.

Current efficiency depends upon leaks and short-circuits (pieces of anodes broken off, new badly fixed cathodes, cathode excrescence which can be reduced with glue.

4.1.6.2.5 Anodes, Cathodes, Tanks

In the multiple processes the anodes are easily cast with their two eyes. Their thickness vary from 32 to 40 mm.

For the cathodes, an expensive procedure is necessary involving the preparation of starting sheets in special tanks called strippers equiped as follows : the electrolyte is less concentrated and more purified. The stripper anodes are larger and alternate with pure copper starting sheets with a thickness of 3-5 mm and are coated with graphite oil. The starting sheets formed in 24 hours with a thickness of 0.5 to 0.6 mm are then stripped from the original sheets and shaped and attached to their loops and suspended to copper rods arranged between the anodes of the commercial tanks.

The axis to axis distance between the anodes is from 100 to 115 mm.

The tanks are frequently no longer made of wood but of reinforced concrete elements assembled with tie rods. The inside walls are lined with antimony lead (6 % Sb), and, more recently, with plastic sheets (mainly polyvinyl chloride). They stand on 350 x 350 mm brick pillars at the basement level. Over each one there is an insulating glass plate and a protective lead cover. Copper containing impurities of up to 1.2 % and with up to 30kg/t of Au + Ag can be treated.

4.1.6.2.6 Purification of the Electrolyte

The solution is progressively building up with As, Sb, Ni, Co, Fe, Bi and almost always with Cu except if a very impure copper is being treated.

The maximum concentration of impurities which is adviseable to observe in electrolytes are shown in the following table :

As	Sb	Ni	Fe
1.2 %	0.10 %	1.2 %	1.10 %

Considerable quantities of electrolyte are then bled (1-2.5 % of circulation volume) with a frequency which varies depending upon the degree of purity of the anode copper. Purification takes place either by crystallization or electroprecipitation or by combining the two methods.

a) Crystallization

The liquor is crystallized (H_2SO_4 content reduced to 1 %) by flow over copper granules and is then concentrated to 43° (Baumé) by indirect heating (steam) for the crystallization of 80-90 % of the copper content in the form of $CuSO_4$, 5 H_2O, blue vitriol. This product is sold or is used for the preparation of replacement electrolyte. If the liquor contains little or no Ni which is the only valuable impurity after Cu, it can be wasted after the precipitation by Fe (scrap) of the remaining copper which is very contaminated and is sent to the matte smelting furnace.

b) Electroprecipitation

This is electrolysis with insoluble anodes used either only for reducing the percentage of Cu in the electrolyte or for removing all Cu and As, Sb before separating if necessary the other impurities. The tanks (2-5 in series) are of the commercial type with hard lead anodes, copper cathodes (starting sheets) and 300-350 A/m^2. Most of the copper is separated in the first tank and is ready for the wire bar furnace while in the others the rest of the copper is deposited with most of the As and Sb (80-93.3 % and 90-95 % respectively). This is done with relatively little circulation but at a low current efficiency i.e. an average of 20 % for 75 % in the first tank and 1 % in the last one. The arsenical copper obtained goes to the smelter. The liquor now containing only Ni, Fe, Co, Bi is subjected to intense concentration followed by crystallization for Ni recovery and so as to allow reuse of the acid mother liquor. If the Ni content of the liquor is low it is wasted.

Crystallization requires an expensive plant. Electrolysis on the other hand involves simple equipment but is consumes much energy. The two methods are usually used together. Electrodeposition alone is sufficient when the anode copper is not very impure.

4.1.6.2.7 Principal Characteristics of Modern Refineries

Currently, plant capacity is increasing and there has been technical progress in the field of mechanisation so as to reduce overhead especially as concerns labor.

As concerns the capacity of electrolytic refineries in Western Europe, the following can be mentioned :

. Olen	330 kt/y
. Hamburg	240 kt/y
. Huelva	105 kt/y
. Lünen	95 kt/y

As concerns progress in the automation field :

- Automatic anode casting (Outokumpu process) makes it possible to produce 300 kg anodes with a precision of 1.5 % at the rate of 100 t/h ;
- Automatic preparation of cathode starting sheets from stripping to locating of cathode groups in the cell.

As concerns <u>technical advances for the process</u> :

- Current density has been increased from 200 to 350 A/m^2 through the application of the per odic current inversion technique developed in Bulgaria.
- The weight of the anode has been standardized through automatic casting.
- Faster circulation of the electrolyte in the tanks improves bath homogeneity as concerns concentration and temperature.

4.1.7 By-Products of Copper Metallurgy

The by-products can be divided into three categories in accordance with the flow sheet for extractive metallurgy :

. Sulphur,

. Metals produced during pyrometallurgical operations,

. Metals produced during electrolytic refining.

4.1.7.1 Sulphur and Sulphuric Acid

Gases with a sufficiently high SO_2 concentration can be produced depending upon the techniques used in the pyrometallurgical plants and, in this case, it is possible to speak of sulphur recovery especially in the form of sulphuric acid.

Reverberatory furnaces and converters give a gas generally too diluted to allow economic recovery. If there were no environmental constraints the metallurgists would simply release these gases into the atmosphere after removal of the dust through stacks chimneys with sufficient height to disperse the SO_2.

For good sulphur recovery it is therefore necessary either to improve the old plants or to modify completely the plants with processes which are best adapted and industrially effective up to the present time. These are the following :

- For roasting — Fluidized bed method
- For smelting — Flash smelting
- For converting — Siphon converter (Hoboken)

The SO_2 content of the g s from the smelter is then between 7 and 10 % on the average and the contact process can be applied for the production of sulphuric acid under the best possible conditions.

We will not discuss this point further except to note that on the average a copper plant producing 100 kt/y of metal with conventional concentrates can produce up to 430 kt/y of pure sulphuric acid.

4.1.7.2 Metals Produced during Pyrometallurgical Operations

Copper concentrates are always relatively contamined by the following elements :

- Lead, zinc, bismuth, cadmium, arsenic, tin, antimony, selenium, tellurium, gold, silver, platinum.

The table below gives an idea of the distribution of these elements in the principal kinds of metallurgical operations :

Distribution of Impurities in %

	Smelting			Converting		
	Matte	Slag	Smokes	Blister	Slag	Smokes
Pb			90			
Bi	7	7	86	4	1	95
Sb	30	54	16	27	23	50
Su	-	-	-	-		
Cd	-	-	-	-		
Zn		95		-	-	
As	34	54	12	16	11	73
Se			60	50		50
Te			60	50		50
Au,Ag,Pt	100			100		

It should be noted that, essentially, bismuth, lead, arsenic, antimony, selenium and tellurium are found in the smoke. The treatment of smoke in copper metallurgy is an important part of the investment and operating costs.

It consists basically in collection with all available technical means and recycling at definite points of the circuit so that concentration of one or several elements finally occurs.

The circuits are quite complex and involve thermal or hydrometallurgical operations which are not, however, performed everywhere (some smelters do custom processing or by-products). The final products are as follows :

- Bismuth metal,
- Cadmium,
- Antimony, sometimes alloyed with lead,
- Selenium,
- Tellurium.

Lead is generally recovered by lead plants.

Arsenic which was an interesting by-product is tending to become quite undesirable because the market is no longer capable of absorbing production and there are problems involved in storage in a form which is not harmful to the environment.

4.1.7.3 Precious metals, gold, silver, platinum, etc. are found almost 100 % in the copper blister and during electrolytic refining these elements are found in the anode slimes which are recovered both by washing the anode skeletons and the bottoms of the electrolysis cells. These slimes contain all the gold and silver and sometimes Sn and Zn (if the plant resmelts miscellaneous copper scrap). Lastly, there is much copper i.e. 20-50 %. This tonnage, 0.4-2 % of the anodes, and their composition vary a good deal depending upon the nature of the anode copper.

After screening of the copper granules, the slimes are pumped towards a special device where treatment takes place. The decopperizing in acid solution is preferably made after roasting in air (15 %) and this leaves the slimes with 1 % Cu. The refining takes place in a small smelting furnace for the gold or precious metal alloys. Gold and silver are separable by dry or wet methods and this can be performed even better by double electrolysis. For the Ag : 600 x 500 mm wood cells with an electrolyte with 3 % $AgNO_3$ and 2 % HNO_3, an average of 300 A/m^2, voltage of 1.4-2 V, cathodes (thickness 1.5 mm) in pure Ag and gilded anodes preferably with between 200 and 300 thousandths gold wrapped in muslin sacks for avoiding the mixture of gold-bearing slimes falling from the anodes with the silver crystals from the cathodes. For the gold : concentration in an enriched gold-bearing solution (800 to 900 thousandths) with 300 x 450 mm sandstone cells, an electrolyte with a gold chloride base (50-80 g of Au/l) and 5-7 % of free HCl, an average of 750 A/m^2, voltage of 0.8-1.1 V and pure gold foil cathodes (0.25 mm) with collect Au while the Ag passes into the slimes ($AgCl_2$) and the Pt into solution with other metals.

The dusts and vapors from the numerous smelting, roasting and boiling cleaning operations are carefully collected by electrostatic filters.

Gold and silver are finally obtained in a final smelting in ingots with a grade of 99.95-99.99 %.

4.2 Hydrometallurgy

We will limit this section to the treatment of copper solutions since the extraction processes are described in chapter 3.2.2.

We will consider only the case of copper sulphate solutions which must be purified and then electrolysed with insoluble anodes.

The principal operations are as follows :

- Purification of solutions :
  - . By conventional wet methods,
  - . By solvent extraction.
- Electrolysis .

4.2.1 Purification of solutions before electrolysis for copper should eliminate the following impurities or keep them at reasonable levels ;

. Nitrate, chloride, $Iron^{+++}$, As.

Trivalent iron considerably reduces efficiency.

In chemical purification, chlorine can be eliminated as cuprous chloride and ferric iron can be partially eliminated for a fraction of the flow circulating between electrolysis and dissolution. There is oxidation if necessary and neutralisation with ore followed by procipitation of $Fe(OH)_3$ which is thus eliminated from the circuit.

The following are typical analyses before and after chemical purification (g/l) :

Elements :	Cu	Total Fe	Fe^{+++}	Fe^{++}
Before	66	2,83	1,64	1,19
After	80	1,14	0,63	0,(&

This is however a complex method involving expensive solid/ liquid separations which are hard to perform and there are also metal losses.

Solvent purification is now performed industrially with success in the United States and Africa. Other projects are under way throughout the world.

Industrial extraction began in 1968 at the Bluebird mine of the Rancher's Exploration Co. (Powers 1970). The origin of this method seems to have been the need to develop an inexpensive method allowing the small producers to make themselves independent of the smelters. It was therefore necessary to develop a technique making it possible to eliminate cementation, smelting and refining and to obtain a metallic concentrate of high purity at the mine. The ideal process would be to successively extract copper from not very concentrated solutions also containing iron.

On the basis of fundamental chemical considerarions solvents were developed which satisfied industrial requirements. These reagents are hydroxyoximes which chelate the copper ions.

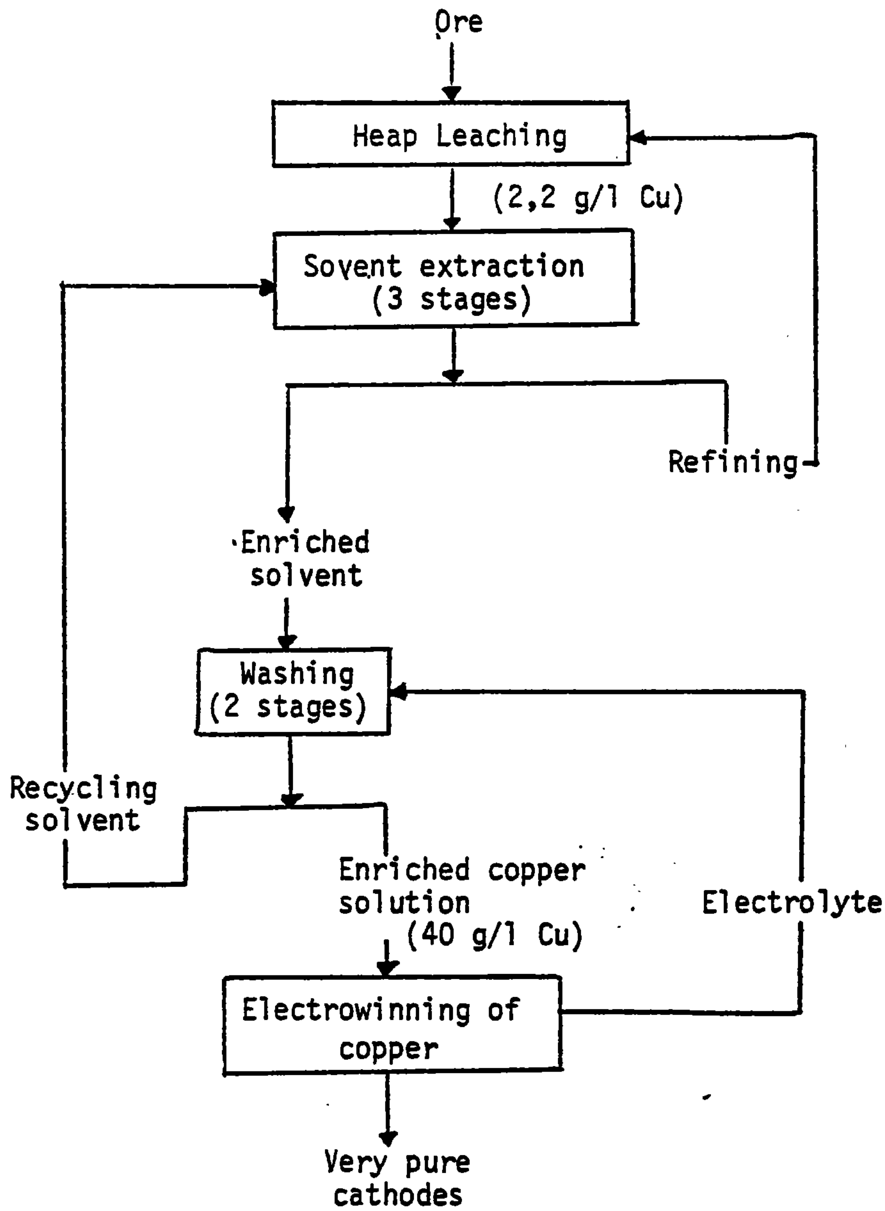
The feed solution contains about 2.2 g/l of copper and comes from the sulphuric heap leaching of silicate and carbonate ore. The enriched solution contains 40 g/l of copper and is electrolysed so as to produce 23 t/d of cathodes with a 99.9 % Cu grade.

Other companies are beginning to use a process of this kind or are using it at the pilot stage.

The N'Changa Consolidated Copper Mines Co. has since 1970 built a solvent extraction and electrowinning plant whose capacity is around 60 kt/y of metal.

4.2.2 Electrowinning of copper from purified solutions is electrolysis with insoluble anodes.

Below are given the principal data for the electrowinning of copper either from chemically purified solutions or from solutions purified with solvents.



FLOW SHEET BLUEBIRD

		U.S.A.	Africa	U.S.A. (Bluebird)
Electrolyte	Cu g/l	31	66	36
	free acid g/l	20	6.7	-
	Fe^{++}	-	1.0	0.5
Cells	Voltage	2.16 V	2.05	2.0
	Faraday Efficiency	0.72	0.90	0.84
	kWh/kg of copper	2.97	2.05	2.64
	Current density	125 A/m^2	180 A/m^2	190 A/m^2
	Temperature	41° C	55° C	?
	Anodes	Pb with 8% Sb	Pb with 6% Sb	Pb with 6%Sb
	Tanks	Concrete+Pb	Concrete+Pb	Concrete+PVC

4.3 Economic Aspect

It is obviously very difficult to present data on this point since the manufacturers tend to withold information.

We have therefore attempted to arrive at a reasonable estimate for the operating cost and necessary investment cost.

4.3.1 Probable Operating Cost for pyrometallurgy and electrolytic refining for a plant producing 100 kt/y of copper.

The tables below show the cases considered.

Blister Production

In our opinion operating cost would be between 1,000 and 1,200 F/t depending upon the sophistication of the process.

We have deducted a credit for vapor and acid (25 % reduction for acid) or 200 x 0.75 = 150 F/t and we arrive at an operating cost of between 500 and 650 F/t of blister produced.

Comparison with costs in West Germany shows that the later agree quite well with this figure of 1,000 to 1,200 F/t of blister.

The cost of 11 cts/lb was observed for 1977 which makes 1,211 F per tonne before deduction of the vapor and acid credits.

Anode Refining

Refining in Europe costs about 6.5 to 7.0 cts/lb. The figures in the following table make it possible to calculate this price.

If refining is automated, it is possible to reduce the labor force and there is a 6-10 % advantage over conventional refining.

TABLE N° 21

ESTIMATE OF OPERATING COST IN EUROPE

(Without Amortization)

Base : 100 kt/y of metal from concentrates with 25 % Cu. Unit : MF

	Reverberatory Furnace	Flash Furnace
Labor	25	24
Electricity	5	5
Fuel Oil	30	18
Refractory Materials	2	3
Fluxes	3.5	2.2
Miscellaneous	0.5	0.5
Spare Parts	6	5.5
Acid Production	25	25
Overhead	20	20
Gross Total	117	103.2
Acid credit (300 kt/y)	45	45
Vapor credit (250kt/y)	10	10
Net Annual Cost	62 MF	48.2 MF
i.e. per tonne of blister	620 F/t	480 F/t

4.3.2 Capital Costs in 1977 for the production of 100 kt/y of copper are difficult to estimate.

A price range will therefore be given but the reader should use these figures with prudence.

PLANT	Investments in MF 1977	
	Minimum	Maximum
Conventional Smelting	600	700
Outokumpu Flash Smelting	550	650
Refining	150	250

4.3.3 Operating Costs for solvent Route, followed by electrolysis.

We will give a rough estimate for a treatment price for a plant with a capacity of 7 kt/y of metal : 8.6 cts/lb in 1977 or 947 F/t of metal.

This estimate was given to us by a British engineering company.

4.3.4 Capital costs in the US for a small plant (from ore to metal) with a capacity of 7 kt/y were in 1976 about 6 M$ i.e. 860 $ per tonne of annual capacity or 4,300 F/T of copper.

TABLE N° 22

ESTIMATE OF OPERATING COSTS FOR ELECTROLYTIC REFINING (100 kt/y)

(Without Depreciation)

	Conventional Plant	Highly Automated Plant
Labor (Including Social Charges)	23	16
Electricity	6	6.5
Fuel	3	3
Reagents and Miscellaneous	13	13
Maintenance	10	12
Total-Technical	55	50.5
Overhead	20	20
Annual Total (MF)	75	70.5
i.e. per tonne of wirebar	750 F/t	705 F/t

4.4 Present Situation and Forecast Trend for Plant in the EEC

4.4.1 General Situation in the EEC (Table 23)

In 1975 the nine EEC countries consumed 1,954 kt of refined copper which can be broken down as follows :

West Germany	32 %
United Kingdom	23 %
France	19 %
Italy	15 %
Belgium-Luxembourg	9 %
Other EEC countries	2 %

In the same year refined copper production was 885 kt with West Germany accounting for 48 %, Belgium 29 %, the United Kingdom 17 %, France 5 % and Italy 1 %. For the EEC, production therefore covered 45 % of consumption but the respective situations of the various countries were very different : while Benelux covered consumption by 149 % and West Germany by 66 %, rates were low or very low for the United Kingdom (34 %), France (11 %), Italy (4 %) and the other EEC countries (0 %) making them highly dependent upon imports of refined copper.

When the refined copper production capacities of the EEC are taken into account, the theoretical maximum rate of coverage of needs by production would be close to 59 % and there would also be very different situations in the various countries.

These rates which may seem to be quite satisfactory should be reduced since it is necessary to import large quantities of unrefined metal for the supply of European units. Thus nearly 400 kt of unrefined metal were imported by the EEC in 1975 and the European share in refinery supply is only on the order of 55 % (EEC smelter and scrap production). Lastly, at the smelter level a large part of the raw material used consists of imported ores whose metal content covers about half the needs while the remainder comes from Community mine production and, especially, resmelted scrap.

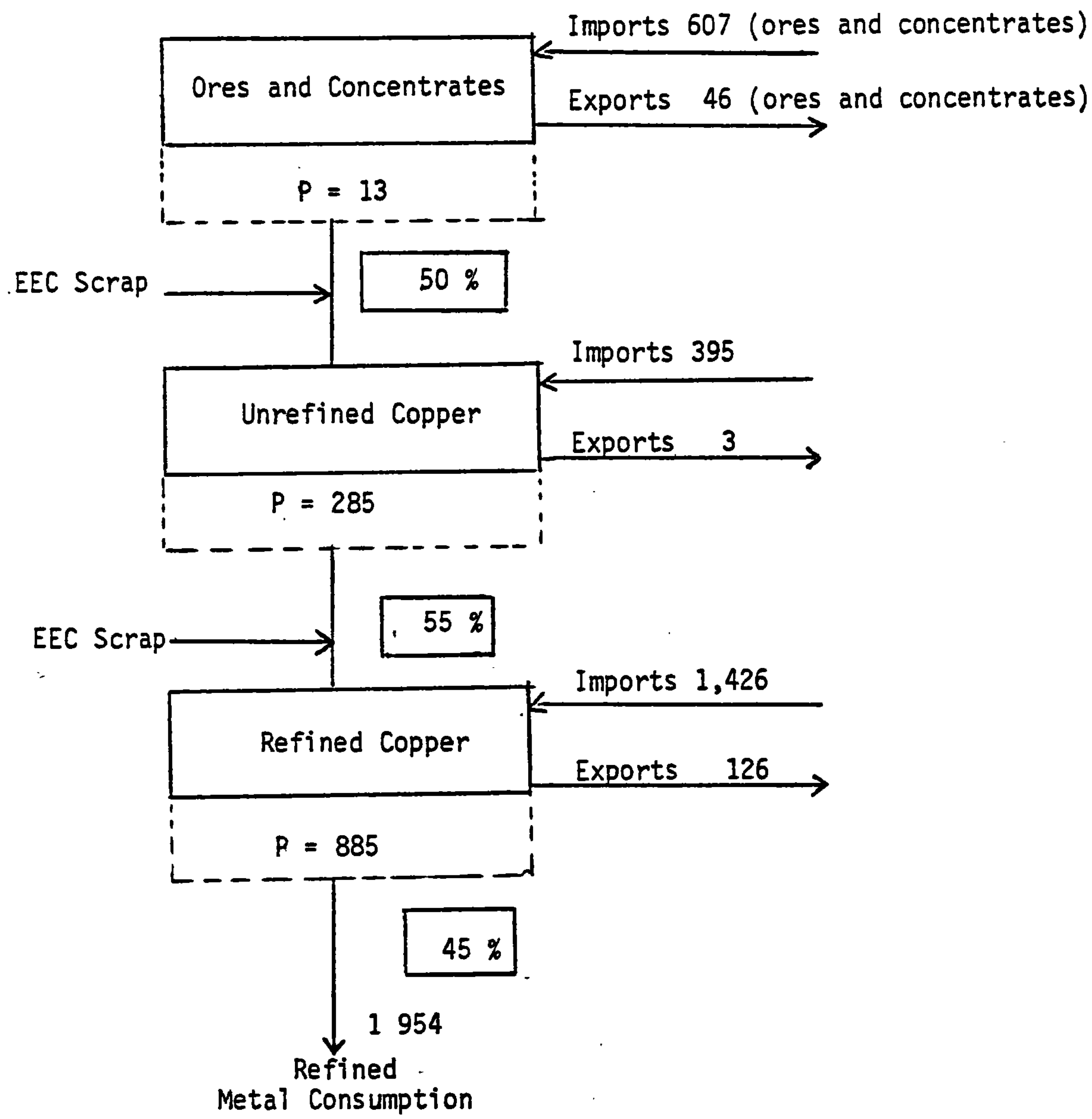
Thus in the EEC copper production sector, it appears that only 17 to 18 % of refined copper consumption is covered by an

EEC SUPPLY IN 1975

in kt Cu

(Not including inter-EEC trade)

(Simplified Figure)



P = EEC production, in metal content

Percentage of flow from EEC production

TABLE N° 23

EEC COPPER CAPACITY, PRODUCTION AND CONSUMPTION IN 1975

	Smelter Capacity (kt) (a)	%EEC	Refinery Capacity (kt) (b)	%EEC	PRODUCTIONS 1975				Refined Copper Consumption 1975 (e)		(d)/(e) in %	(b)/(e) in %	(c)/(a) in %	(d)/(b) in %
					Unrefined Copper (kt) (c)		Refined Copper (kt) (d)							
W. GERMANY	351	70	413	36	216	(76 %)	422	(48 %)	635	(32 %)	66	65	62	102
BELGIUM.LUX.	85	17	375	33	66	(23 %)	259	(29 %)	174	(9 %)	149	216	78	69
FRANCE	11	2	49	4	4	(1 %)	40	(5 %)	365	(19 %)	11	13	36	82
ITALY			33	3			13	(1 %)	40	(5 %)	4	11		39
UNITED KING.	55	11	280	24			152	(17 %)	451	(23 %)	34	62	0	54
Other EEC Countries									40	(2 %)	0	0		
TOTAL EEC (1975)	502	100	1 150	100	285	(100 %)	885	(100 %)	1 954	(100 %)	45	59	57	77
in 1976	546		1 144											

exclusively internal source (except for the direct use of new scrap). Most of this production is linked to scrap and not taking into account the variations in stocks, the former amount to about 350 kt in 1975. The copper scrap recovery rate would be slightly higher if scrap exported outside of the EEC were added but the quantities involved are not very great.

As to the rate of utilisation of the various European units in 1975, smelters used an average of about 57 % of capacity with the Belgian smelters having the highest utilisation rate. As to refineries, the average utilisation rate for the entire EEC was 77 % with the German and French units having higher rates. As concerns West Germany, the rate in Table 23 is above 100 % while the exact figure should be lower since there is a problem involved in the definition of processing units. This is due to the fact that refining units sometimes resemble semi-product manufacturing units and there is also a classification problem for units wholly or partly using scrap. The figures given should therefore be considered as orders of magnitude.

4.4.2 Processing Units in the EEC

Tables 24 and 25 list the smelters and refineries in the EEC. Detailed technical descriptions of the principal plants will be found in the appendix.

For each unit the tables show location, capacity, the producer company and the names and shares of the parent companies.

In 1976 West Germany had 67 % of the overall smelting capacity of the EEC, which was 546 kt, with Belgium representing 16.5 %, the United Kingdom 14.5 % and France 2 %. At the refinery level for an overall capacity of 1,144 kt, West Germany represented 39.5 %, Belgium 32.5 %, the United Kingdom 21 %, France 4 % and Italy 3 %.

In 1976 the first European unit utilising a leaching process followed by electrowinning was opened. This plant located at Duisbourg will start with a capacity of 10 kt.

TABLE N° 24

EEC SMELTERS IN 1976

COUNTRY (Total installed capacity in 1976)	LOCATION	COMPANY	COMPANIES OWNING SHARES	CAPACITY
W. GERMANY	HAMBURG	NORDDEUTSCHE AFFINERIE	METALLGESELLSCHAFT 40 % DEGUSSA 40 % BRITISH METAL CORP. 20 %	175 kt
	LÜNEN	HÜTTENWERKE KAISER AG	L. POSSEHL AND CO	75 kt
(366)	BERLIN REINICKENDORF	BERLINER KUPFER RAFFIN. G m b H	NORDDEUTSCHE AFFINERIE 66.7 % HUTTENWERKE KAISER	80 kt
	DUISBURG	DUISBURGER KUPFERHÜTTE	B.A.S.F. 30.97 % HOECHST 30.97 % BAYER A.G. 30.97 % + 3 other companies	30 kt
	LÜBECK	METALLHÜTTENWERKE LÜBECK A.G.		6 kt

TABLE Nº 24 (Continued)

COUNTRY (Total installed capacity in 1976	LOCATION	COMPANY	COMPANIES OWNING SHARES	CAPACITY
BELGIUM (90)	HOBOKEN	METALLURGIE HOBOKEN OVERPELT	STE GENERALE DE BELGIQUE 13,5 % UNION MINIERE 44,8 %	50 kt
	BEERSE	METALLO CHIMIQUE S.A.	SICOMET SA (BELG.) CHEMICO METALS CORP (US)	40 kt
FRANCE (11)	POISSY	STE FRANCAISE D'AFFINAGE DU CUIVRE	HOBOKEN 45 % PENARROYA 45 % Cie Gle ELECTROLYSE DU PALAIS 10 %	11 kt
UNITED KINGDOM (79)	WIDNES	B.I.C.C. METALS LTD	BRITISH INSULATED CALLENDER'S CABLES LTD	20 kt
	NORTH FERRIBY	CAPPER PASS AND SON LTD	RIO TINTO ZINC CORP. 100 %	4 kt
	WAL SALL	ELKINGTON COPPER REFINERS Co LTD	BRANDEIS GOLDSCHMIDT AND CO LTD	15 kt
	WAL SALL	I.M.I. REFINERS LTD	I.M.I. REFINERY HOLDINGS LTD	35 kt
	WIDNES	Mc KECHNIE CHEMICALS LTD	Mc KECHNIE BROTHERS LTD	5 kt

TABLE N° 25

EEC REFINERIES IN 1976

COUNTRY (Total installed capacity in 1976	LOCATION	COMPANY	COMPANIES OWNING SHARES	CAPACITY t:thermal e:electro
W. GERMANY	HAMBURG	NORDDEUTSCHE AFFINERIE	METALLGESELLSCHAFT 40 % DEGUSSA 40 % BRITISH METAL CORP. 20 %	240 kt (e) 30 kt (t)
	LÜNEN	HÜTTENWERKE KAISER A.G.	L. POSSEHL AND CO	95 kt (e)
	BERLIN REINICKENDORF	BERLINER KUPFER RAFFIN. G m b H	NORDDEUTSCHE AFFINERIE 66,7 % HÜTTENWERKE KAISER	15 kt (t)
(451)	OSNABRÜCK - NÜRNBERG	KABEL UND METALLWERKE GUTEHOFFNUNGS HUTTE A.G.	GUTEHOFFNUNGS HÜTTE A.G. 83,9 %	48 kt (t)
	WERDOHL - NÜRNBERG	VEREINIGTE DEUTSCHE METALLWERKE A.G.	METALLGESELLSCHAFT 98,5 %	13 kt (e)
	DUISBURG	DUISBURGER KUPFERHÜTTE	B.A.S.F. 30,97 % HOECHST 30,97 % BAYER A.G. 30,97 % + 3 other companies	10 kt (EW) (Leaching and electro extraction)

TABLE N° 25 (Continued)

COUNTRY (Total installed capacity in 1976)	LOCATION	COMPANY	COMPANIES OWNING SHARES	CAPACITY (t) thermal (e) electro
BELGIUM (370)	OLEN	METALLURGIE HOBOKEN OVERPELT	STE GENERALE DE BELGIQUE 13,5 % UNION MINIERE	330 kt (e) (conventional smelting 530 kt)
	BEERSE	METALLO-CHIMIQUE S.A.	SICOMET S.A. CHEMICO METALS CORP. (US)	40 kt (e)
FRANCE (47)	LE PALAIS	Cie Gle D'ELECTROLYSE DU PALAIS	P.U.K. 55 % UNION MINIERE 31,3 % PENARROYA 2,5 % METALLURGIE HOBOKEN 11,2 %	45 kt (e)
	TOULOUSE	AFFINERIE DU SUD OUEST		2 kt (t)
ITALY (33)	FORNANCI DI BARGA	SOCIETA METALLURGICA ITALIANA		23 kt(e+t)
	PADERNO DUGNANO	A. TONELLI E CO		10 kt (e)
UNITED KINGDOM (243)	PRESCOT	B.I.C.C. METALS LTD	BRITISH INSULATED CALLENDER'S CABLES LTD	138 kt (t)
	NORTH FERRIBY	CAPPER PASS AND SON LTD	RIO TINTO ZINC CORP. 100 %	4 kt (e)
	WAL SALL	ELKINGTON COPPER REFINERS Co LTD	BRANDEIS GOLDSCHMIDT AND CO LTD	27 kt(e+t)
	WAL SALL	I.M.I. REFINERS LTD	I.M.I. REFINERY HOLDINGS LTD	45 kt (e) 20 kt (t)
	WIDNES	Mc KECHNIE CHEMICALS LTD	Mc KECHNIE BROTHERS LTD	9 kt (t)

4.4.3 Estimated Trend for Plant in the EEC

There are few projects for new units or extensions in the Community. Capacity in West Germany and the United Kingdom will probably increase while in West Germany, Hüttenwerke Kaiser and Norddeutsche Affinerie will be associated with Codelco in a joint venture for the construction of a refinery with a capacity of 120 kt. In the United Kingdom, a new 55 kt electrolytic refinery is to replace the old installation in BICC's Prescot unit. Plant development in the Community will depend upon the availability of blister and concentrate on the copper market since the desire of the producer countries to develop their own processing units might create difficulties as concerns the supply of raw materials to plants in Europe.

5. SUPPLY BY SCRAP AND WASTE

5. SUPPLY BY SCRAP AND WASTE

. Available scrap and waste
. Recycling

5.1 Introduction

Recycling is accomplished by the secondary materials industry. The task of the procurement, identification and sorting, smelting, refining, and sale of copper and copper-base alloy scrap for use by refineries, brass mills, and ingot makers are functions of the recycling industry.

The scrap processors, secondary smelters, ingot makers and other companies have developed effective channels and efficient methods for recycling nearly all waste materials of economic value, in the traditional economic environment.

More recently additional dimensions have been added to this traditional environment. These new dimensions are :

- improvement of the living environment,
- increased national concern with conservation of natural resources,
- necessity to limit to the minimum dependance on foreign raw materials supply.

No longer is economic gain the sole driving force for recycling of waste materials. In an economic-based economical area this creates problems of interpretation and evaluation of non-economic-based goals and activities. It is therefore necessary to tackle comprehensively the problem of extending recycling to marginal and sub marginal solid waste to allow more materials to be recycled by removing impediments of a fiscal and legal nature, by creating some forms of incentive and promotion, and by the realization of research and development programmes to improve collecting and processing methods to ameliorate economies in recycling.

The study is based on statistical and scientific documentation as well as the results of specific surveys carried out by BIPE, Charter Consolidated Ltd., ITE on behalf of the EEC.

The criteria adopted for the compilation of statistics, the lack in almost all industrialized countries of direct and analytical data on commercial flow of products entering the market and on secondary recycled materials, compel to frame hypotheses and to set parameters for estimating scrap arisings which differ as to the various countries because of different patterns of consumption and structure of industry. Consequently the absolute values of recycled materials may result somehow defective. We deem however that the error incurred might not be such as to invalidate the conclusions arrived at in this study.

5.2 The copper recycling industry

Figure 1 shows the two major types of scrap that are the raw materials of the recycling industry : prompt industrial and obsolete. Prompt industrial scrap is the waste generated during a manufacturing operation. The obsolete scrap is generated when an used product is no longer useful and is discarded. This includes a great variety of types of scrap and situations : wide variety in value, time scale of recycling and the form of obsolete scrap.

Scrap trading is performed according to two possible circuits :

- a direct circuit from "scrap producer" to user : this circuit is used almost exclusively for a part of scrap at manufacturing and fabricating stages ;
- an indirect circuit which passes through an intermediate stage formed by scrap dealers/merchants : this is practically the only circuit used for obsolete copper products, it is however used also for a part of scrap at manufacturing and fabricating stages.

Table 1 provides an analysis of the major types of recyclable materials.

5.2.1 Structure of scrap metal trade

Old and new scrap is sold through complex and highly fragmented industry. Merchants provide the link between producers of scrap and end-users. At the base of the collection system are the local scrap dealers and itinerant collectors who buy in small quantities from the public or from industry on factory clearance

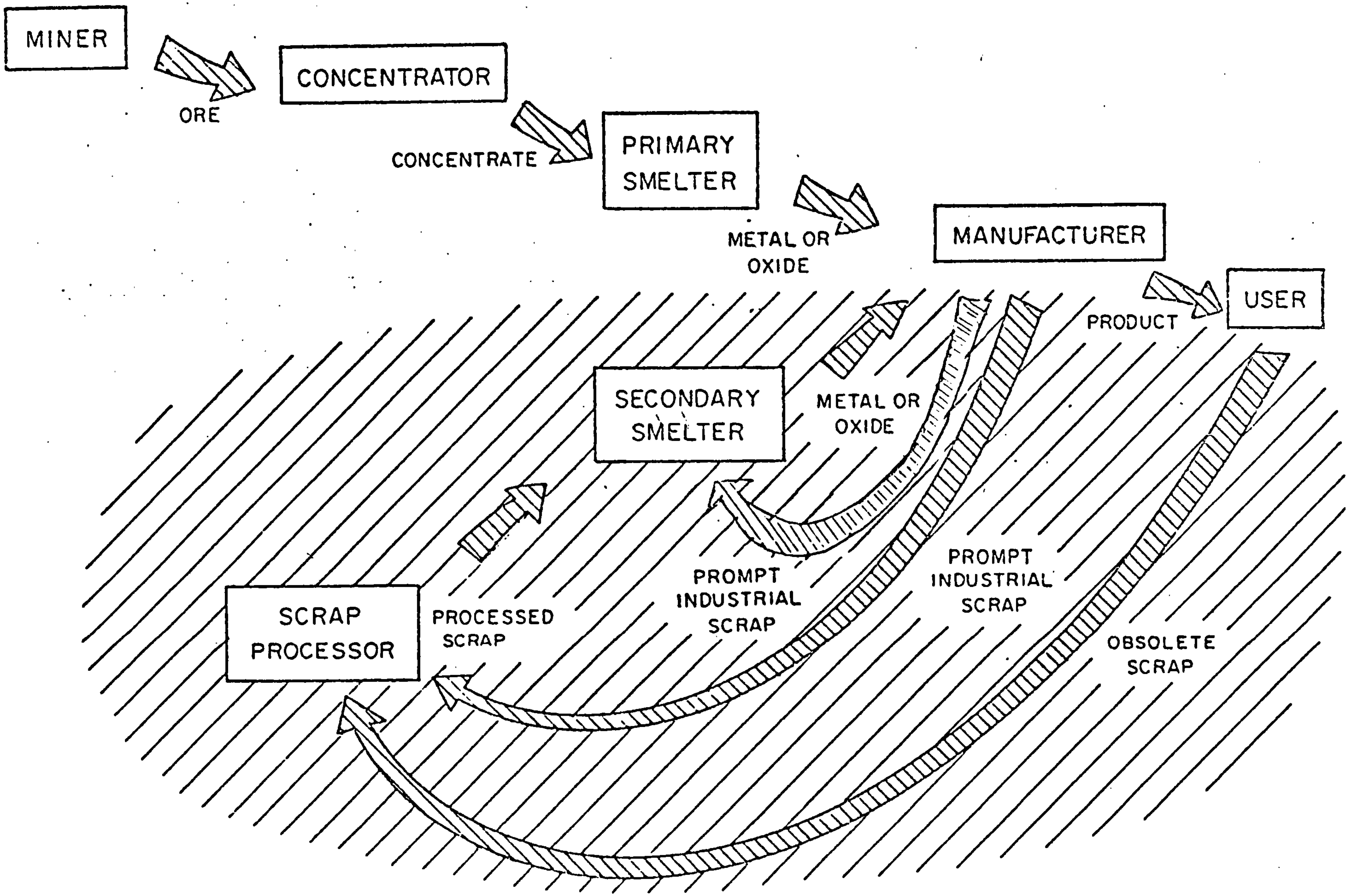


FIGURE I. FLOW OF PRIMARY AND RECYCLED METALS

TABLE 1 - ANALYSIS OF TYPES OF RECYCLABLE MATERIALS

Type of Material	Examples	Condition of Scrap	Sources
Manufacturing residues	Drosses Slags Skimmings	Highly variable in composition depending on the major constituents.	Metal melting operations - smelters, casters
Manufacturing trimmings	Machining wastes Blanking and stamping trimmings Wasting wastes	Highly variable as to size and shape.	Shaping operations - casters, stampers, machiners, fabric cutters, paper cutters.
Manufacturing overruns	Obsolete new parts Extra parts	Usually small size. Variable compositions.	Large manufacturers of mass-produced products.
Manufacturing composite wastes		Highly variable as to composition, size and shape. Often costly to process. Often, not all constituents are recovered.	No significant pattern of sources.
Flue dusts	Brass mill dust Steel furnace dust	Highly variable in composition and bulk density. Often not economical to recover.	Metal smelter and caster
Chemical wastes	Spent plating solutions Processing plant sludges, residues and sewage	Highly variable in composition. High value materials often recoverable.	Platers, metal cleaners, process industry plants.
Old 'pure' scrap	Copper tubing	Highly variable as to size and shape.	Consumers, industrial users, utilities, and other uses of the products that are scrapped.
Old composite scrap	Irony die castings Auto radiators	Highly variable as to composition, size, shape, and difficulty of separation. Often not economical to recover valuable materials.	Consumers, industrial users, utilities, and other uses of the products that are scrapped
Old mixed scrap	Auto hulks Appliances	Highly variable as to composition, size, shape, and difficulty of separation. Not all materials are recovered.	Consumers, industrial users, utilities, and other users of the products that are scrapped.
Solid wastes	Municipal refuse Industrial trash Demolition debris	Completely variable. Nearly always low in valuable materials. Very low recovery rates now.	All individuals and organizations.

Source: Battelle-Columbus estimates of recycle rates
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contracts. At this stage of the industry the scrap dealers/merchants are numerous and widely dispersed though located in those areas of the country which have the largest concentration of industry and hence the largest potential sources of scrap.

From the first tier, the material is sorted and perhaps baled before being passed on to larger dealers who in turn, after further processing, pass the material to the end-users. The large merchants buy scrap wherever it arises in reasonable quantities, i.e. from medium size merchants, direct from process scrap producers, from ship and car breakers, demolition companies and by tender from the Post Office, Government Departments and Electricity Boards, etc.

Since the price large merchants can obtain from the end-user is affected markedly by the homogeneity and size of scrap lot they have to offer, further sorting and processing of old scrap is carried out to increase its value. This movement of scrap from source to end-users through the chain of generation, collection, merchandising, processing and consumption is in many cases carried out within an integrated company structure. Such companies tend to specialize in one or two metals, though maintaining an active interest in all metals.

In the EEC countries the structure is quite uniform with regard to the functions performed and relations existing between local scrap dealers and itinerant collectors, medium size merchants and large merchants.

Scrap processing at an industrial level is carried out by medium size merchants and especially by large merchants.

Medium scrap metal trading represents the most reactive element to the scrap market economic situation and at its level stocks may be built for speculative reasons.

The structure of the large merchants companies are either of a type integrated with industrial groups of primary and secondary metal production or independent companies having medium size merchants as partners. The overall number of large merchants operating in the EEC is estimated to be 400. It is difficult to assess how many people these companies employ because of their different structures. It is deemed there are 150 companies with more than 50 employees. In 1970 in the United States the companies of this size were about 430 with 70 employees on average.

5.2.1.1 Classes of recycling companies

Many companies operate in more than one category. Table 2 is no more than a guide to some of the ways recycling companies may specialize.

5.2.1.2 Operations

Almost all scrap processing operations fall into one of the following categories :

. collection
. separation
. upgrading
. packaging
. shipping

to which are to be added disposal of waste material and the general function of material handling.

Table 3 provides an analysis of the operations of scrap processors. The types of function are difficult to perform : manufacturing operations are extremely difficult considering the wide variety in type, composition form of raw materials, trading function as it is necessary to find and purchase satisfactory scrap and/or solid waste at a price that will allow a profit when sold.

Table 4 gives an analysis of operations of smelters, whose problems are basically the same as for scrap processors.

Table 5 expands this functional list with specific types of equipment along with typical uses for that equipment and a brief discussion of some of the problems and benefits associated with the type of equipment.

TABLE 2 - CLASSES OF RECYCLING COMPANIES

Class of Company	Description of Operations
Nonferrous Scrap Metal Processor	(1) Locates scrap (2) Purchases (3) Identifies (4) Sorts and separates (5) Sizes the scrap (6) Densifies (7) Markets (8) Delivers
Nonferrous Metal Broker	(1) Locates scrap sellers (2) Locates scrap customers (3) Buys (4) Markets (5) Arranges pickup and delivery (6) Stabilizes source of supply
Smelter and Refiner	(1) Buys scrap (2) Upgrades by adjusting composition and casting into ingots or pigs (3) Markets to specifications
Sweater	(1) Buys scrap (2) Upgrades by melting one metal and separating from other metals with higher melting points that remain solid (3) Casts into ingots or pigs (4) Markets
Ingot Maker	(1) Buys scrap (2) Melts selected scrap to composition and casts into ingots (3) Markets to specifications
Brass Mill	(1) Buys scrap (2) Melts selected scraps and other materials to composition and casts into ingots (3) Produces sheet, strip, and other shapes from ingots (4) Markets shapes to size and specification
Primary Metal Producer	(1) May mine ores or purchase (2) May concentrate ores or purchase (3) Upgrades concentrates by reduction to metal, adjusting composition, and casting into ingots or pigs (4) Markets to specifications (5) Sometimes also operates as secondary smelter and refiner
Scrap Iron Processor and Broker	(1) Locates scrap (2) Purchases (3) Identifies grades (4) Separates and sorts (5) Sizes the scrap (6) Densifies (7) Markets (8) Delivers (9) Often also operates as nonferrous processor or paper stock dealer
Importer and Exporter	(1) Locates domestic or foreign scrap sources (2) Locates domestic or foreign scrap customers (3) Buys (4) Markets (5) Arranges transportation
Laboratory and Assayer	(1) Analyzes materials for a fee (2) Certifies composition
Manufacturer of Equipment	(1) Designs and manufactures equipment (2) Includes equipment for recycling industry

TABLE 3 – ANALYSIS OF SCRAP AND PROCESSOR OPERATIONS

Function	Methods
Collection of Scrap	(1) Arrangements with industrial scrap generators to buy and pick up scrap. Sometimes provides special containers and equipment at generators' plants. (2) Spot buying of scrap from factories, brokers, collectors, and other sources and picking up or arranging for delivery of the scrap. (3) Arrangements with organizations for scrap drives. (4) Buying and taking delivery of scrap brought to the processing yard by individuals, truckers, or others.
Identification and Separation of Scrap	(1) Identification and hand separation of various scrap materials from each other and from waste materials. (2) Testing of materials by chemical, spectrographic, and other analytical methods. (3) Burning-off or mechanical removal of organic materials from noncombustible scrap materials. (4) Magnetic separation of ferrous from nonferrous scrap. (5) Separation of heavy materials from light materials by air classification. (6) Separation of low melting from high melting metal scrap by selective melting. (7) Heavy media flotation of heavy from light materials. (8) Chemical solution of one material to separate from another.
Upgrading and Packaging of Scrap	(1) Reducing the size of scrap by torching, shearing, shredding, sawing, or other methods. (2) Packaging the scrap by baling, bundling, briquetting, or other methods to make handling and transportation easier and to meet customer needs. (3) Densifying scrap for ease of handling, storage, and shipment.
Delivery of Scrap	(1) Delivery to customer by owned or leased trucks or barges. (2) Delivery by public truck, rail, barge, or other forms of transportation. (3) Delivery by customer-owned or leased conveyance.
Trading	(1) Finding sources of scrap and customers for scrap. (2) Buying and selling scrap at a profit. (3) Keeping current on scrap prices. (4) Keeping up with market interrelationships, Government regulations, etc.

TABLE 4 - ANALYSIS OF SMELTER OPERATIONS

Function	Methods
Sizing of Scrap	(1) Baling of light scrap (such as wire, clippings, etc.) is sometimes done by smelters to make satisfactory furnace charging material. (2) Shearing of large pieces of scrap is sometimes done to reduce the sizes of scrap for charging to furnaces.
Upgrading of Scrap	(1) Sweating is done to remove low melting metals from higher melting inserts or attachments. (2) Fragmentizing and incineration are used to remove organic materials (such as wire insulation) from metals.
Refining	(1) Heat refining in smelting furnaces is the most widely used method of refining. (2) Electrochemical refining is used for some copper and precious metals. (3) Oxidation is sometimes done to produce metal oxides (such as zinc oxide) rather than the pure metal.
Melting	(1) Some metal scrap is not refined but merely melted and cast into pigs. The composition of the scrap must be carefully controlled since the output metal will have this same composition.
Alloying	(1) Alloying is often done in conjunction with refining. The output is then an alloy of the metal rather than the pure metal. Alloying is common for all the nonferrous metals. (2) Alloying can also be done in a simple melting operation. However, there is less choice of compositions than when alloying is done in conjunction with refining.
Analysis of Composition	(1) Analyses of scrap and recycled metals are made to determine composition for several reasons: • As a basis for pricing • To meet customer specifications • To make sure purchased scrap meets specifications • As a guide to refining procedures (2) Methods of analysis include (a) visual examination, (b) spark tests, (c) chemical tests, (d) chemical analysis, (e) spectrographic analysis. (3) Analysis is done on incoming scrap, on in-process metals, and on finished metals.
Trading	(1) Finding sources of scrap and customers for recycled metals. (2) Buying of scrap and selling of recycled metals at a profit. (3) Keeping current on scrap prices and metal prices.

TABLE 5 — IDENTIFICATION AND ANALYSIS OF SCRAP PROCESSING EQUIPMENT

Equipment	Function(s): Collection	Separation	Upgrading	Packaging	Materials Handling	Disposal	Shipping	Typical Uses	Analysis
Mobile Auto Crusher	X			X				(1) Reduce shipping volume for auto hulks. (2) Produce improved shredder feed. (3) May make auto hulk processing economical for remote areas.	(1) Appear to be gaining popularity. May be partial answer to abandoned auto problem.
Baler, Press, Briquetter			X	X				(1) Increase density of scrap for shipment. (2) Produce scrap that is easier to handle, store, and ship. (3) Produce a "sized" product.	(1) Contamination has been and continues to be a problem. Lower quality product. (2) Seems to be losing popularity to shredded scrap in many markets.
Refuse Compactors, Containers	X				X	X	X	(1) Supplier depository for raw material. (2) Material handling. (3) Part of disposal scheme for solid waste generated during processing. (4) Storage and shipment of high value scrap.	(1) Higher densities are desirable from a collection cost standpoint. (2) Reduces pilferage. (3) Prevents contamination. (4) Encourages generator segregation.
Shredder, Impact Grinder, Mill Hammermill, Crusher, Hogger, Battery Breaker, Fragmentizer		X	X	X				(1) Liberates desired raw material from other components. (Insulated wire and auto bodies for example). (2) Reduce size prior to baling. (3) Produce cleaner scrap. (4) Upgrade (turnings, etc.)	(1) This type of equipment is inherently self destructive and requires extensive maintenance both emergency and preventive. (2) This type of equipment is the heart of any scrap handling system. Much care must be taken in selecting proper model, size, etc. (3) Raw material supply is critical along with assured markets for scrap. (4) Need for lower energy mills that do not require extreme maintenance. (5) May require continuous operation to be profitable.
Shears, Torches, Saws		X	X	X				(1) Reduce size of scrap to marketable size. (2) De-package.	(1) Popular because of versatility
Scale	X					X	X	(1) Record weight of incoming and outgoing material	(1) No scrapyard can operate without scales. (2) Basis for all financial transactions on the buying

TABLE 5 — IDENTIFICATION AND ANALYSIS OF SCRAP PROCESSING EQUIPMENT (Continued)

Equipment	Function(s): Collection	Separation	Upgrading	Packaging	Materials Handling	Disposal	Shipping	Typical Uses	Analysis
Conveyors, Fork Lift Trucks, Other Mobile Materials Handling Equipment					X		X	(1) Physically move raw material and scrap from one point to another. (2) Automated loading for shipment. (3) Combination of conveying and vibratory separation.	(1) Need for developments in the design of automated materials handling equipment for the scrap processing industry. (2) Has been a neglected area from technology standpoint.
Separators--Magnetic, Heavy Media, Air, Screens, Chemical		X	X					(1) Remove impurities prior to shipment. (2) Separation prior to processing to increase capacity of unit or to divert for separate processing.	(1) Most separation processes are still hand operations. (2) Offers opportunity to obtain more revenue (yield) per ton processed. (3) Special purpose separators are available but are difficult to convert to general purpose.
Furnaces--Sweat, Incinerator, Dryers		X	X	X		X		(1) Liberation of raw material from combustible components (auto body, insulated wire). (2) Separate metals by melting point. (3) Produce pigs, etc., for easier shipping, storage, analysis, etc.	(1) Sweat furnaces may be a feasible method for separating white metals from nonmagnetic auto shredder output. (2) Incineration may again become an economical method of separation as improved pollution control equipment becomes available. (3) Often violate pollution codes.
Cranes--Magnetic, Grapple					X			(1) Physically move material during processing, loading, and unloading.	(1) Magnet capacity has reached the upper limit. Any increases will now come from new technology. (2) Is an inefficient method of material handling.
Pollution Control Equipment						X		(1) Allow the use of pollution generating processing equipment.	(1) While solutions are available for most operations, they tend to be very expensive. (2) Selection of equipment often requires trial and error.

TABLE 5 — IDENTIFICATION AND ANALYSIS OF SCRAP PROCESSING EQUIPMENT (Continued)

Equipment	Function(s): Collection	Separation	Upgrading	Packaging	Materials Handling	Disposal	Shipping	Typical Uses	Analysis
Over-the-Road Trucks	X						X	(1) Collection and shipping of material.	(1) Becoming a necessary function in many areas in order to obtain a supply of raw material. (2) A sector of the secondary materials industry is becoming service oriented.
Secondary Smelting, Melting and Other Refining Furnaces		X	X	X				(1) Removing impurities. (2) Changing physical form. (3) Producing various alloys. (4) Analyzing composition.	(1) Pollution control is necessary with most of this equipment.
Identification Equipment--file, chemicals, spectrographs, etc.	X	X				X		(1) Grade raw material and prepared scrap. (2) Establish prices. (3) Controlling specifications.	(1) Automatic identification and sorting equipment not currently available. (2) Much of this type of equipment requires a skilled operator. (3) Most identification processes are manual.
Scrap Handling Systems		X	X	X	X	X		(1) Handle entire processing operation from receipt of raw material through loading for shipment.	(1) Necessarily inflexible--requires specialization. (2) Expensive but perhaps very profitable for a high tonnage operation. (3) Assured sources of supply and markets for product are necessary.
Systems to handle municipal wastes	X	X	X	X	X	X		(1) Handle municipal solid waste as an alternative to disposal. (2) Extract marketable materials from solid waste and sell. (3) Dispose of remaining material through normal channels.	(1) Not yet economical. (2) Not yet being considered as a viable alternative to disposal. (3) Government sponsored demonstration projects currently in process and to be funded in near future should assist development of feasible systems.

5.2.2 Secondary material flow in industry

Enclosed with the appendix are :

- definition of copper scrap ;
- classification of unalloyed copper scrap, copper-base scrap.

Scrap, according to its characteristics, may be recycled directly at the first processing stage and be considered under the item "Direct use of scrap" ; it undergoes fire refining and electrolytic refining and is classified as "refined copper".

In general terms, secondary material flow in industry is the following :

- high grade alloy scrap, that is scrap whose composition is known with a high degree of certainty, goes almost entirely to the direct production of alloy semi-fabrications of the same or closely allied composition ;
- high grade scrap copper, in the form of wire, tube or sheet can be used to make new copper semi-fabrications with a minimum of refining, but can also be used with no further treatment in the manufacture of brasses. In the main it passes into the copper rather than the alloy side ;
- clean casting scrap is used mainly to make foundry ingot but some is refined back to copper ;
- the lowest and most ill-defined grades of copper and alloy scrap such as drosses and slags from copper melting, furnaces, mixed metallics, scraps and residues containing upwards of 5 % copper, car radiators, etc. are treated only in the refining side for recovery of copper.

The foundry ingot side uses mainly old scrap, while the refining side uses both old and new scrap. The copper alloy side is geared to the maximum utilisation of secondary material and only sufficient primary material is added to make up tonnages required. Pure copper semi-fabrications, on the other hand, can be produced either from primary or scrap copper. Lower grade copper and alloy scrap can only be returned to the cycle after being refined to pure copper.

Processing of copper scrap

Copper and copper alloy scrap is recycled in three ways :

- by direct melting for the production of semi-fabrications of rod, sheet, wire and tube ;
- in the manufacture of ingots for the foundry trade ;
- as feed to the refining sector of the industry.

In the main, it is the nature and composition of the scrap that determines which of the above routes is chosen for its rehabilitation. In essence, the processing found in the secondary copper industry, apart from the initial sorting, baling, shredding, etc. common to all non-ferrous scrap, is either melting with a minimum of refining as seen in the production of ingot material or melting with substantial refining as seen in the secondary refining sector.

There are five main categories of users of secondary copper and copper alloys :

++ Ingot making

In ingot production, the crucial factor for commercial success is the identification of the in-going charge. This allows the minimum processing required, in particular refining, in order to obtain the final composition. The skill in scrap utilization lies in blending the charge, usually from a small tonnage of different arisings, to produce the lowest cost feed mixture for a given alloy. Five main types of furnace are used in the secondary ingot industry. These are the reverberatory, rotary, Ajax Wyatt, coreless induction and crucible tilting units.

So far as scrap supplies are concerned, the ingot making sector is in a vulnerable position, situated as it is between the direct users of scrap, who can pay higher prices for the top quality material, and the secondary refiners, who can absorb a more flexible material for their feed requirements. It might, therefore, be expected to diminish in size both on a tonnage basis and in the number of companies concerned. The potential advantages of centralized production are unlikely to be achieved by rationalising the remaining ingot makers. This is because a large production unit would still have to supply the small scale requirements of individual foundries, through a wide range of alloy composition requirements over a widely dispersed geographical area.

++ Secondary refining

The quantity and quality of scrap, which is currently being used in secondary refining, is determined by two main factors :

- the difference in profit which can be made by refining either blister or the various forms of scrap available on the market ; the natural feed to the secondary refineries is material which the ingot makers cannot handle, usually because the grade is too low and the contamination too high ;
- the electrolytic tank house capacity available to treat anodes made from partially refined scrap of blister.

Scrap can enter the circuit in an electrolytic refining plant in three places. High grade copper can be melted in conventional reverberatory furnaces of up to 200 tonnes capacity, together with foundry material and cast into anodes for electro-refining. Lower grade materials, usually excluding the gunmetals and bronzes, go to the blast furnaces. Tin-containing alloys are sent direct to the converters.

Blast furnaces are vertical shaft furnaces (average cross sectional area 30 sq.ft.) heated by the combustion of coke. The average copper content of the charge can be as low as 25 %-30 % copper, though this "average" is made up of slags containing as little as 5 % copper, residues containing about 40 % copper, and radiator and brass scrap containing as much as 60 %-65 % copper. The function of the blast furnace is to blend and melt the scrap to produce a high copper product and a discard slag. Molten black copper from the blast furnace is treated by oxidation in converters silimar to those used in primary copper smelting, but smaller in size —say 10-20 ton capacity. Zinc, tin and some of the lead are removed as a final operation in the converter.

The rough copper product is then fire refined in rotary or reverberatory furnaces, and cast into anodes. This secondary anode is of lower grade compared to the primary production anodes, containing as it does about 98.5 % copper, nickel up to 1 % and lead up to 0.5 %. Nickel has to be removed from the electrolyte continuously and recovered as a crude nickel sulphate. The final cathode, nominally 99.9 % copper, is still not the same quality as the better primary products. Thus, the majority of secondary cathode is unsuitable for the manufacture of materials in which higher conductivity or ductility is required. The third product of the refinery is anode sludge which contains copper, tin, lead, nickel, gold and silver.

++ Wrought industry

In the manufacture of brass semi-fabrications the most common type of furnace used is the electric mains frequency induction furnace. Over 50 % of brass scrap, in the form of swarf rod ends and sheet is commonly used in the melting charge. On occasions 100 % scrap is employed if it is available. Zinc ingot usually has to be added to make up losses by volatilisation. The rest of the charge is made up of the cheapest of the available acceptable grades of copper, zinc and lead.

For the production of copper semi-fabrications, the demand for high grade scrap is virtually insatiable since it is almost directly equivalent to foundry metal in this field of application. It should be noted that this use of scrap represents the tightest possible recycle and the quality of the materials accepted is of the highest level. The wrought producers do little sorting themselves, preferring to buy in scrap from reliable sources. This material is of prime melting quality and wherever possible is reprocessed to produce alloys similar to its original composition.

++ Foundries

Whenever possible, foundries prefer to use high quality scrap of known composition rather than the product of the ingot maker. The economics are more attractive as the added cost of the ingot maker's processing can be eliminated, though admittedly some of this will be offset by the higher price paid for the scrap. Here, identification of the scrap is crucial as the purchase of scrap is a much more risky business than the purchase of similar ingot material. Errors in scrap analysis, if allowed to proceed to the melting stage, can alter the economics adversely in that the result is the production of an alloy of wrong specification or an increase in refining time to remove deleterious impurities.

++ Miscellaneous

As its name implies, this sector is extremely wide though the tonnage is very small, 1 %-2 % of total arisings. The main end users are the producers of paints, chemicals (such as copper sulphate), dues, additives to cast irons, etc.

Within the structures of users of secondary copper and copper alloys in the EEC member countries, generally suited to the importance of industry at the first processing stage, secondary refining calls for a particular mention.

Secondary refining industrial structure of Germany and Belgium, for capacity and specialization degree, and thanks also to the structures of primary metallurgy, is remarkable, higher than domestic demand, so that these two countries are importers of residues and scrap from the EEC and extra-EEC countries.

The United Kingdom shows a balanced structure.

The structure of France and Italy is lower than demand and this gives a certain rigidity to the recycling system.

5.3 The importance of copper-base scrap in industry

On the basis of statistical data available and information provided by surveys conducted in the EEC member countries on behalf of the EEC by BIPE, Charter Consolidated Ltd., and ITE, the following tables have been compiled : Tables I on Production and consumption of copper, and Tables II on Domestic scrap arising, by each EEC member country and the EEC as a whole for the period 1967 to 1976. In addition, tables have been compiled on Secondary production ; Secondary production as percentage of total consumption ; Secondary refined copper production ; Direct use of scrap, regarding the EEC and the other economic areas in the Western world in the years 1963, 68, 73, 74, 75 and 76.

Tables on Total copper consumption referring to the same years and pertaining growth rates are included in Chapter 7 "Structure of consumption".

The criteria adopted to compile the statistics, scrap generation and old scrap recycling rate are described in appendices III, IV and V respectively.

In the Western world secondary copper production accounts for 38 %-39 % of total copper consumption. This figure remained practically constant during the period 1963-76, with the exception of a sharp decrease in 1975, due to the recession period which clearly had an adverse effect on scrap recycling.

In the 70's some general trends are noted of a decline in the recycling rate of scrap with an apparent recover in 1976 ; a growth rate of secondary refined copper lower than that of direct use.

Secondary copper production in the EEC countries accounts for 38 %-39 % of total consumption.

Secondary production from domestic sources and domestic scrap arising rates expressed in percentage of total consumption are, for the major EEC countries, lower than the world average rate, especially in Germany and Italy where the recycling rate of old scrap is particularly low compared with levels of total consumption. The shortage of secondary material is compensated with imports, from extra-EEC countries, of scrap and residues accounting for 10 % compared with total secondary production, while exports of scrap and residues account for 1.5 %-2 %. In fact, the established policy of the Community is to retain as much scrap as possible since this is virtually the only domestic source of raw material for the industries refining or otherwise using secondary copper and alloys. Within the EEC, trading of residues is very active ; particularly favoured countries are West Germany and

Belgium due to their primary metallurgy and refining industrial structure which allows the processing of various types of low-grade residues.

Equally active is the trade of scrap towards Belgium due to its high capacity of refined copper production, and towards Germany and Italy whose generation of old scrap is inadequate compared with total consumption of copper.

SECONDARY PRODUCTION AS PERCENTAGE OF TOTAL CONSUMPTION

	1963	1968	1973	1974	1975	1976
Germany F.R.	39.9	43.6	38.9	39.8	38.6	40.2
France	37.5	34.1	32.1	26.0	25.4	34.4
Italy	35.0	45.5	40.8	41.5	36.2	43.2
Netherlands	57.4	55.9	39.9	42.4	35.6	30.3
Belgium-Lux.	57.6	94.1	47.5	61.1	44.9	41.7
United Kingdom	35.5	42.4	37.9	38.6	34.8	38.5
Ireland	-	-	-	-	-	-
Denmark	13.3	57.4	5.2	37.3	41.5	36.4
E E C	38.7	45.5	38.2	38.9	35.3	39.2
Other Europe	32.5	40.8	41.7	41.5	41.9	42.7
EUROPE	37.7	44.8	38.9	39.4	36.6	39.9
South Africa	25.0	27.5	26.8	27.1	26.0	26.0
Other Africa	20.3	27.2	27.3	27.0	24.1	25.2
AFRICA	23.5	27.4	27.0	27.1	25.6	25.8
Japan	45.0	38.2	34.0	43.1	35.7	32.2
Other Asia	16.0	24.1	26.6	23.1	21.8	24.7
ASIA	39.7	37.1	33.2	40.3	34.0	31.2
Brasil	15.5	29.0	29.5	30.8	26.0	26.0
Canada	20.0	27.1	23.9	21.8	21.9	20.3
USA	42.9	47.7	42.8	45.8	46.5	43.0
Other America	38.5	27.5	28.4	31.6	25.4	29.1
AMERICA	40.6	44.5	40.1	42.3	41.5	39.3
OCEANIA	38.3	51.4	42.2	44.9	45.8	45.9
WESTERN WORLD	39.1	43.5	38.2	40.6	37.9	38.0

SECONDARY PRODUCTION

000 tons

	1963	1968	1973	1974	1975	1976
Germany F.R.	252.1	334.2	357.2	348.3	290.0	362.3
France	137.6	141.5	182.8	139.1	118.7	183.6
Italy	116.0	174.0	195.0	208.7	157.2	224.8
Netherlands	34.4	43.5	25.4	27.8	20.6	20.0
Belgium-Lux.	67.8	143.5	96.2	129.2	93.1	106.0
United Kingdom	246.7	288.5	272.5	255.1	200.3	232.7
Ireland	-	-	-	-	-	-
Denmark	0.6	3.9	0.3	2.8	2.7	2.8
E E C	855.2	1,129.1	1,129.4	1,111.0	882.6	1,132.2
Other Europe	140.6	197.9	284.0	297.7	267.2	291.2
EUROPE	995.8	1,327.0	1,413.4	1,408.7	1,149.8	1,423.4
South Africa	10.0	11.0	23.0	25.0	23.0	19.0
Other Africa	4.0	4.0	9.0	9.0	7.0	8.0
AFRICA	14.0	15.0	32.0	34.0	30.0	27.0
Japan	253.1	386.2	549.3	570.2	416.9	454.0
Other Asia	20.0	21.0	50.0	50.0	36.0	57.0
ASIA	273.1	407.2	599.3	620.2	452.9	511.0
Brasil	7.0	20.0	42.0	57.0	45.0	52.0
Canada	40.0	72.0	68.0	62.0	45.0	46.0
USA	998.9	1,217.8	1,359.8	1,317.1	951.1	1,100.4
Other America	40.0	40.0	55.0	68.0	53.0	70.0
AMERICA	1,085.9	1,349.8	1,524.8	1,504.1	1,094.1	1,268.4
OCEANIA	41.6	72.2	76.1	75.0	64.0	70.0
WESTERN WORLD	2,410.4	3,171.2	3,645.6	3,642.0	2,790.8	3,299.8

SECONDARY REFINED COPPER PRODUCTION

'000 tons

	1963	1968	1973	1974	1975	1976
Germany F.R.	113.8	176.8	165.4	204.0	173.4	205.0
France	21.0	20.0	22.0	18.0	15.0	17.0
Italy	13.0	18.0	17.0	21.7	22.2	26.8
Belgium-Lux.	40.0	113.0	58.0	96.0	60.0	80.0
United Kingdom	109.1	148.0	95.0	91.0	76.0	85.7
E E C	296.9	475.8	357.4	430.7	346.6	414.5
Other Europe	41.2	61.8	76.0	88.0	85.0	92.0
EUROPE	338.1	537.6	433.4	518.7	431.6	506.5
ASIA (Japan)	43.1	71.2	133.3	127.2	76.9	95.0
Brasil	-	8.0	25.0	34.0	27.0	31.0
Canada	-	39.0	32.0	25.0	25.0	26.0
USA	261.4	364.3	402.9	439.0	299.9	320.0
Mexico	-	-	5.0	6.0	7.0	8.0
AMERICA	261.4	411.3	464.9	504.0	358.9	385.0
AUSTRALIA	16.6	35.2	33.1	32.0	28.0	31.0
WESTERN WORLD	659.2	1,055.3	1,064.7	1,181.9	895.4	1,017.5

DIRECT USE OF SCRAP (1)

000 tons

	1963	1968	1973	1974	1975	1976
Germany F.R.	138.3	157.4	191.8	144.3	116.6	157.3
France	116.6	121.5	160.8	121.1	103.7	166.6
Italy	103.0	156.0	178.0	187.0	135.0	198.0
Netherlands	34.4	43.5	25.4	27.8	20.6	20.0
Belgium-Lux.	27.8	30.5	38.2	33.2	33.1	26.0
United Kingdom	137.6	140.5	177.5	164.1	124.3	147.0
Ireland	-	-	-	-	-	-
Denmark	0.6	3.9	0.3	2.8	2.7	2.8
E E C	558.3	653.3	772.0	680.3	536.0	717.7
Other Europe	99.4	136.1	208.0	209.7	182.2	199.2
EUROPE	657.7	789.4	980.0	890.0	718.2	916.9
South Africa	10.0	11.0	23.0	25.0	23.0	19.0
Other Africa	4.0	4.0	9.0	9.0	7.0	8.0
AFRICA	14.0	15.0	32.0	34.0	30.0	27.0
Japan	210.0	315.0	416.0	443.0	340.0	359.0
Other Asia	20.0	21.0	50.0	50.0	36.0	57.0
ASIA	230.0	336.0	466.0	493.0	376.0	416.0
Brasil	7.0	12.0	17.0	23.0	18.0	21.0
Canada	40.0	33.0	36.0	37.0	20.0	20.0
USA	737.5	853.5	956.9	878.1	651.2	780.4
Other America	40.0	40.0	50.0	62.0	46.0	62.0
AMERICA	824.5	938.5	1,059.9	1,000.1	735.2	883.4
OCEANIA	25.0	37.0	43.0	43.0	36.0	39.0
WESTERN WORLD	1,751.2	2,115.9	2,580.9	2,460.1	1,895.4	2,282.3

(1) Including alloy ingots

TABLE EEC-1

PRODUCTION AND CONSUMPTION OF COPPER IN THE EEC - 1967-1976

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. Smelter production	216.5	260.7	247.6	273.5	232.2	273.0	303.7	329.6	267.1	296.0
1.1. From domestic ores	1.2	1.4	1.6	1.5	1.5	1.3	1.4	1.7	2.0	1.6
1.2. From imported ores	89.1	104.3	101.2	92.9	102.4	137.0	173.8	188.8	194.9	206.1
1.3. From scrap and waste	126.2	155.0	144.8	179.1	128.3	134.7	128.5	139.1	70.2	88.3
2. Net import of blister	429.1	402.8	351.6	376.3	399.8	356.9	387.8	345.4	401.8	406.1
3. Changes in stocks	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
4. Domestic availability of refined copper from blister	542.7	597.7	605.2	653.5	605.5	583.1	677.0	667.1	642.3	555.3
5. Secondary production of refined copper	260.4	320.8	283.7	281.8	253.8	254.8	228.9	291.6	276.4	326.2
5a. Leach cathodes from Zaire	81.4	71.4	51.6	61.6	69.9	81.9	89.2	69.3	48.4	183.1
6. Total production of refined copper	884.5	989.9	940.5	996.9	929.2	919.8	995.1	1,028.0	967.1	1,064.6
7. Net imports of refined copper	710.2	856.1	1,077.1	1,120.1	1,020.4	1,198.6	1,117.6	1,215.2	1,219.9	1,190.6
8. Changes in stocks: Increase-	-	19.9	61.0	71.6	-	37.5	-	64.2	223.0	84.5
Decrease+	46.6	-	-	-	5.2	-	71.9	-	-	-
9. Consumption of refined Cu	1,641.3	1,826.1	1,956.6	2,045.4	1,954.8	2,080.9	2,184.6	2,179.0	1,964.0	2,170.7
10. Direct use of scrap and consumption of alloy ingots	632.9	653.3	736.4	636.8	633.4	602.6	772.0	680.3	536.0	717.7
11. Total copper consumption	2,274.2	2,479.4	2,693.0	2,686.2	2,588.2	2,683.5	2,956.6	2,859.3	2,500.0	2,888.4
12. Secondary production as % of total consumption	44.9	45.5	43.3	41.1	39.4	37.0	38.2	38.9	35.3	39.2

TABLE EEC-2

DOMESTIC SCRAP ARISING IN THE EEC - 1967-1976

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	1,020.3	1,129.1	1,166.7	1,103.7	1,019.0	993.7	1,129.4	1,111.0	882.6	1,132.2
Δ (E-I) alloy ingots	- 5.3	- 14.4	- 19.1	- 20.5	- 17.3	-20.2	- 12.8	- 15.1	- 8.8	
Δ (E-I) scrap	- 51.1	-128.2	-126.6	-140.6	- 72.5	-40.2	- 85.4	- 74.7	-55.1	
Δ (E-I) ashes and residues	-	- 3.0	- 6.2	- 7.7	- 3.0	- 9.3	- 16.3	- 26.8	-23.6	
Copper recovered from domestic scrap	963.9	983.5	1,014.8	934.9	926.2	924.0	1,014.9	994.4	795.1	
of which:										
• new scrap: at fabricating stage	450.1	532.5	554.7	527.8	514.6	564.1	582.2	534.7	453.2	
total	546.1	642.2	668.5	635.8	617.0	670.2	696.9	643.8	542.6	
• old scrap	417.8	341.3	346.3	299.1	309.2	253.8	318.0	350.6	252.5	
Copper recovered from domestic scrap as % of total consumption	42.4	39.7	37.7	34.8	35.8	34.4	34.3	34.8	31.8	

TABLE D-1

PRODUCTION AND CONSUMPTION OF COPPER IN GERMANY F.R. 1967-1976

'000 tons

		1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. Smelter production		149.1	191.8	184.4	213.1	161.5	203.5	232.5	245.2	224.6	244.5
1.1. From domestic ores		1.2	1.4	1.6	1.5	1.5	1.3	1.4	1.7	2.0	1.6
1.2. From imported ores		72.1	94.3	91.2	82.9	84.4	124.0	157.8	172.8	174.9	192.1
1.3. From old and new scrap		75.8	96.1	91.6	128.7	75.6	78.2	73.3	70.7	47.7	50.8
2. Imports of blister	+	164.8	146.4	144.8	133.3	155.7	117.9	121.2	98.2	119.8	130.8
3. Exports of blister	-	3.0	4.0	1.2	0.9	0.6	20.9	32.8	50.2	38.5	38.9
4. Changes in stocks: Increase / Decrease	- / +	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
5. Domestic availability of refined copper from blister (1+2+3+4)		309.8	326.7	319.6	336.1	308.2	294.6	314.6	290.3	296.5	292.4
6. Production of secondary refined copper (1)		45.9	80.7	82.5	69.7	91.9	103.9	92.1	133.3	125.7	154.2
7. Production of refined copper		355.7	407.4	402.1	405.8	400.1	398.5	406.7	423.6	422.2	446.6
8. Imports of refined copper	+	286.3	352.8	362.3	399.7	375.4	404.5	414.4	449.7	404.9	409.7
9. Exports of refined copper	-	161.5	143.8	108.5	93.5	140.7	121.7	119.5	115.9	97.3	66.3
10. Changes in stocks: Increase	-	-	7.6	0.2	14.5	4.3	9.2	-	26.4	95.2	45.4
Decrease	+	20.7	-	-	-	-	-	25.6	-	-	-
11. Consumption		501.2	608.8	655.7	697.5	630.5	672.1	727.2	731.0	634.6	744.6
12. Direct use of scrap / 13. Consumption of alloy ingots		167.6	157.4	202.3	157.2	174.2	147.6	191.8	144.3	116.6	157.3
14. Total consumption		668.8	766.2	858.0	854.7	804.7	819.7	919.0	875.3	751.2	901.9
15. Secondary production (1.3+6+12+13) as % of production (7+12+13)		55.3	59.2	62.3	63.2	59.5	60.4	59.7	61.3	53.8	60.0
16. Secondary production from domestic sources as % of production		34.8	33.8	36.1	37.8	39.1	39.4	34.4	39.4	31.3	
17. Secondary production as a % of total consumption		43.3	43.6	43.9	41.6	42.5	40.2	38.9	39.8	38.6	40.2
18. Secondary production from domestic sources as % of total consumption		27.2	24.9	25.4	24.9	27.9	26.2	22.4	25.6	22.5	

(1) Excluding production from secondary blister (item 1.3)

TABLE D-2

DOMESTIC ARISING SCRAP

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	289.3	334.2	376.4	355.6	341.7	329.7	357.2	348.3	290.0	362.3
Δ (E-I) alloy ingots	-27.9	-35.5	-40.5	-34.9	-29.6	-33.7	-31.6	-17.7	-22.7	-26.8
Δ (E-I) scrap	-44.9	-72.2	-82.9	-70.4	-42.4	-33.8	-67.2	-54.4	-53.8	-64.0
Δ (E-I) ashes and residues	- 2.2	- 5.6	- 8.1	- 7.1	- 8.0	-10.2	-14.2	- 8.8	-12.9	
Copper recovered from domestic scrap	214.3	220.9	244.9	243.2	261.7	252.0	244.2	267.4	200.6	
of which:										
. new scrap										
at fabricating stage	106.1	154.8	181.6	176.7	165.6	173.6	193.5	156.4	146.4	179.4
total	131.2	186.9	218.6	212.2	199.3	207.5	230.8	189.4	175.5	215.5
. old scrap	83.1	34.0	26.3	31.0	62.4	44.5	13.4	78.0	25.1	
Copper recovered from domestic scrap as % of total consumption	32.0	28.8	28.5	28.5	32.5	30.7	26.6	30.5	26.7	

PRODUCTION AND CONSUMPTION OF COPPER IN FRANCE 1967-1976

TABLE F-1

'000 tons

		1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. Smelter production		7.6	7.2	7.4	6.4	5.6	3.5	5.2	5.9	2.5	3.5
1.1. From domestic ores		-	-	-	-	-	-	-	-	-	-
1.2. From imported ores		-	-	-	-	-	-	-	-	-	-
1.3. From old and new scrap		7.6	7.2	7.4	6.4	5.6	3.5	5.2	5.9	2.5	3.5
2. Imports of blister	+	11.6	16.4	15.8	16.5	16.4	15.7	18.3	23.7	26.8	23.9
3. Exports of blister	-	12.5	12.7	10.0	8.6	6.7	6.3	6.8	7.8	2.2	4.4
4. Changes in stocks: Increase	-	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Decrease	+										
5. Domestic availability of refined copper from blister (1+2+3+4)		19.6	23.7	22.4	20.0	14.8	15.6	16.3	31.8	27.1	25.8
6. Production of secondary refined copper (1)		17.4	12.8	14.6	13.6	14.4	14.5	16.8	12.1	12.5	13.5
7. Production of refined copper		37.0	36.5	37.0	33.6	29.2	30.1	33.1	43.9	39.6	39.3
8. Imports of refined copper	+	235.3	257.9	323.1	315.4	301.6	366.4	377.3	375.3	350.2	338.1
9. Exports of refined copper	-	9.3	10.6	6.2	3.0	3.6	1.4	3.6	2.8	4.4	6.5
10. Changes in stocks: Increase	-	-	-	19.1	15.3	-	4.9	-	2.2	20.9	3.8
Decrease	+	8.3	9.1	-	-	16.4	-	1.0	-	-	-
11. Consumption		271.3	292.9	334.8	330.7	343.6	390.2	407.8	414.2	364.5	367.1
12. Direct use of scrap 13. Consumption of alloy ingots		122.8	121.5	156.0	128.3	124.4	105.6	160.8	121.1	103.7	166.6
14. Total consumption		394.1	414.4	490.8	459.0	468.0	495.8	568.6	535.3	468.2	533.7
15. Secondary production (1.3+6+12+13) as % of production (7+12+13)		92.5	89.6	92.2	91.6	94.0	91.1	94.3	84.3	82.8	89.2
16. Secondary production from domestic sources as % of production		85.9	81.4	86.6	84.5	86.1	81.1	87.0	71.6	72.0	
17. Secondary production as % of total consumption		40.0	34.1	36.3	32.3	30.9	24.9	32.1	26.0	25.4	34.4
18. Secondary production from domestic sources as % of total consumption		34.8	31.1	34.0	29.8	28.2	22.2	29.7	22.1	22.0	

(1) Excluding production from secondary blister (item 1.3)

TABLE F-2

DOMESTIC ARISING SCRAP

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	147.8	141.5	178.0	148.3	144.4	123.6	182.8	139.1	118.7	183.6
Δ (E-I) alloy ingots	+ 2.0	+ 0.2	+ 0.5	+ 1.0	–	– 0.8	+ 0.9	– 2.7	– 2.1	
Δ (E-I) scrap	+23.4	+15.4	+23.0	+29.9	+21.4	+35.8	+50.4	+45.3	+33.6	
Δ (E-I) ashes and residues	+ 1.7	+ 2.2	+ 1.1	+ 0.7	+ 1.1	+ 0.6	+ 4.9	+ 4.0	+ 2.9	
Copper recovered from domestic scrap	174.9	159.3	202.6	179.9	166.9	159.2	239.0	185.7	153.1	
of which:										
• new scrap at fabricating stage	66.5	69.2	83.2	78.4	77.9	82.5	96.7	91.6	74.9	92.7
total	80.6	83.2	100.4	93.6	92.9	97.0	115.5	108.1	88.3	111.2
• old scrap	98.8	76.1	102.2	86.3	74.0	62.2	123.5	77.6	64.8	
Copper recovered from domestic scrap as % of total consumption	44.4	38.4	41.3	39.2	35.7	32.1	42.0	34.7	32.7	

PRODUCTION AND CONSUMPTION OF COPPER IN ITALY 1967-1976

TABLE I-1

'000 tons

		1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. Smelter production		-	-	-	-	-	-	-	-	-	-
1.1. From domestic ores		-	-	-	-	-	-	-	-	-	-
1.2. From imported ores		-	-	-	-	-	-	-	-	-	-
1.3. From old and new scrap		-	-	-	-	-	-	-	-	-	-
2. Imports of blister	+	-	-	-	-	-	-	-	-	-	-
3. Exports of blister	-	-	-	-	-	-	-	-	-	-	-
4. Changes in stocks: Increase	-	-	-	-	-	-	-	-	-	-	-
Decrease	+	-	-	-	-	-	-	-	-	-	-
5. Domestic availability of refined copper from blister (1+2+3+4)		-	-	-	-	-	-	-	-	-	-
6. Production of secondary refined copper		17.5	18.0	16.5	13.7	9.5	15.0	17.0	21.7	22.2	26.8
7. Production of refined copper		17.5	18.0	16.5	13.7	9.5	15.0	17.0	21.7	22.2	26.8
8. Imports of refined copper (1)	+	202.9	200.7	237.2	273.1	261.8	286.5	271.6	306.7	284.1	293.4
9. Exports of refined copper (1)	-	5.5	5.5	2.8	5.4	4.4	3.7	7.1	8.0	3.3	2.3
10. Changes in stocks: Increase	-	-	-	12.9	9.4	-	7.8	-	4.4	4.0	-
Decrease	+	7.1	9.5	-	-	3.1	-	18.5	-	-	4.1
11. Consumption		222.0	226.0	238.0	274.0	270.0	290.0	300.0	316.0	299.0	322.0
12. Direct use of scrap 13. Consumption of alloy ingots		139.0	156.0	164.0	169.0	145.0	152.0	178.0	187.0	135.0	198.0
14. Total consumption		361.0	382.0	402.0	443.0	415.0	442.0	478.0	503.0	434.0	520.0
15. Secondary production (1.3+6+12+13) as % of production (7+12+13)		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
16. Secondary production from domestic sources as % of production		68.8	69.7	68.9	66.4	62.6	63.6	67.2	62.2	67.7	70.8
17. Secondary production as a % of total consumption		43.4	45.5	44.9	41.2	37.2	37.8	40.8	41.5	36.2	43.2
18. Secondary production from domestic sources as % of total consumption		29.8	31.7	30.9	27.4	23.3	24.0	27.4	25.8	24.5	30.3

(1) Including unrefined copper

TABLE I-2

DOMESTIC ARISING SCRAP

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	156.5	174.0	180.5	182.7	154.5	167.0	195.0	208.7	157.2	224.8
Δ (E-I) alloy ingots	-21.8	-16.0	-16.9	-22.8	-19.5	-20.2	-15.9	-23.0	- 8.4	-13.1
Δ (E-I) scrap	-26.0	-33.0	-35.4	-36.1	-33.6	-35.5	-39.9	-41.8	-37.6	-39.4
Δ (E-I) ashes and residues	-	-	+ 0.4	+ 3.9	+ 5.3	+ 2.2	+ 3.7	+ 0.8	+ 3.9	+ 3.8
Copper recovered from domestic scrap	108.7	125.0	128.6	127.7	106.7	113.5	142.9	144.7	115.1	176.1
of which:										
• new scrap at fabricating stage	69.3	70.3	74.5	84.0	76.2	85.3	95.2	99.7	78.5	100.9
total	84.0	86.0	91.0	101.5	91.5	102.2	114.5	120.1	94.2	122.2
• old scrap	24.7	39.0	37.6	26.2	15.2	11.3	28.4	24.6	20.9	53.9
Copper recovered from domestic scrap as % of total consumption	30.1	32.7	32.0	28.8	25.7	25.7	29.9	28.8	26.5	33.9

TABLE NL-1

PRODUCTION AND CONSUMPTION OF COPPER IN THE NETHERLANDS 1967-1976

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. + Imports	31.0	36.9	40.1	39.7	44.2	41.9	48.6	47.3	48.0	60.0
2. - Exports	1.6	4.2	1.5	1.4	0.9	5.2	7.2	16.8	10.5	7.0
3. Changes in stocks:										
- Increase	-	-	0.2	-	1.7	-	3.2	-	0.3	7.0
+ Decrease	-	1.6	-	0.9	-	-	-	7.2	-	-
4. Refined consumption	29.4	34.3	38.4	39.2	41.6	36.7	38.2	37.7	37.2	46.0
5. + Direct use of scrap	33.8	43.5	34.4	29.2	25.9	23.5	25.4	27.8	20.6	20.0
6. Total consumption	63.2	77.8	72.8	68.4	67.5	60.2	63.6	65.5	57.8	66.0

TABLE NL-2

DOMESTIC ARISING SCRAP

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	33.8	43.5	34.4	29.2	25.9	23.5	25.4	27.8	20.6	20.0
Δ (E-I) alloy ingots	+ 1.7	+ 2.0	+ 2.8	+ 2.3	+ 1.5	+ 1.0	+ 2.9	+ 3.0	+ 2.0	+1.5
Δ (E-I) scrap	+19.4	+19.1	+22.8	+18.1	+17.1	+26.2	+27.8	+22.2	+11.4	
Δ (E-I) ashes and residues	n.a.	n.a.	n.a.	+ 1.3	+ 2.0	+ 0.9	+ 1.0	+ 1.7	- 0.5	
Copper recovered from domestic scrap	54.9	64.6	60.0	50.9	46.5	51.6	57.1	54.7	33.5	
Copper recovered from domestic scrap as % of total consumption	86.9	83.0	82.4	74.4	68.9	85.7	89.8	83.5	58.0	

PRODUCTION AND CONSUMPTION OF COPPER IN BELGIUM/LUXEMBOURG 1967-1976

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. Smelter production	59.8	61.7	55.8	54.0	65.1	66.0	66.0	78.5	40.0	48.0
1.1. From domestic ores	-	-	-	-	-	-	-	-	-	-
1.2. From imported ores	17.0	10.0	10.0	10.0	18.0	13.0	16.0	16.0	20.0	14.0
1.3. From scrap and waste	42.8	51.7	45.8	44.0	47.1	53.0	50.0	62.5	20.0	34.0
2. + Imports of blister	236.6	214.4	157.8	198.4	203.6	197.4	223.7	201.2	214.5	229.9
3. - Exports of blister	-	-	0.2	3.2	1.8	0.1	2.1	5.9	1.2	1.4
4. Changes in stocks (-) Increase / (+) Decrease	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
5. Domestic availability of refined copper from blister (1+2+3+4)	177.4	197.6	213.9	248.0	233.0	213.3	270.3	275.9	243.2	185.6
6. Secondary production of refined copper (from old and new scrap) (1)	46.2	61.3	21.2	28.0	9.9	19.0	8.0	33.5	40.0	46.0
6a. Leach cathodes from Zaire	81.4	71.4	51.6	61.6	69.9	81.9	89.2	69.3	48.4	183.1
7. Total production of refined copper	305.0	330.3	286.7	337.6	312.8	314.2	367.5	378.7	331.6	414.7
8. + Imports of refined copper (excluding 6a)	58.1	117.7	134.9	121.5	119.1	115.3	124.9	118.5	139.5	119.1
9. Exports of refined copper	286.4	309.8	254.3	295.2	273.4	265.8	319.4	288.9	248.6	307.5
10. Changes in stocks (-) Increase	-	16.2	29.3	18.9	11.5	10.7	8.6	30.1	48.3	-
(+) Decrease	22.3	-	-	-	-	-	-	-	-	1.8
11. Consumption of refined copper (7+8+9+10)	99.0	122.0	138.0	145.0	147.0	153.0	164.4	178.2	174.2	228.1
12. Direct use of scrap / 13. Consumption of alloy ingots	26.1	30.5	34.1	35.9	35.9	37.8	38.2	33.2	33.1	26.0
14. Total copper consumption	125.1	152.5	172.1	180.9	182.9	190.8	202.6	211.4	207.3	254.1
15. Secondary production (1.3+6+12+13) as % of production (7+12+13)	34.7	39.8	31.5	28.9	26.6	31.2	23.7	31.3	25.5	24.1
16. Secondary production from domestic sources as % of production	n.a.	n.a.	n.a.	0.6	9.6	16.4	1.2	3.2	6.8	
17. Secondary production (1.3+6+12+13) as % of total consumption	91.9	94.1	73.3	74.4	63.2	71.8	47.5	61.1	44.9	41.7
18. Secondary production from domestic sources as % of total consumption	n.a.	n.a.	n.a.	1.3	18.4	30.3	2.4	6.2	12.0	

(1) Excluding production from secondary blister (item 1.3)

TABLE B/L-2

DOMESTIC ARISING SCRAP

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	115.1	143.5	101.1	107.9	92.9	109.8	96.2	129.2	93.1	106.0
Δ (E-I) alloy ingots	n.a.	n.a.	+ 2.2	+ 1.6	+ 2.1	+ 0.5	+ 0.8	+ 1.0	+ 1.9	+ 1.5
Δ (E-I) scrap	-29.5	-59.2	-55.2	-81.7	-41.4	-34.3	-58.1	-72.6	-40.5	
Δ (E-I) ashes and residues	n.a.	n.a.	n.a.	- 7.4	- 3.5	- 3.4	-11.8	-24.2	-15.9	
Copper recovered from domestic scrap	85.6	84.3	48.1	20.4	50.1	72.6	27.1	33.4	38.6	
Copper recovered from domestic scrap as % of total consumption	68.4	55.3	27.9	11.3	27.4	38.1	13.4	15.8	18.6	

DOMESTIC SCRAP ARISING IN BENELUX

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	148.9	187.0	135.5	137.1	118.8	133.3	121.6	157.0	113.7	126.0
Δ (E-I) alloy ingots	+ 1.7	+ 2.0	+ 5.0	+ 3.9	+ 3.6	+ 1.5	+ 3.7	+ 4.0	+ 3.9	+ 3.0
Δ (E-I) scrap	-10.1	-40.1	-32.4	-63.6	-24.3	- 8.1	-30.3	-50.4	-29.1	
Δ (E-I) ashes and residues	n.a.	n.a.	n.a.	- 6.1	- 1.5	- 2.5	-10.8	-22.5	-16.4	
Copper recovered from domestic scrap	140.5	148.9	108.1	71.3	96.6	124.2	84.2	88.1	72.1	
of which:										
• new scrap at fabricating stage							33.8	35.7	32.3	
total							43.3	47.2	41.1	
• old scrap							40.9	40.9	31.0	

TABLE UK-1

PRODUCTION AND CONSUMPTION OF COPPER IN THE UK 1967-1976

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. Smelter production (blister)	-	-	-	-	-	-	-	-	-	-
1.1. Domestic ores	-	-	-	-	-	-	-	-	-	-
1.2. Imported ores	-	-	-	-	-	-	-	-	-	-
1.3. Old and new scrap	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
2. + Imports of blister	31.6	42.3	44.6	40.8	33.2	53.2	66.3	86.2	82.6	66.2
3. - Exports of blister	-	-	-	-	-	-	-	-	-	-
4. Changes in stocks: (-) Increase	-	-	-	-	-	-	-	17.1	7.1	14.7
(+) Decrease	4.3	7.4	4.9	8.6	16.3	6.4	9.5	-	-	-
5. Domestic availability of refined copper from imported blister (2+3+4)	35.9	49.7	49.3	49.4	49.5	59.6	75.8	69.1	75.5	51.5
6. Secondary production of refined copper (from scrap)	133.4	148.0	148.9	156.8	128.1	102.4	95.0	91.0	76.0	85.7
7. Total production of refined copper	169.3	197.7	198.2	206.2	177.6	162.0	170.8	160.1	151.5	137.2
8. + Imports of refined copper	419.6	416.3	416.7	409.5	366.1	395.5	399.4	380.7	369.0	367.9
9. - Exports of refined copper	62.4	56.3	67.6	44.9	28.1	18.8	66.1	35.0	15.7	12.3
10. Changes in stocks: (-) Increase	12.2	18.5	0.5	17.1	-	4.1	-	8.9	54.3	35.2
(+) Decrease	-	-	-	-	1.7	-	37.1	-	-	-
11. Consumption of refined copper (7+8+9+10)	514.3	539.2	546.8	553.7	517.3	534.6	541.2	496.9	450.5	457.6
12. Direct use of scrap } 13. Consumption of alloy ingots }	138.1	140.5	143.6	119.9	127.8	134.0	177.5	164.1	124.3	147.0
14. Total copper consumption	652.4	679.7	690.4	673.6	645.1	668.6	718.7	661.0	574.8	604.6
15. Secondary production (1.3+6+12+13) as % of production (7+12+13)	88.3	85.3	85.6	84.8	83.8	79.9	78.2	78.7	72.6	81.9
16. Secondary production from domestic sources as % of production	87.8	83.4	83.4	81.6	80.0	74.3	72.6	69.2	65.3	
17. Secondary production (1.3+6+12+13) as % of total consumption	41.6	42.4	42.4	41.1	39.7	35.4	37.9	38.6	34.8	38.5
18. Secondary production from domestic sources as % of total consumption	41.4	41.5	41.3	39.5	37.9	32.9	35.2	33.9	31.4	

DOMESTIC ARISING SCRAP

TABLE UK-2

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	271.5	288.5	292.5	276.4	255.9	236.4	272.5	255.1	200.3	232.7
Δ (E-I) alloy ingots	+40.4	+34.1	+32.4	+31.2	+26.8	+31.4	+29.3	+23.4	+20.4	+18.5
Δ (E-I) scrap	+ 0.3	- 5.2	- 4.9	- 7.6	- 2.6	- 3.0	- 9.3	+13.6	+22.6	
Δ (E-I) ashes and residues	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Copper recovered from domestic scrap	312.2	317.4	320.0	300.0	280.1	264.8	292.5	292.1	243.3	
of which:										
• new scrap at fabricating stage	136.4	149.5	150.5	141.2	133.3	140.6	156.2	142.1	115.0	125.4
total	163.6	178.9	180.2	169.1	159.4	166.5	185.4	169.1	136.5	149.6
• old scrap	148.6	138.5	139.8	130.9	120.7	98.3	107.1	123.0	106.8	
Copper recovered from domestic scrap as % of total consumption	47.9	46.7	46.3	44.5	43.4	39.6	40.7	44.2	42.3	

TABLE IRL-1

DOMESTIC SCRAP ARISING IN IRELAND 1967-1976

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	–	–	–	–	–	–	–	–	–	–
Δ (E–I) alloy ingots	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	0.7	0.2	
Δ (E–I) scrap	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	4.2	3.9	
Export unalloyed scrap	2.8	2.9	3.2	3.3	4.6	2.8	3.2			
Δ (E–I) ashes and residues	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	–	–	
Copper recovered from domestic scrap	2.8	2.9	3.2	3.3	4.6	2.8	3.2	4.9	4.1	

TABLE DK-1

PRODUCTION AND CONSUMPTION OF COPPER IN DENMARK 1967-1976

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
1. Production of unalloyed and alloyed copper	6.3	3.9	3.8	3.6	3.7	3.7	3.0	3.6	2.5	3.0
2. + Imports	3.8	4.2	3.9	4.7	3.9	6.0	4.0	5.5	4.5	5.5
3. - Exports	0.5	1.3	0.8	1.8	2.7	3.4	1.2	1.6	0.5	0.8
4. Total consumption	9.6	6.8	6.9	6.5	4.9	6.3	5.8	7.5	6.5	7.7

TABLE DK-2

DOMESTIC ARISING SCRAP

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Secondary production	6.3	3.9	3.8	3.6	3.7	3.7	3.0	3.6	2.5	3.0
Δ (E-I) alloy ingots	+0.3	+0.8	+0.4	+1.1	+1.4	+1.6	+0.8	+0.2	-0.1	-0.1
Δ (E-I) scrap	+3.4	+4.0	+2.8	+3.9	+4.4	+1.6	+7.7	+8.8	+5.3	
Δ (E-I) ashes and residues	+0.5	+0.4	+0.4	+0.9	+0.1	+0.6	+0.1	-0.3	-1.1	
Copper recovered from domestic scrap	10.5	9.1	7.4	9.5	9.6	7.5	11.6	12.3	6.6	

5.3.1 Scrap availability

Scrap availability in the Western world amounts to 3.6 million tons per year for the period 1972-74.

According to UNCTAD Expert Group on Copper, old scrap recycled in 1973-74 is estimated to amount to 1.3-1.4 M tons.

After taking into consideration historic copper consumption, use of goods in non-recoverable situations in industrialized and non-industrialized countries, and dispersive uses, it is estimated that around 2.5 M tons of copper-base scrap may be potentially recovered annually from obsolete plants, equipment and consumer goods. Therefore, the amount of copper-base scrap available to industry worldwide could be increased by some 1 M tons per annum if the combination of economic circumstances were such as to provide the maximum incentive to increased collection.

Concerning scrap availability in the EEC countries, tables II show domestic scrap arisings in the EEC member countries and the EEC as a whole, brokendown in old and new scrap.

The availability of copper from domestic scrap in the EEC in the years 1968-74 amounted to 1 M tons, a figure which remained practically constant since 1968, and not in accordance with increase in total consumption.

From 1968 to 1975 the amount of old scrap recycled on the whole in the EEC remained around 300,000-350,000 tons. The percentage of old scrap recycled in respect of secondary production is of 30 % as to a world figure estimated to be 40 % ; therefore, compared with total consumption, we have a 11 %-12 % value against, for example, a 20 % in the United States.

There are different opinions regarding the possibility of an increase in old scrap recycling rate. According to some experts, the size of the old scrap market will undergo slight variations : of the order of 200,000 tons in 5-10 years. This opinion is linked to the consideration that, with the present structure of collection methods, the technology available for metal segregation, the standards of quotations of scrap and copper prices, as well as the level of economic activity, to expect an increase in the recovery rate would not be justified.

On the other hand, in the study carried out on US demand for copper by Bureau of Mines "Mineral facts and problems, 1975 Edition", it is reported that in consideration of the anticipated increased emphasis on and incentives for recycling, the portion of supply furnished by old scrap was increased from the approximate 20 % of recent years to 25 % for 1985 and 30 % for 2000. This resulted in an annual probable growth rate from 1973 to 2000 of 5 % for secondary supply. The remainder supplied by primary material will grow at an annual rate of 2.9 %.

Instruments could be the use of highly sophisticated methods of collection, segregation and marketing, and re-using, in different fields, of a wide range of copper and copper-alloyed scrap.

According to the proposals contained in the report of the United Kingdom Delegation submitted to the UNCTAD team of experts on copper, besides the instruments suggested, an important role could be played by a "copper buffer stock" whose function should be the following, as reported in the draft paper :

"would typically be, by quick and flexible intervention on the world market, backed by export controls, to help adjust supply to demand in such a way as to influence and moderate movements in the world price quotation for primary metal.

In the longer term, however, if the effect of a buffer-stock arrangement were to raise the trend of world copper prices in real terms there is a possibility that the importing countries would be encouraged to maximise their use of scrap, and to step up the development of substitutes.

While most of the scrap which is economically recoverable in present conditions is probably being recovered already, any substantial change in the economic balance between primary metal and scrap could stimulate further recovery of scrap.

As the above analysis shows, there is apparent scope for the recovery in the longer run of further amounts which could be very substantial in relation even to an increasing global consumption of copper.

It is possible therefore that from day to day the existence of the copper-base scrap trade would not have much impact on the operation of a buffer stock. There is no short-term possibility of the diversion of a major proportion of consumption from primary metal to scrap.

On the other hand, if, as described above, economic conditions developed which could encourage the recovery and use of important extra quantities of scrap, this could in the long term and against the background of an assumed increase in world output of primary metal, increase the pressure on buffer stock in its attempts to stabilize the world market for primary metal and require increasing buffer stock capacity for this purpose."

5.3.2 <u>Scrap recycling rate</u>

As to supply of raw materials, new scrap is considered as primary metal, and the recycling rate of new scrap is generally estimated to be 100 %.

Old scrap represents the source of raw materials supply as an alternative to primary metal. It is therefore essential to assess the present and future availability of old scrap.

To this end we examined several studies conducted by different organizations on old scrap recycling rates, we calculated the present recycling rates in the EEC countries, and we carried out a general examination of recycling technologies.

The results of these studies are summarised as follows :

a) USA - Bureau of Mines - 108622

Recovery of secondary copper and zinc in the United States

1970 - obsolete copper products	sh. tons	1,877,000
1970 - recovered copper products	sh. tons	577,000

The average recovery rate of products marketed on a national level is of 30.7 %.

Recovery rates by sector of employment are the following :

. building	28.7 %
. transport	30.6 %
. consumer and general goods	18.4 %
. industrial machinery and equipment	36.9 %
. electric and electronic materials	36.0 %

Higher rates generally occur for :

. electric materials 42 % (bare wires, communication, output)
. tubes for transport 44 % for industrial machinery 40 %
. tubes for building 36 %

30 %-50 % of unrecovered copper is found in urban refuses amounting to 190 million tons/year and formed by :

. domestic refuses	56.5 %
. commercial refuses	19.0 %
. industrial refuses	11.0 %
. demolition refuses	3.5 %

The difference to 100 % copper recovered and copper disposed of in solid waste system is made up by losses occuring during the segregation processes, or by corrosion, abrasion, oxidation losses (i.e. all unrecoverable).

b) Battelle Memorial Institute, Columbus Laboratories

A study to identify opportunities for increased solid waste utilization EPA - SW - 40 D 1-72.

The study was performed on behalf of the National Association of Secondary Material Industries Ins. (NASMI) on a grant program by the Office of Solid Waste Management, Environmental Protection Agency. The study concerns the development of greater solid waste utilization through analysis of the secondary material industry, its sources of supply, its consuming market, and its economic and technological problems.

The materials examined are aluminium, copper, lead, zinc, nickel and stainless steel, precious metals, paper and textiles.

Problems inhibiting increased recycling of these materials are identified, and recommended actions are proposed.

The following table illustrates the estimated copper scrap recycling for both prompt industrial and obsolete by kind and type of scrap and the average percent recycling on obsolete products in 1969.

ESTIMATED COPPER SCRAP RECYCLING, 1969

Kind and Type of Scrap(1)	Copper Content Available for Recycling(2) (thousand tons)	Copper Content Recycled(3) (thousand tons)	Percent Recycled	Copper Content Not Recycled (thousand tons)
Electric Wire and Copper Tube				
Prompt Industrial	379.7	379.7(12)	100	--
Obsolete	471.2	319.4	68	151.8
TOTAL	850.9	699.1	82	151.8
Magnet Wire				
Prompt Industrial(4)	--	--	--	--
Obsolete(5)	158.0	13.5	9	144.5
TOTAL	158.0	13.5	9	144.5
Cartridge Brass				
Prompt Industrial	92.8	92.8	~100	--
Obsolete(6)	112.1	35.4	31	76.7
TOTAL	204.9	128.2	63	76.7
Automotive Radiators				
Prompt Industrial(7)	--	--	--	--
Obsolete	53.0	48.5	91	4.5
TOTAL	53.0	48.5	91	4.5
Railroad Car Boxes				
Prompt Industrial(8)	--	--	--	--
Obsolete	22.6	20.0	88	2.6
TOTAL	22.6	20.0	88	2.6
Other Brass, Cast and Wrought				
Prompt Industrial	310.0	310.0	~100	--
Obsolete	703.3	213.9(11)	30	489.4
TOTAL	1,013.3	523.9	52	489.4
Low Grade Scrap and Residues				
Prompt Industrial	37.2	37.2	~100	--
Obsolete(9)	--	--	--	--
TOTAL	37.2	37.2	~100	--
Other Scrap(10)				
Prompt Industrial	12.8	12.8	~100	--
Obsolete	6.1	6.1	--	--
TOTAL	18.9	18.9	100	--
Copper Alloying Additives(13)	96.9	--	00	96.9
TOTAL OBSOLETE	1,623.2	656.8	40	966.4
GRAND TOTAL	2,455.7	1,489.3	61	966.4

Source: Battelle Memorial Institute
EPA - SW - 40 D 1-72

The study indicates the percentages which could be reached under ideal conditions, namely if different incentives, i.e. price, cost, etc. would exist :

	1969 percent	Goal percent
Copper wire and tube	68	85
Magnet wire	9	20
Other brass	30	52

c) Technische - Wirtschaftliche Kenurdaten für die Herstellung und Verwendung von Kupfer (GST Gesellschaft für Systemtechnik mbH, Essen) - Germany

1974

. obsolete copper products	tons	251,000
. recovered copper products	tons	96,000
. copper products in solid waste disposal site	tons	122,000

Total recovery rate : $\frac{96,000}{251,000}$ = 38 %

	Collection rate	Segregation efficiency	Recovery rate	Discarded in solid waste system
	%	%	%	%
Building	22	70	15	78
Transport	83	65	54	17
Consumer goods	22	50	11	78
Industrial machinery and equipment	52	61.5	32	48
Electric industry	70	83.5	58.5	30

The BIPE study "Analyse technico-économique de la récupération et du recyclage des métaux nonferreux dans la Communauté Européenne - France 1977" states that on average the percentage of obsolete products considered as practically recoverable in France is of 52.5 %.

However, the actual recovery refers only to 70 % of the figure indicated, the recovery rate therefore results to be of 37 %.

Recycling rate for the main countries has been calculated using mobile averages of the 3rd order, a 20-year life cycle and assuming 10-15 % export of finished products containing copper, which cannot be derived from statistics on copper, to extra-EEC countries. The result has been : 43 % for France , 35 % for Italy, and 34 % for the UK. Germany has not been considered as data on apparent consumption of semifinished products and castings are not available for the period of time preceding 1968, and for this country a 38.5 % rate given in the GST study may be taken as a reference term.

The method used to calculate the recycling rate is somehow defective when assessing some parameters (see enclosures) and establishing the most probable average life of copper products. The latter depends on technical factors, but it is also influenced by independent causes such as the general state of the economy and the price of copper. If a higher average life cycle is assumed as a reference term, the recycling rate results higher.

On the other hand, the recycling rates stated in the literature as well as those calculated by us, which all fall between 35 % and 40 %, are significant. They indicate that, in many sectors of utilization, large quantities of copper scrap are not collected and subsequently processed for recovery.

5.3.3 Analysis of recycling of products by sector of utilization

5.3.3.1 Collection

Electrical sector

Part of the power and communication cables cannot recovered : namely submarine cables, or underground cables when these cannot be easily reached.

Considering the new methods of demolition of civil and industrial buildings, a notable part of power wire, communication wire, others insulated and appliances are disposed of in solid waste of demolition origin. Magnet wire in obsolete large apparatuses may be collected at a high rate, some of the small ones may be disposed of in household refuse.

Construction

With the current methods of building demolition, a large quantity of tubing, plumbers fittings and brassware are disposed of in solid waste and not collected. Collection is possible only for those fittings and tubing which are visible.

Collection of materials represents the most difficult aspect of recovery and requires a particular study on techniques to recover scrap during the demolition stage. Notwithstanding these difficulties, the ITE report foresees a 70 %-80 % recovery for the copper being currently used in the construction sector.

Transport

A high rate of collection can be achieved in this sector by setting appropriate rules on solid waste management.

General engineering

In this sector too, a high rate of collection should be achieved as it is the case for the transport sector.

Domestic goods and miscellaneous

It is expected that part of these uses is completely lost, especially chemical uses, or is disposed of in household refuse. Their recovery is therefore connected with that of municipal waste treatment.

5.3.3.2 Segregation

Electrical sector

Segregation rate

Recovery of copper contained in cable and wire does not represent a problem from a technological view-point. A high percentage of this copper is therefore recoverable by using the following techniques :

++ Insulated wire
   . mechanical automated means
   . cryogenic processing
   . burning with proper facilities

++ Plastic, rubber (lead sheathing) insulated cables
   . granulation and segregation in a dry fluidized bed separator or fluidized pinched sluice
   . sheathing removal by sweating or burning
   . incineration
   . research in progress on depolymerizing to make plastic biodegradable

From a technological viewpoint the recovery of the magnet wire presents some problems because of low copper content in scrap —5 % to 20-25 %— and the high cost for stripping the magnet wire from some apparatuses.

Effective recovery of high-grade copper and iron from copper-bearing ferrous scrap such as generators, starters, alternators, etc. by preferential melting in molten salts, slags, or other suitable media ; other techniques, hand and tool dismantling and cupric ammonium leaching and sulphur as a precipitant for the solubilized copper.

In the presence of volatile coatings such as varnish, plastic or rubber insulation a feasible approach is to smelt in a unit with suitable fume collection after burning and gas scrubbing equipment. The high iron results in production of large quantities of slag. Understanding and control of factors affecting slag composition are essential to prevent copper loss to the slag. One approach to decreasing slag losses is to run the hot slag from a secondary copper blast furnace through an electric arc furnace to fume off volatile impurities such as zinc and lead and collect copper and nickel in a black copper which can be further refined.

Transport

A high degree of segregation can be achieved in the case of road vehicles by means of preliminary removal of significant copper components, shredding and recovery of metal from shredder non-magnetic rejects. Alternatively for areas which do not discard enough cars to support a shredder plant = incineration, hand tool dismantling and baling.

General engineering

The problem concerning scrap processing is a rather more complex one. The following technologies may be indicated for the various types of apparatuses. These technologies can also be applied to similar articles of other sectors of utilization.

++ Electric components in panel board, complex multicomponent cables, electrical and electronic scrap :

competent concerns are successfully treating copper and brass scrap not containing precious metals by smelting of leaching. Selected high-grade electronic scrap, containing precious metals, is now being treated commercially by leaching or smelting and electrolysis. Research and development is being concentrated on lower grade scrap from complex assemblies containing much aluminium and iron not now economically treatable by usual methods.

Technology consists in incineration, sodium hydroxide leaching to eliminate aluminium, smelting to form precious metal rich copper bullion, copper electrolysis and precious metal recovery from sludges. For treatment of aluminium-bearing electronic scrap : incineration, aluminium sweating furnace —fused salts electrolysis— smelting of unmelted residue and anode residue to copper —precious metal bullion.

++ Separation of aluminium and copper from scrap diecasting

Cryogenic technique (indirect drilling to permit use of a liquid CO_2 - dry ice system).

5.3.4 Future outlook for secondary production in the EEC in 1985 and 1990

Secondary production from new scrap can be considered as proportional to the total consumption of copper by using the present parameters. On the contrary, there are doubts as to the assessment of secondary production from old scrap. It is however certain that an increase will occur if we consider the increases in copper consumption registered from the mid fifties onwards.

We have deemed advisable therefore to estimate a recycling rate for the period 1985-90 by assuming that it were possible for copper prices to stabilize around figures such as to stimulate the recovery of copper from old scrap ; that improvements in the technology of recovery had been achieved, and that positive results had been obtained from the introduction of incentives for the utilization of the EEC domestic resources, old scrap included.

To calculate the recyclable copper contained in obsolete products in 1985 and 1990, it was carried out the breakdown of the total consumption of copper expressed as semifinished products and castings, and of the final products by sectors of utilization and their components.

On the basis of the average life cycle of single components of sectors of utilization, the average life of every semifinished product and casting was established, and then the average life cycle of all the products, which resulted in 22 years.

The collection and segregation rates considered feasible for each sector of utilization were then calculated.

In general, collection rates over 85 % were assigned in the case of products which were identifiable, classifiable, and the collection of which can be organized : underground cables, overhead cables, transport, general engineering ; 50 % for renewed material ; 30 % for wiring cables and tubing in building ; 50 % for domestic goods.

As to the segregation rate, a 90 % yield was supposed for non-composite copper materials = cable, wiring, bars, flat products and tube ; 70 % for electric components and magnet wire ; 90 % for flat products and tubes in brass ; 65 % for composite brass materials ; 50 % for miscellaneous and domestic goods.

Copper recovered from scrap was then calculated using the indices assumed.

In general principles, by adopting an average life cycle of the order of 27 years and assuming 40 %-50 % recycling rates, the copper recycled from old scrap in 1985 and 1990 should be comprised between 800,000 tons/year and 1,100,000 tons/year, remarkably higher than the levels registered in the last decade.

In addition, it was considered the recovery of copper from 40 % of the total amount of domestic solid waste through incineration and metal reclaiming from incinerator residues, or methods of maximum recovery of all values (paper, plastics, and organic for fuel or recycling ; and ferrous metals, copper-zinc metals, aluminium-base metals, flint glass, and amber and green glass for recycling).

The copper recoverable is considered to amount to 25,000 tons, which is not a significant quantity .

Although this study is primarily concerned with the recovery of obsolete copper products, mention should be made of areas where copper is potentially recoverable from sources which are not end-use products. These areas are : flue dust from primary copper smelters and refineries ; electroplating and electroforming process and the manufacture of printed circuits ; chemical sludges ; pyrometallurgical residues.

An estimate of the total amounts would require a detailed specific survey ; it may be guessed in 5-6,000 ton/year copper contained.

BREAKDOWN OF COPPER CONSUMPTION BY FINAL SECTORS OF UTILIZATION

SECTORS (%)	UTILIZATION	SEMIS	FINISHED
ELECTRICAL		47.0	52.3
(21)	Power wire and cable	9.9	11.0
(7.5)	Bare wire	3.5	3.9
(18)	Magnet wire	8.5	9.4
(28.5)	Other insulated wire	13.5	14.9
(20)	Communication wire & cable	9.4	10.5
(5)	Appliances	2.4	2.6
BUILDING		16.7	14.1
(40)	Tubing	6.7	5.6
(34)	Plumbers fitting & brass-ware	5.7	4.8
(12.5)	Water heating system	2.1	1.8
(9)	Builders hardware	1.5	1.3
(4.5)	Architectural use	0.7	0.6
TRANSPORT		11.0	10.7
(1.5)	Aircraft	0.2	0.2
(14)	Railways	1.5	1.5
(24.5)	Ship	2.7	2.6
(60)	Road vehicles	6.6.	6.4
GENERAL ENGIN.		16.7	15.7
(29)	Heat exchangers	4.9	4.6
(35)	Valves	5.8	5.4
(14)	Pumps	2.3	2.2
(9.5)	Refrigeration, air condit.	1.6	1.5
(12.5)	Bearings	2.1	2.0
DOMESTIC GOODS & MISCELLANEOUS		8.6	7.2

LIFE CYCLE; COLLECTION, SEGREGATION AND RECYCLING RATE OF SEMIS & CASTINGS

BY SECTOR OF UTILIZATION

SEMIS %	UTILIZATION	%	LIFE CYCLE (years)	COLLEC. RATE %	SEGREG. RATE %	RECYCLING RATE %
COPPER		66.3	26	61	82	50
WIRE		46.2	26	66	81	53
(21.4)	Cable	9.9	40	85	90	76
(20.0)	Wiring - building	9.2	30	30	90	27
(4.7)	" - transport	2.2	13	85	60	51
(3.6)	" - consumers	1.7	10	85	60	51
(7.6)	Transformers	3.5	20	80	70	56
(4.6)	Motors & generators-Transp.	2.1	13	80	70	56
(6.5)	" " -Consum.	3.0	10	80	70	56
(9.2)	Switch gear	4.3	10	50	70	35
(15.2)	Telecommunication cable	7.0	30	85	90	76
(3.0)	" equipment	1.4	30	50	70	35
(4.2)	Wiring accessories	1.9	30	30	90	27
BARS		2.6	28	78	86	67
(87.8)	Electrical - Transmission	2.3	30	80	90	72
(3.8)	General engineering	0.1	18	80	70	56
(8.4)	Domestic goods	0.2	10	50	50	25
SHEETS		5.1	18	62	80	50
(23.0)	Electrical - Switch gear	1.2	10	50	80	40
(36.0)	Building	1.8	30	60	90	54
(18.0)	Transport	0.9	10	80	90	72
(10.0)	General engineering	0.5	18	80	70	56
(13.0)	Domestic goods	0.7	10	50	50	25
TUBES		12.4	27	40	87	35
(8.0)	Electrical - Equipment	1.0	20	80	90	72
(78.5)	Building	9.8	30	30	90	27
(1.8)	Transport	0.2	9	80	90	72
(8.9)	General engineering	1.1	18	80	70	56
(2.8)	Domestic goods	0.3	10	50	50	25

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COPPER ALLOYS		24.7	15	72	69	49
WIRE		1.9	13	73	61	45
(21.3)	Electrical - Equipment	0.4	10	80	65	52
(12.3)	Transport	0.2	9	80	65	52
(13.0)	General engineering	0.8	18	80	65	52
(23.4)	Domestic goods	0.5	10	50	50	25
BARS		11.2	15	74	66	49
(9.8)	Electrical - Equipment	1.1	10	80	65	52
(13.0)	Building	1.5	10	50	70	35
(8.2)	Transport	0.9	9	80	90	72
(61.0)	General engineering	6.8	18	80	65	52
(8.0)	Domestic goods	0.9	10	50	50	25
SHEETS		6.4	13	74	70	52
(34.3)	Electrical-Equipment	2.2	10	80	70	56
(5.8)	Building	0.4	30	60	90	54
(18.0)	Transport	1.1	9	80	90	72
(26.5)	General engineering	1.7	18	80	65	52
(15.1)	Domestic goods	1.0	10	50	50	25
TUBES		5.2	18	64	75	48
(7.6)	Electrical - Equipment	0.4	10	80	65	52
(28.0)	Building	1.5	30	30	90	27
(17.7)	Transport	0.9	9	80	90	72
(38.4)	General engineering	2.0	18	80	65	52
(8.3)	Domestic goods	0.4	10	50	50	25
CASTINGS		9.0	20	66	63	42
(2.2)	Electrical Equipment	0.2	10	80	65	52
(20.2)	Building	1.8	30	30	65	20
(25.6)	Transport	2.3	20	80	65	52
(40.0)	General engineering	3.6	18	80	65	52
(12.0)	Domestic goods	1.1	10	50	50	25
TOTAL COPPER		100.0	22	64	77	49

5.3.5 Conclusions

The results of the various surveys conducted on the different markets and on the basis of the documentation available indicate that the recycling rate of old scrap copper is 40 %, remarkably lower than the 75 % rate deemed theoretically possible.

An increase in the recycling rate represents an important objective to be achieved for the supply of raw materials to the country. Besides, no negative aspects resulted under the technical side of recycling.

The low recycling rate is due to the fact that scrap collection is oriented towards those obsolete products considered of an economic value within the present structure of the recycling industry and technologies applied.

To improve the recycling rate, it is necessary a global action concerted by government bodies who are responsible for the environment protection policies, preservation of resources and raw materials supply policies, associations of economic operators in the field of recovery and recycling, experts in the recycling industry.

In our opinion, the means for achieving an increase in the recycling rate of old scrap on a medium to long term are discussed hereunder :

- To improve the present methods of collecting statistical data by introducing direct data collections on new scrap and finished copper products destined to exports so that to have more information available for carrying out more accurate assessments on the quantity of finished products entering the market and of the old recycled scrap.
- To ameliorate the image of the recyclina industry by establishing certain rules and to intensify the process, already being carried out, of concentrating the companies at a large-merchant level in order to increase their financial capacity and specialization degree, acquiring at the same time an industrial size.

  This would result in increased opportunities for recycling and improved dealing capacities with the aim to set for the recycled materials the same prices quoted for the primary materials of equal quality.

- To revise specifications and standards so that the recycled materials are not considered unfavourably in comparison to the primary materials.
- To introduce economic incentives for the realization of recycling plants for materials considered as marginal and submarginal, and methods of depletion allowances on recycling and to modify tax structure accordingly.
- To study the possibility of applying differentiated tariffs for the transport of recycling materials.
- To provide a central source of technical information and advice for the recycling industry concerning technological problems, purchasing, installing, using and maintaining processing plants, rationalization of processes for manufacturing products so that the finished products are more easily recyclable.
- To promote classification of solid waste.
- To provide some incentives for people discarding copper-base products to have them recovered.

Specific problems consist in the necessity of intensifying collection through a different technical and economical set up in the construction sector for heating and sanitary systems ; to introduce appropriate regulations in the sector of cable and wire for telecommunications and electric energy distribution, at present slightly recovered ; to make improved arrangements for collection in the general engineering sector.

Recovery should be extended to areas of solid waste system : of municipal, industrial, commercial, and household origin, characterized by non-economic incentives.

To identify those sectors which are characterized by very low collection rates or which require a finalization of the segregation metallurgical process, the collection rates and the segregation rates of every single copper product by the various uses have been analyzed, the documentation on the present segregation processes on an industrial scale has also been examined as well as that on the applied metallurgy for the finalization of processes concerning the recovery of copper, zinc, and other minor metals contained in complex scrap.

The analysis emphasizes the sectors of application for which R O D actions would be necessary to improve collection and segregation rates.

Collection

The sectors of copper application with the lowest collection rates (less than 30 %) due to new demolition methods used are :

- a notable part of power wire, communication wire, others insulated and appliances ;
- copper tubing and partly copper sheet ;
- plumbers fittings and brassware copper alloys tubing.

The above three sectors account for 15 % to 20 % ; 10 % to 12 % ; 5 % to 7 % respectively of total consumption.

Processing

Electrical sector

. Magnet wire : processing presents some problems because of low copper content in scrap –5 % to 20-25 %– and the high cost for stripping the magnet wire from some apparatuses. Techniques used consist in preferential melting in molten salts, slags, and suitable media ; hand and tool dismantling and cupric ammonium leaching and sulphur as a precipitant for the solubilized copper ; smelting and special approaches to decreasing slag losses.

General engineering

. Electric components in panel board, complex multicomponent cables, electrical and electronic scrap : Research and Development is being interested in the problem as a whole and is being concentrated on lower grade scrap from complex assemblies containing much aluminium aluminium and iron.

. Separation of aluminium and copper from scrap diecasting. Cryogenic technique.

. Scrap formed by : brass, other copper-alloys, cast and wrought, not included in easily recyclable finished products.

Rate of recovery is estimated at 50 %-60 % and this sector of utilization accounts for roughly 45 % of copper-alloy semis and castings consumption. Research and Development should cover the entire problem.

Solid waste system

It would be advisable to carry out a systematic research for all the EEC countries to establish the breakdown of copper content in waste products and to set the opportunities for the recovery of copper (and zinc) and other metals.

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Definition of copper scrap

In the Brussels Nomenclature for the classification of goods in Customs tariffs, 'scrap and waste material' is defined as scrap and waste metal (including ash and residues containing metal or metallic compounds) fit only for the recovery of metal or for use in the manufacture of chemicals.

Three classes of copper-bearing scrap and waste are recycled by industry. These are 'new' or process scrap from current fabrication and manufacturing operations, 'old' scrap from discarded or obsolete capital and consumer goods, and ashes and residues which arise largely from pyrometallurgical, foundry and metal finishing operations.

'New' or process scrap

'New' scrap consist mainly of the metallic residue, i.e.: trimmings, off-cuts, grindings and turnings (swarfs) which remain after metal-working operations. New scrap arise at factories which fabricate semifinished products such as sheets, strips, rods, tubes, or wires during production operations, and at the fabricating stage of finished products. The new scrap which are re-used within the plant itself at refineries, semimanufacturers' plants and possibily fabricating plants are to be considered 'circulating scrap' or 'run around scrap'. They are not identified statistically and are not considered in this study. This study concerns the new scrap arising at the fabricating stage which are returned directly to the mill supplying the semimanufactured products from which they were produced or sold to scrap metal merchants and a minor part of new scrap produced at semimanufacturers' plants.

'Old' scrap

'Old' scrap consist of materials which are of no further use for the purpose for which they were originally made. They include obsolescent, scrapped or worn-out machinery, manufacturing plants, old ships, aircraft, vehicles, cabling, spent ammunition, building demolition scrap as well as domestic consumer goods such as refrigerators, vacuum cleaners, etc.

Metallurgical ashes and residues

The third category of copper scrap is, in the direct sense of the term, not scrap metal as such, but residual products from pyrometallurgical processes, including ashes and residues from metallurgical processes other than copper, and metal-finishing operations. These include slags, skimmings, dross anode slimes and sludges and fume from refining, alloy ingot manufacture, foundries and wastes from metal-finishing operations. When metal values are present in high concentrations these residues are usually recycled within the plant or sold for further processing. If gross contamination by undesirable elements has occurred or if the metal content is so low that there ceases to be an economic attraction in reclamation the material will be disposed of by landfill or other means.

APPENDIX II

U.K. - STANDARD CLASSIFICATIONS FOR NON-FERROUS SCRAP METALS

ISSUED BY: NATIONAL ASSOCIATION OF NON-FERROUS SCRAP METAL MERCHANTS

Camel — **Copper, Braziery**

Shall consist of Soldered Tinned and Brazed Copper and may contain Copper Vat Wire. It may contain up to 5% of Brass or Gunmetal attachments. It must be free from Electrotype Shells, Back Boilers and Iron and commercially free from any other foreign matter.

Capon — **Copper Cuttings**

Shall consist of clean new Untinned Copper Sheet cuttings of various gauges. It must be free from soldered and plated material, Iron and any other foreign matter.

Carps — **Copper Hair Wire, Tinned**

Shall consist of clean Unburnt Tinned Copper Hair Wire free from Tin/Lead Alloy Coated Copper Wire. It must be free from Iron and any other foreign matter.

Chick — **Copper Hair Wire, Untinned**

Shall consist of clean Unburnt Copper Hair Wire, free from Enamelled Wire. It must be free from Iron and any other foreign matter.

Chimp — **Copper, Heavy**

Shall consist of crucible size clean untinned Copper not less than a thickness of 1/16 in. such as cut-up pipe, tubing and sheet. It must be free from brazed, soldered and plated material and Iron and commercially free from any other foreign matter.

Chows — **Copper, Light**

Shall consist of collected clean old Copper Sheet and Tube Scrap, free from tinned, brazed, soldered and plated material, perished and brittle Copper and Iron and commercially free from any other foreign matter.

Clams — **Copper Wire, Bright Tinned**

Shall consist of clean bright Unburnt Tinned H.C. Copper Wire free from Hair Wire. It must be free from Iron and any other foreign matter.

Cobra — **Copper Wire, Bright Untinned**

Shall consist of clean bright Unburnt H.C. Copper Wire not thinner than 22 gauge. It must be free from Iron and any other foreign matter.

Colts — **Copper Wire No. 1**

Shall consist of clean untinned H.C. Copper Wire not thinner than 22 gauge, free from Hard Wire, Brittle Burnt Wire and Iron and commercially free from any other foreign matter.

<u>Cord</u> <u>Copper Wire No. 2</u>

Shall consist of Copper Wire such as Burnt Tinned Wire and may contain a percentage of Unburnt Tinned and Soldered Wire. It must be free from Hair Wire, Vat Wire, Burnt Wire which is brittle, Iron, and commercially free from any other foreign matter.

<u>Otter</u> Radiator Blocks Unsweated Copper.

STANDARD CLASSIFICATION OF GERMAN SCRAP METAL AND METALLIC RESIDUES

Kader — Electrolytic copper wire, min. thickness 1 mm, solderless and free from lead, not tinned, free from burnt or brittle wire, free of foreign inclusions.

Kanal — Electrolytic copper wire, min. thickness 0.15 mm solderless and free from lead, not tinned, free of burnt, brittle wires. Free of foreign inclusions.

Karat — Mixed copper wire, free from capillary wires and burnt brittle wires. May be tinned to 15%, with mixed tinning, and may contain wires with joins. Free of foreign inclusions.

Karin — Tinned copper wire, also with mixed tinning, each thickness, without solder- or lead-adhesions. May contain to 10% capillary wires. Free of foreign inclusions.

Kaste — Copper wire from dismantled telephone cables without iron core. Free of foreign inclusions.

Kelle — Parts of split-up combustion chambers with bolts knocked out. May contain up to 10% loose pins and bolts. Permissible amount of foreign inclusions: 0.5% of the weight of material to be delivered, consisting of scale and dirt which can be scaled off. If the amount is over 0.5%, the corresponding loss in weight and the cost of cleaning is to be paid for by the seller. Delivery to be in crucible form.

Kerzo — New waste from copper plate and copper strip, solderless. May contain up to 10% perforations. Free of foreign inclusions.

Keule — Tube-, plate- and other pieces of copper. Min thickness 1 mm, soft solderless, without tinning or nickel plating. Permissible amount of foreign inclusions: max. 1% dirt and scale. If the amount is over 1% the corresponding loss in weight and the cost of cleaning is to be paid for by the seller. Delivery to be in crucible form.

Klara — Tube-, plate- and other pieces of copper. Min. 1 mm thick. May be tinned or untinned, nickel-plated or not nickel-plated and may contain soldered material. Permissible amount of foreign inclusions: 2%. Delivery to be in crucible form.

Klima — Pieces of tube and plate, mixed wire including capillary wire, copper equipment of all kinds, free of copper from blocks, condensers and electroplating. Permissible amount of foreign inclusions: 3%.

Kobra — Borings, drillings and millings, free of filings, grindings and shavings. Permissible amount of foreign inclusions: mechanical impurities due to humidity and iron max. 3% due to iron max. 2%.

Konto — Pure, unmelted copper condenser without foreign inclusions. Impurities max. 3% incl. 1% iron which is difficult to remove.

BIR CLASSIFICATION FOR COPPER AND COPPER ALLOYS SCRAP

1. CUIVRE

EURO I/1 «cabin»

Fil de cuivre brillant

Fil de cuivre propre, non brûlé, brillant, non allié, d'un diamètre minimum de 1 mm, barres de connection et lames de collecteurs en cuivre.

Exempt de matériel plaqué et de matières étrangères.

Enfournable, sauf stipulation contraire

EURO I/2 «cabro»

Tombants de planches neuves de cuivre

Tombants de planches neuves de cuivre non-allié d'une épaisseur minimum de 0,2 mm.

Exempts de matériel plaqué et de matières étrangères.

Tolérance: 10 % menus homogènes

Enfournables, sauf stipulation contraire.

EURO I/3 «cadel»

Fil de cuivre non allié

Fil de cuivre non allié d'un diamètre minimum de 1 mm.

Exempt de matériel plaqué et autres matières étrangères, ainsi que de fil cassant, fortement oxydé par excès de brûlage ou provenant de brûlage de cables sous plastique.

EURO I/4 «calyx»

Déchets de fil de cuivre non allié, diamètre minimum de 0,15 mm

Fil de cuivre non allié, d'un diamètre minimum de 0,15 mm.

Exempt de matériel plaqué et autres matières étrangères, ainsi que de fil cassant fortement oxydé par excès de brûlage ou provenant de brûlage de câbles sous plastique.

EURO I/5 «caper»

Déchets de fils de cuivre mêlés

Fil de cuivre non allié avec un maximum de 15 % de fil étamé ou plaqué.

Exempt de fil cheveu ou fil cassant fortement oxydé par excès de brûlageou provenant de brûlage de câbles sous plastique.

EURO I/6 «cerro»

Cuivre lourd

Déchets de cuivre d'une épaisseur minimum de 1 mm, non brasé, non plaqué.

Tolérance: 1 % maximum de matières étrangères non métalliques.

Marchandises prêtes au creuset.

EURO I/7 «cider»

Déchets de cuivre mêlé

Déchets de cuivre mêlé plaqué et ou non plaqué d'une épaisseur minimum de 0,5 mm.

Fil diamètre minimum 0,15 mm.

Exempts de fil cassant of fortement oxydé par excès de brûlage ou provenant de brûlage de câbles sous plastique.

Tolérance: 1 % max. de matières étrangères.

Marchandises enfournables sauf stipulations contraires.

EURO I/8 « colon »

Cuivre léger

Déchets de tubes et de tôles, fils de cuivre mêlé y compris fil cheveu, ustensiles de cuivre de tous genres avec une teneur minimum de 88 % de cuivre.

Exempts de cuivre-cliché, de radiateurs et de galvanos.

Tolérance: 3 % max. d'autres matières étrangères.

EURO I/9 « coral »

Autres déchets de cuivre

A vendre d'après la teneur en cuivre, doit contenir au minimum: 85 % de cuivre. Au cas où la teneur en cuivre se situe entre 80 et 85 %, l'acheteur ne peut refuser la livraison, mais il y a lieu de négocier une réduction de prix. Les rabais éventuels sont à convenir au préalable.

Marchandises enfournables.

EURO I/10 « cycle »

Tournures de cuivre

Tournures de cuivre non allié de toutes origines, mais exemptes de tournures de polissage, limailles et meulures.

Tolérance: au maximum: 3 % de fer, graisse et humidité.

Toutes les qualités non spécifiées ci-dessus, comme par exemple: fils de trolley et de téléphone, plaques de foyer avec/ou sans entretoise (?), ainsi que Cu-Te — Cu-Al, Cu-Mn, Cu-Be, exigent des accords de vente particuliers.

II. BRONZE

EURO II/1 «racer»	**Mitrailles de bronze commercial**
	Pièces titrant au minimum 85 % de cuivre et étain avec un minimum de 4 % d'étain et un maximum de 6 % de plomb contenus, le tout en moyenne du lot.
	Exemptes de tous autres alliages.
	Tolérance 1 % maximum de fer libre.
	Marchandises prêtes au creuset.
EURO II/2 «radar»	**Mitrailles de bronze de robinetterie**
	Pièces titrant au minimum 78 % de cuivre et étain avec un minimum de 3 % d'étain.
	Exemptes de bronze au plomb, d'alliages à aluminium et au manganèse.
	Tolérance 1 % maximum de fer libre.
	Marchandises prêtes au creuset.
EURO II/3 «rally»	**Tournures de bronze ordinaire**
	Tournures et fraisures de bronze titrant au minimum 70 % de cuivre et 3 % d'étain.
	Exemptes de meulures et de limailles.
	Doivent être vendues sur échantillon et/ou analyse.
EURO II/4 «rebel»	**Toiles de bronze pur**
	Toiles de bronze à l'étain propres et en rouleaux, composés de fils de diamètres différents.
	Tolérance 2 % de matières non métalliques. Dans le cas où le poids de matières non métalliques est supérieur à 2 %, il est effectué d'une déduction correspondante au poids.
EURO II/5 «river»	**Toiles de bronze mixte**
	Toiles propres en rouleaux et/ou en paquets, composées de fils de différents diamètres, dont la trame doit être en fil tombac (Similor), titrant au minimum 88 % de cuivre et étain, avec un minimum de 3 % d'étain.
	Tolérance 1 % de plomb, 2 % de matières non métalliques. Dans le cas où le poids de matières non métalliques est supérieur à 2 %, il est effectué d'une déduction correspondante du poids.

P. B. Tous les autres déchets, débris ou tournures de bronze, tels que bronze phosphoreux, bronze au plomb, bronze au silicium etc., doivent être vendus sur échantillon ou analyse.

III. LAITON

EURO III/1 **«babel»**

Tombants de planches neuves, en laiton sans plomb

Chutes de fabrication de tôles et de bandes en laiton sans plomb, contenant 10 % au maximum de débouchures homogènes. Titre Cu: 63 % au minimum.

Exemptes de matériel plaqué et de toutes autres matières étrangères.

De dimensions enfournables.

EURO III/2 **«basis»**

Tombants de planches neuves en laiton au plomb

Chutes de fabrication de planches et de bandes laiton au plomb, contenant 10 % au maximum de débouchures homogènes. Teneur en cuivre et en plomb à spécifier.

Exemptes de matériel plaqué et de toutes autres matières étrangères.

De dimensions enfournables.

EURO III/3 **«bacon»**

Chutes de barres en laiton

Chutes de barres en laiton. Teneur en cuivre: 57 % au minimum. Teneur maximum en Sn: 0,3 %, en Fe dans l'alliage: 0,2 %.

Exemptes de chutes de barres en laiton spécial et de tout matériel plaqué.

De dimensions enfournables.

EURO III/4 **«bacus»**

Tournures de barres de laiton

Tournures et fraisures de barres en laiton.

Exempts de meulures, de limailles, ainsi que de tournures de barres en laiton spécial. Teneur en cuivre: 57 % au minimum: en Sn: 0,3 % au maximum, en Al: 0,1 % et en Fe contenue dans l'alliage: 0,2 % au maximum.

Tolérance: 4 % au maximum d'humidité, de graisse et de fer libre, mais seulement 1 % au maximum de fer magnétique. Tout excédent d'humidité, de graisse et de fer libre est à déduire du poids.

EURO III/5 **«bamel»**

Tournures de laiton

Tournures, fraisures et filasses de types divers de laiton. Il importe de prêter attention au post scriptum concernant les alliages spéciaux.

Exemptes de meulures.

Tolérance: 4 % au maximum de fer, de graisse et d'humidité, mais seulement 1 % au maximum de fer libre. Tout excédent d'humidité de graisse ou de fer libre est à déduire du poids.

Il est recommandé de vendre sur échantillon ou analyse.

EURO III/0 **«baron»**

Douilles d'obus désamorcées en laiton

Douilles en laiton, sans amorces, propres, ni explosées ni grillées. Alliage: 70/30. Tout autre alliage doit être spécifié obligatoirement.

Exemptes d'amorces et de matières étrangères, surtout explosives. Diamètre: 37 m/m au minimum.

EURO III/7 Douilles d'obus en laiton non désamorcées
« basin » Douilles avec amorces grillées, pouvant avoir été grillées. Alliage: 70/30. Tout autre alliage doit être spécifié obligatoirement.
Exemptes de matières étrangères, surtout explosives. Diamètre: 37 m/m au minimum.

EURO III/8 Etuis de cartouches propres, tirés, en laiton
« basty » Etuis de cartouches propres, tirés, de différents types non grillés, à l'exclusion d'étuis en alliage d'aluminium.
Exempts de matières étrangères, surtout explosives. Diamètre minimum: 6 m/m.

EURO III/9 Etuis de cartouches, grillés, en laiton
« belly » Etuis de cartouches en laiton, propres, grillés, de différents types. Etuis de cartouches éclatés sujets à accord spécial.
Exempts de balles, de fer et toutes autres impuretés.

EURO III/10 Tubes porte amorces et capsules d'amorces en laiton
« berto » Tubes porte amorces et capsules d'amorces en laiton, grillés et/ou tirés.
Exempts de fer et autres matières étrangères.

EURO III/11 Déchets de tubes en laiton
« bezel » Déchets et débris de tubes en laiton, exempts d'aluminium et d'étain dans l'alliage.
Exempts de matériel plaqué.
Tolérance: 2 % au maximum de tartre.

EURO III/12 Déchets de tubes de condenseurs en laiton à l'étain ou à l'aluminium
« bingo » Déchets de tubes de condenseurs en laiton à l'étain ou à l'aluminium. Alliage à spécifier.
Exempts de matières plaquées.
Tolérance: 2 % au maximum de tartre.

EURO III/13 Déchets de tubes de condenseurs en laiton mêlés
« bison » Déchets de tubes de condenseurs en laiton, mêlés.
Exempts de matériel, plaqué.
Tolérance: 2 % au maximum de tartre.

EURO III/14 Déchets de vieux laiton laminé
« blado » Déchets de tubes et vieilles chutes de laminés en laiton, conformes à la description EURO III/11 non corrodés.
Exempts de tubes de condenseurs.
Matériel enfournable.

EURO III/15 **Déchets laiton lourd**
« bogie » Déchets divers en laiton lourd, contenant 15 % au maximum de matériel plaqué et/ou dessoudé.
Exempts d'alliage à l'aluminium et/ou au manganèse, chargés de plomb et de soudure de radiateurs ou parties de radiateurs, de fer et autres matières étrangères. Marchandises prêtes au creuset.

EURO III/16 **Déchets laiton mêlés**
« bravo » Déchets de laiton mêlés, contenant environ 40 % de laiton lourd.
Exempts de: Alliages à l'aluminium et/ou au manganèse, de parties de radiateurs ou de boites à eau de radiateurs, chargés en plomb ou en soudure.
Tolérance: 1 % au maximum de fer libre.
Matériel enfournable.

EURO III/17 **Déchets en laiton léger**
« bulbo » Déchets de laiton léger, plaqué ou non, exempts de matériel chargé en plomb et de joints de culasses.
Exempts d'alliages à l'aluminium et/ou au mangenèse.
Tolérance: 3 % au maximum de fer.
Matériel enfournable.

USA CLASSIFICATION

TYPES OF UNALLOYED COPPER SCRAP

NO. 1 COPPER WIRE

Shall consist of clean, untinned, uncoated, unalloyed copper wire and cable, not smaller than No. 16 B and S wire gauge, free of burnt wire which is brittle. Hydraulically briquetted copper subject to agreement.

NO. 2 COPPER WIRE

Shall consist of miscellaneous, unalloyed copper wire having a nominal 96 percent copper content (minimum 94 percent) as determined by electrolytic assay. Should be free of the following: excessively leaded, tinned, soldered copper wire; brass and bronze wire; excessive oil content, iron, and nonmetallics; copper wire from burning, containing insulation; hair wire; burnt wire which is brittle; and should be reasonably free of ash. Hydraulically briquetted copper wire subject to agreement.

NO. 1 HEAVY COPPER

Shall consist of clean unalloyed, uncoated copper clippings, punchings, bus bars, commutator segments, and wire not less than 1/16 of an inch thick, free of burnt wire which is brittle; but may include clean copper tubing. Hydraulically briquetted copper subject to agreement.

NO. 2 COPPER

Shall consist of miscellaneous, unalloyed copper scrap having a nominal 96 percent copper content (minimum 91 percent) as determined by electrolytic assay. Should be free of the following: excessively leaded, tinned, soldered copper scrap; brasses and bronzes; excessive oil content, iron and nonmetallics; copper tubing with other than copper connections or with sediment; copper wire from burning containing insulation; hair wire; burnt wire which is brittle; and should be reasonably free of ash. Hydraulically briquetted copper subject to agreement.

LIGHT COPPER

Shall consist of miscellaneous, unalloyed copper scrap having a nominal 92 percent copper content (minimum 88 percent) as determined by electrolytic assay and shall consist of sheet copper, gutters, downspouts, kettles, boilers, and similar scrap. Should be free of the following: burnt hair wire; copper clad; plating racks; grindings; copper wire from burning, containing insulation; radiators; fire extinguishers; refrigerator units; electrotype shells; screening; excessively leaded, tinned, soldered scrap; brasses and bronzes; excessive oil, iron and nonmetallics; and should be reasonably free of ash. Hydraulically briquetted copper subject to agreement. Any items excluded in this grade are also excluded in the higher grades above.

Source: National Association of Secondary Material Industries, Circular NF-66.

TYPES OF COPPER BASE SCRAP

COMPOSITION OR RED BRASS

Shall consist of red brass scrap, valves, machinery bearings, and other machinery parts, including miscellaneous castings made of copper, tin, zinc, and/or lead. Should be free of semi-red brass castings (78 percent to 81 percent copper); railroad car boxes and other similar high-lead alloys; cocks and faucets; gates; pot pieces; ingots and burned brass; aluminum and manganese bronzes; iron and nonmetallics. No piece to measure more than 12" over any one part or weigh over 100 pounds.

RED BRASS COMPOSITION TURNINGS

Shall consist of turnings from red brass composition material and should be sold subject to sample or analysis.

GENUINE BABBITT-LINED BRASS BUSHINGS

Shall consist of red brass bushings and bearings from automobiles and other machinery, shall contain not less than 12 percent high tin base babbitt, and shall be free of iron-backed bearings.

HIGH GRADE-LOW LEAD BRONZE SOLIDS

It is recommended these materials be sold by analysis.

BRONZE PAPER MILL WIRE CLOTH

Shall consist of clean genuine Fourdrinier wire cloth and screen having a minimum copper content of 87 percent, minimum tin content of 3 percent, and a maximum lead content of 1 percent free of stainless steel and Monel metal stranding.

HIGH LEAD BRONZE SOLIDS AND BORINGS

It is recommended that these materials be sold on sample or analysis.

MACHINERY OR HARD BRASS SOLIDS

Shall have a copper content of not less than 75 percent, a tin content of not less than 6 percent, and a lead content of not less than 6 percent--nor more than 11 percent, and total impurities, exclusive of zinc, antimony, and nickel of not more than 0.75 percent; the antimony content not to exceed 0.50 percent. Shall be free of lined and unlined standard red car boxes.

UNLINED STANDARD RED CAR BOXES (CLEAN JOURNALS)

Shall consist of standard unlined and/or sweated railroad boxes and unlined and/or sweated car journal bearings, free of yellow boxes and iron-backed boxes.

LINED STANDARD RED CAR BOXES (LINED JOURNALS)

Shall consist of standard babbitt-lined railroad boxes and/or babbitt-lined car journal bearings, free of yellow boxes and iron-backed boxes.

COCKS AND FAUCETS

Shall consist of mixed clean red and yellow brass, including chrome or nickel-plated, free of gas cocks, beer faucets, and aluminum and zinc base die cast material, and to contain a minimum of 35 percent semi-red.

MIXED BRASS SCREENS

To consist of clean mixed copper, brass and bronze screens, and to be free of excessively dirty and painted material.

YELLOW BRASS SCRAP

Shall consist of brass castings, rolled brass, rod brass, tubing and miscellaneous yellow brasses, including plated brass. Must be free of manganese-bronze, aluminum-bronze, unsweated radiators or radiator parts, iron, excessively dirty and corroded materials.

YELLOW BRASS CASTINGS

Shall consist of yellow brass castings in crucible shape, no piece to measure more than 12 inches over any one part; and shall be free of brass forgings, silicon bronze, aluminum bronze and manganese bronze, and not to contain more than 15 percent nickel plated material.

OLD ROLLED BRASS

Shall consist of old pieces of yellow sheet brass and yellow light tubing brass, free from solder, tinned and nickel plated material, iron, paint and corrosion, rod brass and condenser tubes.

NEW BRASS CLIPPINGS

Shall consist of the cuttings of new unleaded yellow brass sheet, to be clean and free from foreign substances and not to contain more than 10 percent of clean brass punchings under 1/4". To be free of Muntz metal and naval brass.

BRASS SHELL CASES WITHOUT PRIMERS
Shall consist of clean fired 70/30 brass shell cases free of primers and any other foreign material.

BRASS SHELL CASES WITH PRIMERS
Shall consist of clean fired 70/30 brass shell cases containing the brass primers and which contain no other foreign material.

BRASS SMALL ARMS AND RIFLE SHELLS, CLEAN FIRED
Shall consist of clean fired 70/30 brass shells free of bullets, iron and any other foreign material.

BRASS SMALL ARMS AND RIFLE SHELLS, CLEAN MUFFLED (POPPED)
Shall consist of clean muffled (popped) 70/30 brass shells free of bullets, iron and any other foreign material.

YELLOW BRASS PRIMER
Shall consist of clean yellow brass primers, burnt or unburnt. Free of iron, excessive dirt, corrosion and any other foreign material.

BRASS PIPE
Shall consist of brass pipe free of plated and soldered materials or pipes with cast brass connections. To be sound, clean pipes free of sediment and condenser tubes.

YELLOW BRASS ROD TURNINGS
Shall consist of strictly rod turnings, free of aluminum; manganese, composition, Tobin and Muntz metal turnings; not to contain over 3 percent free iron, oil or other moisture; to be free of grindings and babbitts; to contain not more than 0.30 percent of tin and not more than 0.15 percent of alloyed iron.

NEW YELLOW BRASS ROD ENDS
Shall consist of new, clean rod ends from free turning brass rods or forging rods, not to contain more than 0.30 percent tin and not more than 0.15 percent alloyed iron. To be free of Muntz metal and naval brass or any other alloys. To be in pieces not larger than 12" and free of foreign matter.

YELLOW BRASS TURNINGS
Shall consist of yellow brass turnings, free of aluminum, manganese and composition turnings; not to contain over 3 percent of free iron, oil or other moisture; to be free of grindings and babbitts. To avoid dispute, to be sold subject to sample or analysis.

MIXED UNSWEATED AUTO RADIATORS
Shall consist of mixed automobile radiators, to be free of aluminum radiators, and iron finned radiators. All radiators to be subject to deduction of actual iron. The tonnage specification should cover the gross weight of the radiators, unless otherwise specified.

ADMIRALTY BRASS CONDENSER TUBES
Shall consist of clean sound Admiralty condenser tubing which may be plated or unplated, free of nickel alloy, aluminum alloy, and corroded material.

ALUMINUM BRASS CONDENSER TUBES
Shall consist of clean sound condenser tubing which may be plated or unplated, free of nickel alloy, and corroded material.

MUNTZ METAL TUBES
Shall consist of clean sound Muntz metal tubing which may be plated or unplated, free of nickel alloy, aluminum alloy, and corroded material.

PLATED ROLLED BRASS
Shall consist of plated brass sheet, pipe, tubing, and reflectors, free of soldered tinned, corroded, and aluminum painted material, Muntz metal and Admiralty tubing, and material with cast brass connections.

MANGANESE BRONZE SOLIDS
Shall have a copper content of not less than 55 percent, a lead content of not more than 1 percent, and shall be free of Aluminum bronze and Silicon bronze.

MACHINERY OR HARD BRASS BORINGS
Shall have a copper content of not less than 75 percent, a tin content of not less than 6 percent, and a lead content of not less than 6 percent--nor more than 11 percent and total impurities, exclusive of zinc, antimony, and nickel of not more than 0.75 percent; the antimony content not to exceed 0.50 percent. Shall be free of lined and unlined standard red car boxes.

Source: National Association of Secondary Material Industries, Circular NF-66.

MAIN LOW GRADE SCRAP AND RESIDUES USED AS FEEDS FOR RECYCLED SMELTERS

REFINERY BRASS

Shall contain a minimum of 61.3 percent copper and maximum 5 percent iron and to consist of brass and bronze solids and turnings, and alloyed and contaminated copper scrap. Shall be free of insulated wire, grindings, electrotype shells and nonmetallics. Hydraulically briquetted material subject to agreement.

COPPER-BEARING SCRAP

Shall consist of miscellaneous copper-containing skimmings, grindings, ashes, irony brass and copper, residues and slags. Free of insulated wires; copper chlorides; unprepared tangled material; large motors; pyrophoric material; asbestos brake linings; furnace bottoms, high lead materials; graphite crucibles; and noxious and explosive materials. Fine powdered material by agreement. Hydraulically briquetted material subject to agreement.

Source: National Association of Secondary Material Industries, Circular NF-66.

APPENDIX III

Statistics compiling criteria

Statistics available for each country are the following:

Mine production: it is referred to the metal content determined by analysis on the concentrates.

Blister production: it is referred to metal produced as blister and anodes from ores, concentrates and other primary materials as well as to secondary blister obtained from scrap. The figures indicate foundry production for single country, part of which may be exported for being successively refined.

Refined production: it ~~is~~ referred to the total production of either electrolytically or fire refined metal. It covers the production from blister, anodes and other primary materials, from scrap and similar materials.

Refined consumption: it is referred to the consumption of refined copper, either primary or secondary, the direct use of scrap being excluded. Consumption is defined as follows:
production + import - export $\pm$ variations in stocks (if any). It is therefore referred to the quantity of refined metal available for consumption without considering whether products are consumed in the country or exported.

Secondary refined production: it refers to the production of refined metal from secondary blister and from scrap and other secondary materials.

Direct use of scrap: it refers to scrap, expressed in recovered copper, which are directly used by consumers. It includes copper contained in alloyed ingots. It is calculated as difference between total consumption and consumption of refined metal.

Total consumption: it includes copper under any form (raw, alloyed, direct use of scrap) and it is expressed in metal content used for semifinished products, castings, chemicals, etc. , without considering whether they are totally used in the country or exported. Total consumption is defined by OECD as consumption at the first processing stage. .

In the majority of OECD member countries , total consumption is calculated by means of coefficients taking into account fire losses. Some other countries do not take into account such losses.

Copper and copper alloy semis production: as concerns the main countries only, it refers to the manufacture of semifinished products, expressed in gross weight, divided as follows:

- copper semifinished products: wires, bars and rods, rolled sections, tubes;
- copper alloy semifinished products: wires, bars and rods, rolled sections, tubes.

For some of the minor countries, figures are collected as a whole.

Foreign trade: data on foreign trade indicate import and export expressed in gross weight of:

- concentrates
- unrefined copper
- refined copper
- alloyed copper
- copper and copper alloy semifinished products
- scrap.

The following OECD countries: F.R. of Germany, France, Italy, Netherlands, United Kingdom, Austria, Finland, Switzerland, regularly publish statistics on copper consumption at the first processing stage divided as follows:

SEMIFINISHED PRODUCTS:

Copper: sheets, strips and plates, wires and plaits, bars and rods, tubes;

Brass: sheets, strips and plates, wires, bars and rods, tubes;

Other alloys:

CASTINGS:

Bronze, brass, and other alloys

CHEMICALS

MISCELLANEOUS

USA, instead, publish statistics on the consumption of refined copper and scrap by the following users:

REFINED COPPER:

Wire mills
Brass mills
Ingot makers
Foundries
Powder plants
Other industries

COPPER AND COPPER ALLOY SCRAP:

Brass mills
Ingot makers
Foundries
Powder plants
Other industries

As concerns the USA, data are also available concerning copper recovered from scrap divided into 'old' and 'new'.

Estimate of copper recovered from scrap

Europe does not compile direct statistics on copper scrap. Secondary production of copper has been derived from the statistics available in each country by adding up production of secondary refined copper and secondary blister to direct use of scrap. Copper recovered from domestic scrap has been derived by adding up the export - import balance (expressed in metal content) of:

- copper alloy ingots
- copper and copper alloy scrap
- copper ashes and residues

to secondary production.

APPENDIX IV

Scrap generation

New scrap

The new scrap generated at the fabricating stage are estimated by means of coefficients applied to the apparent consumption of semi-finished products of copper and copper alloys including castings, expressed in copper content. Apparent consumption (production + import - export), instead of actual consumption, is taken into account, data on stocks being not available.

Sources of coefficients for calculating new scrap at fabricating stage are stated below:

- as to France, they are indicated in the BIPE study titled 'Analyse Technico-Economique de la Recuperation et du Recyclage des Metaux non Ferreux dans la Communaute Europeenne - France - Tome 1';
- as to Italy, they were worked out by Assomet;
- as to the United Kingdom, they are stated in a survey conducted by Charter Consolidated Ltd. and, as a general average, are deducible from a report by BNF Metals Technology Center;
- as to the F.R. of Germany, they were worked out on the basis of indications contained in the following publication 'Angewandte Systemanalyse, Jahresbericht 1976, Anlageband II, H. Blume, Technologien Der Rohstoffnutzung'.

Coefficients regarding bars and rods, plates and strips, tubes, castings vary from one country to another likely according to the pattern of finished products and the structure of manufacturing industry.

As to the previously mentioned semifinished products and castings, the coefficients applied for each country are shown in the table below.

COEFFICIENTS OF NEW SCRAP GENERATION

	Assomet %	BIPE %	Charter %	ASA %
SEMI-MANUFACTURED PRODUCTS				
Copper				
bars and rods	10	10	48	26
plates and strips	40	15	33	60
tubes	10	5	27	36
Brass				
bars and rods	50	45	48	26
plates and strips	40	35	33	60
tubes	10	5	27	36
Other alloys				
bars and others	40	35		
CASTINGS				
Brass	15	15	12	7
Bronze and others	20	20	12	7

With regard to the generation of new scrap in the wire sector, which on average accounts for 50% of copper total consumption, we deemed advisable to apply the same coefficient to all the countries considered.
To this end the data provided in the documentation available were considered; pertinent associations, producers of semi-finished products, fabricators and end users were contacted.
Cable production for transmission and distribution of electric energy, and telecommunications is realized by integrated companies so that a large amount of the new scrap originated is included in the circulatin scrap, and, moreover, their installation is carried out for large quantities. For this category, which accounts for 35%-40% of wire utilization, a 5% coefficient was applied.

With regard to production of transformers, motors and generators, which represents 20% of the total, it was applied a 7.5% coefficient as average value of the wide range of uses which is detected in this sector.

As to wiring cables, wiring accessories, switchgear, telecommunication equipment which assimilate 35%-45% of the wire, a 15% coefficient was applied.

The ponderal average of the three above-mentioned sectors oscillates around 10% and this percentage equals that one drawn from the study by ASA.

In addition, to take into account the fact that in any case part of new scrap generated during the semifinished products production enters the scrap circuit, the new scrap calculated at the fabricating stage was increased by approx. 10% according to an indication derived from the BIPE study.

Ashes, residues and wastes

Neither statistics are available nor assessments appear to have been made from the surveys conducted on the quantity of ashes, residues and wastes generated in the various countries. However, their quantity is deemed to be very modest in comparison with total scrap; it was estimated to represent 5% of secondary production on the basis of information obtained from the surveys conducted on the problem of copper industry recycling. The above products have been included, in relation to their origin, in the new scrap.

Old scrap

The quantity of old scrap recycled is calculated as being the difference between the copper recovered from domestic scrap and new scrap. Though realizing that this procedure is not faultless, the old scrap calculated in this way is considered suitable enough to the aim of this study.

APPENDIX V

Scrap recycling rate

It is defined as the ratio of copper recycled to copper available in scrap; in the case of old scrap it is defined as the ratio of copper contained in recovered old scrap to copper contained in obsolete products.

The life cycle of a product is the time period during which some use for that product exists. It is assumed a product is not available for recovery until its life cycle is complete. Once a life cycle is complete a product may (i) be recovered; (ii) enter a period of non use; (iii) be discarded into an area of future availability, or (iv) be no longer available for recovery because copper was scattered or sacrified during the product's useful life. To evaluate the amount of copper products becoming obsolete in a country in a definite year 'i', it is necessary to know in principle (i) the pattern of consumption of copper products; (ii) the life cycle 'n' of any product; (iii) the foreign trade of semi-manufactured products and castings, and (iv) the foreign trade of the finished products.

Two methods may be used:

I) Global method

This method consists of:

- fixing the average ponderal life 'n' of products according to a given pattern of consumption assumed as constant during the time period considered;
- calculating the finished products entering the domestic market in the 'i – n' time. These finished products are calculated as

follows: apparent consumption of copper and copper alloy semi-finished products minus the new scrap arising from them. These figures are expressed in copper content.

For the highly industrialized countries, the export balance of finished products is considered to be about 20%.

Average life for 1950-1970 period was established in 17 years for USA and 20-22 years for Europe.

II) Method by end-uses

This method is based on the:

- consumption of semifinished products and castings by single type and end-uses;
- life cycle by single end-use.

The quantity of old scrap in the year (i+n) for a specific end-use may be calculated as follows:

$$\sum_{j} (\text{consumption } Cu_i - f)$$

j = different semifinished products and castings by end-uses

n = life cycle of the end-use considered

Cu_i = copper consumed in the end-use considered in the year i

f = 1 - factor of generation of new scrap

The summation of the scrap recoverable from the various end-uses gives the quantity of old scrap theoretically available in the year. This method could give more satisfactory results as compared with the first method but its application is impaired by lack of analytical data, even on an estimate basis.

6. STRUCTURE OF WORLD COPPER PRODUCTION

6. STRUCTURE OF WORLD COPPER PRODUCTION

6.1 The producer countries [1]

6.1.1 World production of ore

Between 1965 and 1974, world production of copper ore increased by about 5 % per year,although there can be very large fluctuations from one year to the next (+ 9 % in 1972 in relation to 1971, - 5 % in 1975 in relation to 1974).

World mine production was 7,296 kt of copper content in 1975, and 7,653 kt in 1974, against 4,962.7 kt in 1965 More than fifty countries are producers, although only a dozen have a production share of more than 2 % (see Table 26).

These twelve countries are responsible for about 87 % of world copper ore production.

In recent years, the principal ore producers were : the United States (17-20 % of world production), the USSR (14-15 %), Chile (10-12 %), Canada (10-11 %), Zambia (9 %) and Zaire (6-7 %).

The share of the United States declined during the last five years (24.4 % of world mine production in 1970 against 17.5 % in 1975), and their production remains under the level reached in 1970.

Zaire's production has also fallen recently because of the problems of ore transport through Angola during the civil war.

The appearance, in 1972, of Papua New Guinea amongst the countries with a 2-4 % share of world production should be noted, as well as the rapid development of Polish production and the relative decline of the share of Peru whose production has stagnated for several years.

In recent years, the developing countries produced around 38-40 % of the world's copper ore, against 36-41 % for the market economy industrialized countries, and 21-24 %

TABLE N° 26

WORLD PRODUCTION OF COPPER ORE IN 1965, 1970, 1973, 1974 and 1975

(in kt Cu)

	1965	1970	1973	1974	1975
United States......	1 226	1 560	1 558	1 449	1 280
U.S.S.R................	650	925	1 060	1 060	1 100
Chile....................	585	692	735	902	828
Canada..................	461	610	824	821	713
Zambia..................	696	684	707	698	677
Zaire.....................	289	387	489	495	495
Poland...................	15	83	152	185	230
Philippines..........	63	160	221	226	226
Australia..............	92	158	220	251	219
South Africa........	60	149	176	179	179
Peru......................	180	220	220	213	174
Papua New Guinea	0	0	0	184	173
Sub-Total	4 317	5 628	6 362	6 663	6 294
World Total	4 963	6 384	7 509	7 653	7 296

for the socialist countries. The distribution of production amongst the three principal groups of nations has remained relatively constant for the last ten years.

6.1.2 World production of blister

World production of blister amounted to 7,276 kt in 1975, 7,537.2 kt in 1974 against 4,963.5 kt in 1965.

The thirteen major producing countries account for 90 % of this production (Table n° 27).

In the last years, the greater part of this production was accounted for by the United States (18-22 % of world blister production), followed by the USSR (15 %), Japan (10-12 %), Chile (9-10 %) and Zambia (9-10 %). These five countries produce around 65 % of the world's blister.

It should be noted that Japan and West Germany, whose mine production is very low, are amongst the ten principal blister producers.

During the last five years, US production declined (24 % in 1970 against 18 % in 1975) while Japanese production has greatly increased (8 % in 1970 against 10 % in 1975).

Of the countries producing between 2 and 7 %, Poland's production has increased rapidly while Peru has shown a relative decline.

In the last years, the developing countries supplied 29-31 % of world blister production, against 45-49 % for the market economy industrialized countries and 22-24 % for the socialist countries, and this proportion has remained relatively constant for the last five years.

6.1.3 World production of refined copper

By refined copper, we mean all metal refined by electrolytic or thermal means, i.e. extracted from ores, concentrates, ash, matte, copper blister, scrap and tailings in the form of wirebars, ingots, cathodes, cakes and billets.

TABLE N° 27

WORLD PRODUCTION OF COPPER BLISTER ([1]) IN 1965, 1970, 1973, 1974 and 1975

(In kt Cu)

	1965	1970	1973	1974	1975
United States	1 240	1 489	1 582	1 424	1 313
U.S.S.R.	650	925	1 060	1 060	1 100
Japan	260	501	817	869	742
Chile	557	647	590	724	724
Zambia	696	683	683	710	659
Canada	385	450	462	516	496
Zaire	289	386	461	468	463
Poland	27	69	152	185	230
Australia	75	112	163	196	180
West Germany	74	84	159	174	168
China and North Korea ..	87	120	140	150	160
Peru	159	177	173	180	156
South Africa	56	148	150	148	151
Sub Total	4 555	5 791	6 592	6 804	6 542
World Total	4 963	6 310	7 287	7 537	7 276

([1]) Excluding production from scrap

TABLE N° 28

WORLD PRODUCTION OF REFINED COPPER IN 1965, 1970, 1973, 1974 and 1975

(In kt Cu)

	1965	1970	1973	1974	1975
United States	1 942	2 035	2 098	1 940	1 609
U.S.S.R.	770	1 075	1 300	1 350	1 420
Japan	366	705	951	996	819
Zambia	522	581	638	677	629
Chile	289	465	415	538	535
Canada	394	493	498	559	529
West Germany	342	406	407	424	422
Zaire (1)	151	251	232	324	305
Belgium-Luxembourg	300	276	367	283	259
China and North Korea..	110	130	220	240	250
Poland	37	72	157	195	249
Australia	93	146	178	195	193
Sub-Total	5 316	6 635	7 461	7 721	7 219
World Total	6 058	7 578	8 524	8 899	8 369

(1) Including recoverable copper content in the exported cathodes to Belgium refineries

World production was 8,369 kt in 1975, 8,899.4 kt in 1974, against 6,059 kt in 1965, and in the last years most of which was accounted for by the United States (19-25 %), the USSR (15-17 %), Japan (10-11 %), Zambia (8 %), Chile (5-6 %), Canada (6 %) and West Germany (5 %). These seven countries produce more than 65 % of the world's refined copper (Table 28).

As concerns the principal refined copper producers, the fall in US production during the last five years, as well as the increase in Poland's share, should be noted.

In 1975, the developing countries produced 16-19 % of the world's refined copper, against 57-59 % for the market economy industrialized countries and 23-27 % for the socialist countries. There have been little changes in these proportions during the last five years since, in 1965, the developing countries accounted for 20 % of world refined copper production against 63 % for the market economy industrialized countries and 17 % for the socialist countries. The share of the market economy industrialized countries has dropped and that of the socialist countries has grown.

6.1.4 Principal producer countries

World copper production is thus distributed amongst many countries in which should be distinguished (Table 29):

- The two principal ore producing countries, the United States and the USSR, with industries which are highly integrated downstream and are therefore relatively independant of the world market;

- The ore producing countries, which export refined copper along with ore and blister. This group consists basically of Canada and the CIPEC countries;

- Countries with only few raw material resources, but which have a large processing and consuming industry (basically, the EEC and Japan).

TABLE N° 29

PRODUCTION OF ORE, BLISTER AND REFINED COPPER IN 1975

(In kt Cu)

	Ore Production	Blister Production	Refined Copper Production
Countries with integrated industries			
. United States	1 280	1 313	1 609
. U.S.S.R.	1 100	1 100	1 420
Basically ore producing countries			
. Chile	828	724	535
. Canada	713	496	529
. Zambia	677	659	629
. Zaire	495	463	305
. Philippines	226	0	0
. Australia	219	180	193
. Peru	174	156	53
. Papua New Guinea	173	0	0
Industrialised countries importing ore or blister			
. Japan	85	742	819
. EEC	13	188	886
Sub-Total	5 982	6 021	6 978
World Total	7 296	7 276	8 369

6.2 The CIPEC, a producer Association

The CIPEC - (Conseil Intergouvernemental des Pays Exportateurs de Cuivre, or Intergovernmental Council of Copper Exporting Countries) - was created in 1967 in Lusaka by Chile, Peru, Zaire and Zambia which were joined, in 1975, by Indonesia, with Australia and Papua New Guinea becoming associate members at the same time.

In 1975, the countries of CIPEC (including the associate members) represented 36 % of world mine production, (20.5 % of refined copper production) but 70 % of world trade.

The CIPEC was created in order to coordinate and harmonize member countries' decisions and policies concerning copper production and marketing problems, supply complete and accurate information to the member countries and increase resources derived from copper in order to promote economic and social development, while taking into account the interests of the consumers.

The CIPEC has not, in recent years, achieved its intended objectives as concerns its efforts tending to stabilize the international copper market and support metal prices.

From 1967 to 1974, it only carried out copper studies. In February 1974, it acted with respect to the United States, so that the latter country would not abruptly sell part of its strategic stock; this was later made only on the domestic market. In July 1974, it acted with respect to Japan in order to convince this country to end its re-exports of refined copper. Lastly, in November 1974, the CIPEC decided to reduce member countries' exports by 10 % in order to stop the fall in prices. In November 1975, the reduction was raised to 15 % and accompanied by a production cut of 10 %. The effect on prices was very slight, and the planned reductions were not always applied by the member countries.

6.3 Producer companies and the policies of producer countries

6.3.1 Producer companies

Of the ten principal copper ore producing companies in 1973 (table 30), four are US companies and four are controlled by State.

The total production of these ten companies represented 42 % of world mine production in 1973 (including the socialist countries).

At the copper metal production level four companies amongst the first twelve are US (table 31). Two belong to Zambia and international interests, one is Zaīrian, one Chilean, one Canadian, one Japanese and two European.

The activity of the copper producing companies is generally intergrated vertically. At the ore-metal transformation level, most of the ten principal mining companies produce at least as much metal as was contained in the mined ore.

Of the principal copper metal producing companies five are not amongst the principal ore producing companies i.e. Noranda and Asarco whose mining activity is more limited and Hoboken Nippon Mining and Norddeutsche Affinerie which import ore for smelting and refining.

6.3.2 Policies of the Producer Countries

US and Canadian companies essentially account for production in these two countries.

The copper industry is privately owned in the free enterprize countries such as the US, Canada, Republic of South Africa, Australia, etc.. In the developing countries, however, there is a definite tendancy towards nationalisation through the imposition of state ownership and/or the participation of local capital. It should be noted that this tendancy also exists in some industrialized countries such as Australia where since 1976 local participations in all projects must be at least 50 %. Below will be given some examples of national participation in the copper industry of several developing countries.

In Zaire, since the government owns 100 % of Gécamines which exploits 95 % of Zairian copper while Sodimiza (government

TABLE N° 30

PRINCIPAL MINING COMPANIES IN 1973

Company	Shareholders	Production in kt
Codelco	100 % Chilean State	615
Gecamines	100 % Zaïrian State	460
Kennecott	US company	430
N'Changa Cons. Mines	51 % Zambian State 49 % Anglo American	410
Phelps Dodge	US company	290
Roan Cons. Copper Mines	51 % Zambian State 36,75% Amax 12,25% Anglo American	230
Newmont Mining	US company	195
Anaconda	US company	190
Bougainville Copper Ltd	53,6 % Conzinc Rio Tinto of Aust. Ltd 20 % Government of Papua New Guinea and Investment Corp. of Papua New Guinea 26,4 % Others	185
Inco	Canadian company	150

TABLE N° 31

PRINCIPAL METALLURGICAL COMPANIES IN 1973

Company	Shareholders	Production in kt
Phelps Dodge	US company	480 *
Gecamines	Zaīrian state	460
Kennecott	US company	420 *
Codelco	Chilean state	415
Noranda	Canadian company	380
Asarco	US company	375
N'Changa Cons. Mines	51 %Zambian state 49 %Anglo American	365
Hoboken	Belgium company	330 *
Nippon Mining	Japanese company	320
Roan Cons. Copper Mines	51 %Zambian state 36,75 %Amax 12,25 %Anglo American	275
Norddeutsche Affinerie	West Germany company	240 *

* Capacity

capital 20 %, Japanese capital 80 %) accounts for the rest. Zaire owns 20 % of the Tenke Fungurume project (delayed).

On 1st January 1970, Zambia took over a 51 % share in all the copper mines of the country and in 1973 the bonds issued as part of this operation were reimbursed and the government took over actual control of the mines which previously belonged to Amax and Zambian American Trust (subsidiary of Charter and Anglo American of South Africa). The government thus now has a majority share in the two operating companies, Roan Consolidated Copper Mines and N'Changa Consolidated Copper mines.

In Peru the principal mining company is the Southern Peru Copper Corporation (subsidiary of US companies) and the second copper mine producer is Centromin Peru (state company) which in December 1973 took over the interests of the Cerro de Pasco Corporation in Peru.

The government has no share in the Cuajone mine whose exploitation has just begun but the two state companies Minero Peru and Centromin Peru are in charge of all other projects (Cerro Verde, etc.).

In Papua New Guinea the government has a 20 % share in the Bougainville mine which accounts the country's entire copper production.

In Chile, the Allende government nationalised the five largest mines (Chuquicamata, El Teniente, Salvador, Andina and Exotica which in 1974 represented nearly 85 % of Chilean copper production). These mines belonged to US groups i.e. Anaconda, Cerro Corporation, Kennecott.

In 1974, the Pinochet government recognised the need for foreign technological and financial support along with national investments and tried to attract foreign capital. In December 1975 Chile decided to develop the Andacollo mine with Noranda where Chile will have a 51 % share against 49 % for Noranda. Other projects of the type are also planned.

6.4 Cost Prices for the Production Sector

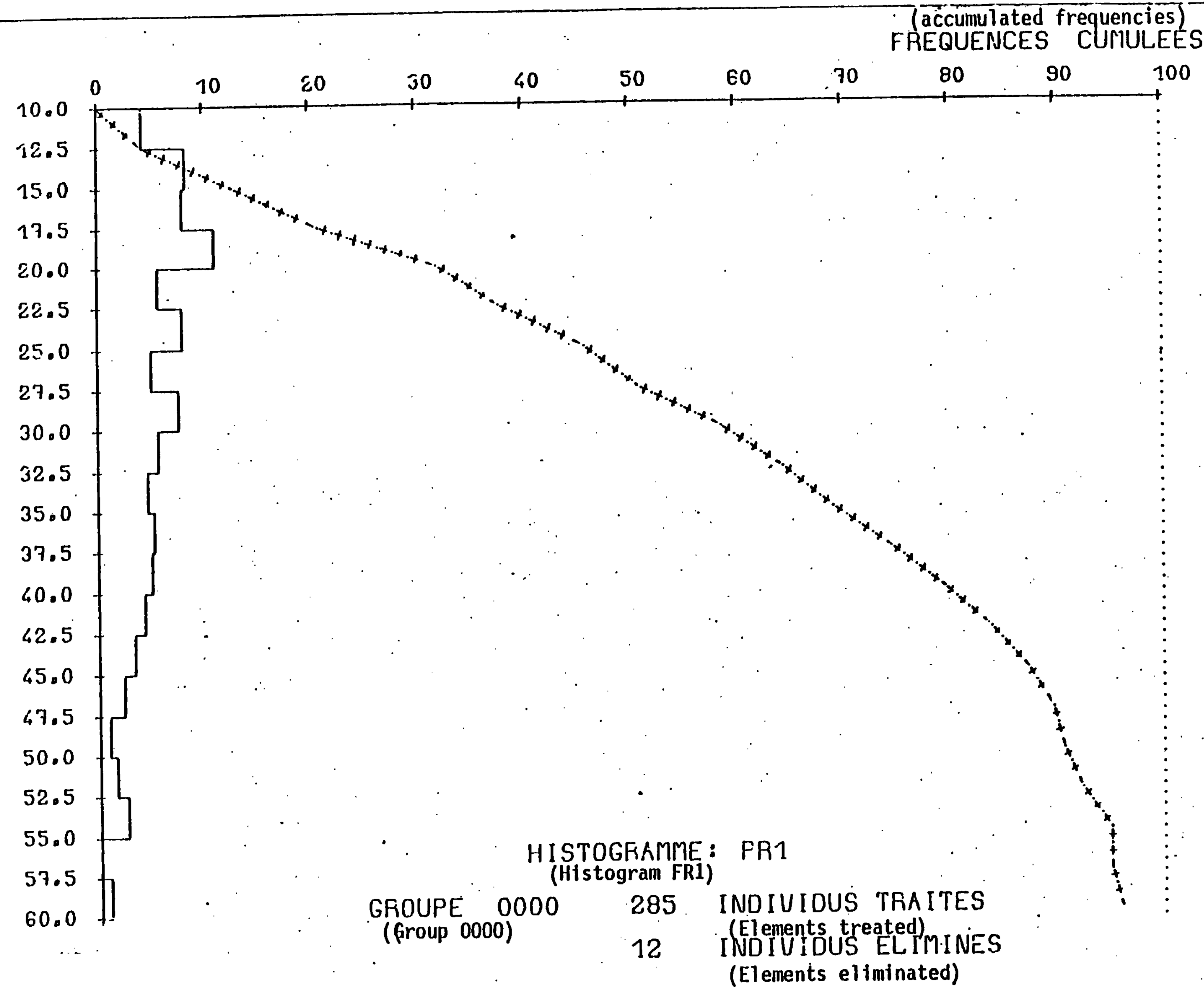
The copper industry is characterised by large differences in production costs. The range seems to be less pronounced for investments. The characteristic common to both production and investment costs in the copper industry is the large increases in both without a corresponding rise in sales prices. Many projects have therefore been confronted with depressed markets and very high costs while some exploitations are no longer sufficiently profitable. The result of the imbalance between cost prices and sales prices has been that some projects have been abandoned and the most vulnerable mines have been closed. While in the short and middle term this might seem to be a regulatory mechanism for a surplus market, the long term situation could be different in view of the time necessary for starting new projects essential to the future supply of the world market.

6.4.1 Copper Industry Production Costs

In recent years costs have risen considerably at the three conventional production stages i.e. mining, smelting and refining. Mining costs were especially affected by the general inflation, the drop in grades, increasing difficulties in exploitation and the decrease in the activity of production units. Costs for the processing of concentrates were affected by energy prices and anti-pollution measures. Hydrometallurgical methods may seem more competitive in view of these rises which have a special effect on pyrometallurgical methods but their large scale development depends upon replacement needs and new projects.

There are large differences in mining production costs. More than 650 exploitations (or groups of mines) were surveyed in a BRGM study in 1975 and their economic characteristics were compared[2]. More than 300 data were available concerning concentrate costs and the joined histogram shows the distribution of these costs in 1974. The average was then close to 28 cts/lb. (Fig. 29)

Production costs depend to a considerable extent upon transport costs and tariffs related to the geographical location of the deposit. Profits are very dependent upon tax laws and many countries draw most of their income from taxes on mineral raw materials. More generally, deposit conditions, type of



HISTOGRAMME DES PRIX DE REVIENT
(HISTOGRAM OF COST PRICES)

Fig. 29

exploitation, ore grade and local economic conditions all result in variations in average costs amongst the different geographical zones. Thus North American costs are highest and are followed by those for Africa, Western Europe, Latin America, Asia and Australia. While, in the middle term, North America seems to be the region which have the highest costs, these should rise in Europe and Latin America, while deposits in Africa, Asia and Australia should have the lowest costs.

In North America estimated cost prices have risen by 36 % between 1974 and 1975 although this region is relatively favored as concerns energy. The major cause of the increase in cost prices is to be found in the fall in activity of the production units. In 1977 United States copper production has an average cost of 62 to 65 cts/lb[3]. The range of US costs is 58-76 cts/lb which breaks down as follows : mining (10-15), concentration (15-20), smelting-refining (23-26), overhead (10-15). There was a rise in the relative share of processing costs in recent years especially in the United States where these costs are now 30-40 % of the operating costs of the sector. The reason for this is mounting energy and anti-pollution costs.

More generally, in a recent study, CIPEC estimated that world wide production costs for the entire sector could hardly be below 50 cts/lb [4] .

6.4.2 Investments for the Copper Industry

In recent years, investments have increased more quickly than the rate of inflation which is only one of the factors involved. Costs, and therefore the financial charges of investments, are increased by the low accessibility of new projects, the general fall in the ore grades of new deposits and anti-pollution measures in the developed countries.

According to the CRU of London [5] , investments in constant 1975 terms for a tonne of copper capacity rose from $ 2,900 in 1955 to $ 3,400 in 1965, $ 4,500 in 1970 and $ 5,900 in 1976. Estimates by CIPEC run from 5,000 to 7,000 $/t.

A Canadian study for UNCTAD [6] estimated capital needs for developing new deposits on the basis of existing data for mines and processing units opened in Canada between 1969 and 1975. According to this study, investment costs per tonne of annual production capacity for copper are close to 7,000 $/t with 1,000 to 1,700 $/t for the smelter and 400 to 500 $/t for the refinery. Some projects have higher figures for the processing units, for example a Philippines project has costs of 3,000 $/t for the smelter-refinery. Anti-pollution measures have had a very important effect in the developed countries and in Japan, a smelter-refinery complex with costs estimated at 550 $/t a few years ago saw these rise to 2,500 $/t mainly as a result of these regulations.

Only a few projects are presently starting up. The last large project, Cuajone in Peru, cost $ 4,800 per tonne of blister to which should be added 400 $/t for the Ilo refinery. The last estimates for the Sar Chesmeh project in Iran are above 7,000 $/t although the ore grade is on the order of 1.12 %. Costs per tonne may be lower for average mines depending upon mine conditions.

6.4.3 Conclusion

Investment costs for new projects should be near 7,000 $/t which represents about 40 cts/t of amortization. Since operating costs could be close to 50 cts/lb, it is clear that copper from these new deposits will have a cost price of 90 cts/lb which is much higher than 1977 sales prices. Since it is unlikely that there will be a short term change in the causes of this trend, it is likely that costs will increase still further. Only new technical developments and the utilisation of cheaper hydrometallurgical processes could change this trend whose long term effects could endanger world supply if it is not possible to develop new projects.

6.5 Perspective for World Production and Trend of Production Structure

The foreseeable trend of the structure of world production depends upon the data available concerning the development of mining and refinery projects. The projects have been inventoried and their characteristics (capacity, starting data, etc.) have been reviewed.

We have thus taken into account all available data for the Western World in mid 1977 and each project has been allocated a realistic capacity and starting date (i.e. by correcting some estimates which seemed overly optimistic).

Tables 32 and 33 present this timetable.

There are considerable differences of opinion as to the starting dates when corrective hypothesis are applied. Since there are problems involved in estimating production capacities and world supply forecasts cannot be formulated solely on the basis of general surveys of new capacity starting dates. It is not possible to present an exact timetable even for the short term because projects, especially the large ones, are affected by technical, financial, commercial and political conditions.

Long term forecasts can therefore only be made on the basis of trend estimates for the production structure which is much less sensitive to the temporary effects of the economic situation on a given project.

An analysis of this kind can be made taking into consideration the projects mentioned in the preceding chapter, whatever may happen in any given case, since the broad lines will probably prove to be accurate.

6.5.1 Perspectives for Mine Production in the Western World and Production Structure Trends

In spite of possible variations in the starting dates for certain projects, copper mine production for the Western World should show a relatively large increase of more than 5 % per year from the present up to 1980 (not taking into account mine closures). The forecasts are less certain after this date and all the large projects which have been slowed down recently because of technical or economic considerations seem to have been programmed for 1982 while in reality these projects will

TABLE N° 32

COPPER MINE PROJECTS IN THE WESTERN WORLD *

(In kt Cu)

Capacity in 1976	7 366
Additional capacity in	
1977	503
1978	384
1979	429
1980	465
1981	246
1982	805
After 1982	70

* Data on BRGM estimates

TABLE N° 33

COPPER REFINERY PROJECTS IN THE WESTERN WORLD*

(In kt Cu)

Capacity in 1976	8 811
Additional capacity in :	
1977	140
1978	565
1979	545
1980	278
1981	155
1982	478
After 1982	100

* Data on BRGM estimates

surely not be completed before this date but rather only after 1982.

Taking into account these assumptions, there should be a marked trend in the mine production structure in the next few years (Table 34). The share of the developing countries should markedly increase and supply more than 85 % of mine production growth with their countries share in western production rising from 52 % in 1975 to 64 % after 1982.

The largest projects in the developing countries will be in Latin America (Peru, Chile, Mexico) rather than in Central Africa where the lack of infrastructure as well as risks due to the continent's political instability do not encourage investments. However if local conditions improve this trend could change.

On the other hand, in the market economy industrialized countries the drop in the share of North America appears inevitable even if the present timetable does not include all extensions or small projects which might be completed in short periods of time.

The share of South and Central America in world supply should therefore increase. It is however true that this trend which is certainly well founded may be more or less marked depending upon the financial possibilities of the countries concerned.

At the worldwide level, the shares of other groups will remain more or less as they are even if within each group the trend will vary for the various countries.

The opposing trends for the two principal blocks within the CIPEC leads to the supposition that the increased influence of this organisation in world production will be due to new joining members rather than to the production increase in the founder members.

TABLE N°34

STRUCTURE OF COPPER MINE PRODUCTION IN THE WESTERN WORLD IN 1976 AND TREND AFTER 1982

(In kt Cu)

	Capacity at end of 1976		Forecast New Capacity from now to 1982		Total after 1982	
Europe	399	(5,4)*	26	(0,9)	425	(4,1)
Africa	1 756	(23,8)	328	(11,3)	2 084	(20,3)
Asia	593	(8,1)	292	(10,1)	885	(8,6)
North America	2 842	(38,6)	314	(10,8)	3 156	(30,7)
Central and South America	1 324	(18,0)	1 756	(60,5)	3 080	(30,0)
Oceania	452	(6,1)	186	(6,4)	638	(6,2)
TOTAL	7 366		2 902		10 268	

* Numbers in parentheses show region's production as a percentage of the Western World's.

6.5.2 Perspectives for Refined Copper Production in the Western World and Production Structure Trends

The projects for copper refinery development up to the end of 1979 seem to be in agreement with the planned increases in mine capacity, but after 1980 these forecasts are much lower. This is due to the fact that the time necessary for developing a refinery is much shorter than that for a mine and the shortage of refinery projects after 1980 is therefore only apparent.

The refined copper production structure of the Western World (Table 35) shows tendancies which are similar but less marked than those of the mine production trend :

- Increased importance of Latin America ;
- Fall in the shares of Central Africa and North America.

The developing countries' share in the Western World's refined copper production should remain smaller than that of the market economy industrialized countries (rising however from 28 % in 1975 to 43 % after 1982).

TABLE N° 35

STRUCTURE OF REFINED COPPER PRODUCTION IN THE WESTERN WORLD

IN 1976 AND TREND AFTER 1982

(In kt Cu)

	Capacity at end of 1976		Forecast New Capacity from now to 1982		Total Capacity after 1986	
Europe	1 747	(19,8)*	160	(7,1)	1 907	(17,2)
Africa	1 109	(12,6)	225	(10,0)	1 334	(12,0)
Asia	1 333	(15,1)	349	(15,4)	1 682	(15,2)
North America	3 497	(39,7)	230	(10,2)	3 727	(33,7)
Central and South America	901	(10,2)	1 227	(54,3)	2 128	(19,2)
Oceania	224	(2,5)	70	(3,1)	294	(2,7)
TOTAL	8 811		2 261		11 072	

* Numbers in parentheses show region's production as a percentage of the Western World's.

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7. STRUCTURE OF CONSUMPTION, WORLDWIDE

7 STRUCTURE OF CONSUMPTION, WORLDWIDE

7.1 Ore consumers

No direct uses of copper ore for the manufacture of products at the first processing stage or finished products are known.

7.2 Metal consumers

The largest use of copper is in electrical equipment and supplies. The manufacture of electric motors, power generators, motor-generator sets, dynamotors, fans, blowers, industrial controls, and related apparatus requires copper for the best electrical performance. Electrical instruments and test equipment, power distribution systems including transformers, bus bars, and switchgear, and electric lighting and wiring equipment require large quantities of copper. Dependable all-weather service on crowded airways requires sophisticated electronic navigation and communication systems that rely on copper in the form of cable and related electric parts. Although aluminium is used for virtually all high-voltage overhead power transmission lines, copper is widely used in underground lines and dominates the smaller gage wire market.

The corrosion resistance of copper and its alloys results in many uses in the construction industries. Construction materials for roofing and plumbing and brass and bronze for decorative and utilitarian items for public buildings and private homes require significant quantities of copper.

Copper finds widespread use in the production of nonelectrical industrial machinery, household and commercial air-conditioning, and farm machinery. Copper and copper-base alloys that resist the corrosive effects of seawater at temperatures up to 120°C and that also have high heat-transfer capability are used for tubing and valves in distillation plants for seawater desalination.

Copper finds wide application in the automobile industry, in railroad transportation, in airplane manufacture, and in marine parts. This results both from the trend toward convenience items such as power windows, seats, brakes, and steering, and air-conditioning and from the more utilitarian uses in radiators, heaters and defrosters, bearings, bushings, carburettors, oil lines, and wiring. Large quantities of copper are used in diesel locomotives, railroad passenger cars, and switching and signal devices.

Copper items used for ordnance include shell casings, projectile fuses and rotating bands, and fire-control equipment.

Miscellaneous uses of copper include chemicals and inorganic pigments. Copper is also used extensively in watches, clocks, microscopes, projectors, and many types of gages. Solid copper, brass and bronze are popular materials in utensils, jewelry, furnishings, and decorative items. Conversion of coinage from silver alloys to coins having a sandwich structure containing copper has increased copper consumption.

7.2.1 Consumption of refined copper

During 1963-1973 decade, consumption of refined copper in the Western world rose to 7.0 M tons from 4.4 M tons with 4.6 % annual average rate of growth. During the same period of time the Eastern world registered 5.2 % average annual growth rate thus reaching 1.8 M tons from the previous 1.1 M tons. Thereafter, due to the very serious economic crisis which hit the Western world, a recession phase occurred which reached its peak in 1975, when consumption in the Western world fell to 5.5 M tons (- 11.4 % average annual rate compared to 1973).

Towards the end of 1975 demand started to become lively and in 1976 consumption, by accounting for 6.4 M tons, showed a partial recovery. However, the average annual decrease was of 2.7 % in 1973 to 1976. The Eastern world on the contrary was not particularly affected by the crisis suffered by the Western world and it did not register drops in demand, but just a slight slowing down of the average annual growth rate which declined to 4.8 % in the years 1973 to 1976.

With regard to the evolution of demand in the main economic areas of the Western world, the United States, which accounted for 36.0 % of world consumption in 1963, fell to 31.9 % in 1973, though registering an average annual increase of 3.4 %.

During the decade in question, the leading economy was in fact represented by Japan's. This country reached 17.2 % of world demand in 1973 against 7.9 % in 1963 with an annual growth rate of 13.1 %. The other countries of Asia, Africa and Oceania, though registering increase rates even higher than the general average (slightly over 5 % of the total).

American countries others than the United States covered by contrast a more important role than in the past by reaching 7.5 % in 1973 against 5.9 % registered in 1963.

The growth rate of European countries who are not EEC members resulted as a whole lower than the average world growth rate (+ 3.6 %) therefore from 1963 to 1973 their quota on world total went down to 6.8 % from 7.5 %.

In 1973 to 1976, Oceania, Africa, minor countries of Asia and America countries others than the United States slightly increased as a whole their incidence on the world total by reaching 13.5 % in 1976 from 12.7 % registered in 1973.

This improvement was due mainly to increases occurred in Brazil, minor countries of Asia and America and it was partially counterbalanced by decreases occurred in Africa, Canada and Australia. The non-EEC member European countries who totally registered 0.7 % average growth rate in the last three years accounted again for 7.5 % of the world total over the last year.

The copper industry in the United States and Japan was affected more than in other countries by the slump of the recent years and it showed greater difficulties in recovering in 1976.

In 1973 to 1976 the average annual decrease was of 4.4 % in Japan and 7.1 % in the USA thus representing 16.4 % and 27.7 % respectively of world consumption.

EEC as a whole moved upward to 2.2 M tons in 1973 from 1.6 M tons registered in 1963. In 1974 it maintained the same level registered in 1973. In 1975 it fell slightly below 2 M tons and in 1976 it recovered all the consumption lost in the previous year.

EEC demand therefore behaved differently from that in the other main economic areas of the Western world. Both growth and recession phenomena were in fact mitigated. Growth rates as well as decrease rates were therefore more limited compared to both the world average and, especially, USA and Japan.

In 1963 to 1973 EEC registered an average annual increase of 2.9 % and during the following threee years it registered a decline of just 0.2 %. As a consequence EEC quota in respect of the total declined to 31.4 % from 37.3 % in the period 1963 to 1973, it then rose again during the following years (in 1976 it reached 33.9 %).

Evolution of demand behaved differently in the EEC member countries. With regard to the main countries only, the F.R. of Germany by registering 4.0 % average annual increase rate in 1963 to 1973 and 0.8 % during the following three years maintained practically unchanged its quota on the world total oscillating generally between 11 % and 12 %.

In the United Kingdom, consumption remained almost constant from 1963 to 1973 thus representing a unique example amongst the main industrialized countries. From 1973 to 1976 it showed on the contrary a 5.4 % average annual decrease. As a consequence the UK quota on the world total gradually declined to 7.2 % in 1976 from 12.6 % registered in 1963.

In France consumption increased by 5.0 % in 1963 to 1973, and it showed a 3.4 % average decline thereafter. France accounts for approx. 6 % of world consumption.

Italy, which accounts for about 5 % of the world total, registered 2.8 % average increase rate in 1963 to 1973, lower than of the other main EEC countries, but unlike these ones and notwithstanding a drop occurred in 1975, it maintained 2.4 % average growth rate in 1973 to 1976.

Within EEC, Belgium by registering 6.2 % average annual increases in 1963 to 1973 and 11.5 % thereafter, was the country who showed a higher increase in percent and its quota on the world total rose to 2.7 % from 1.6 %.

7.2.2 Total consumption of copper

Total consumption of copper in the Western world increased to 9.5 M tons in 1973 from 6.2 M tons in 1963 with 4.5 % average annual rate of growth.

Later on after the decrease registered in 1974 and the serious fall in 1975, there was a partial recovery in 1976 which brought consumption to 8.7 M tons with an average annual decrease of 3.1 %. In terms of percent, the quotas of total consumption and consumption of refined copper in the Western world held by Africa, minor countries of Asia, Oceania, Americ American countries others than the USA, non-EEC member countries are practically the same.

In the United States the total consumption quota is on average higher than that referring to refined copper by about 2 percentage points, whilst in the EEC the opposite phenomenon occured with a difference of 1-2 percentage points and in Japan both quotas overtook each other alternatively, fluctuations however remaining modest.

In particular with regard to the main EEC member countries compared with the Western world as a whole, it is to be emphasised that Germany F.R. which accounter for 11 %-12 % as consumption of refined copper registered only 10 % of total consumption.

Also in the UK and Belgium the quotas were slightly reduced.

As to France, variations in the two quotas were registered over the years which were however very modest, whilst as to Italy, despite fluctuations, percentage of total consumption was higher than that of refined copper consumption.

The above differences depend on the different breakdown of the total consumption in refined copper and direct use of scrap including alloy ingots.

As to considerations on direct use of scrap, the chapter regarding supply by scrap and wastes refers.

In the Western world, despite fluctuations through the years, total consumption consisted of on average 71 %-72 % of refined copper in the 60s, and 73 %-74 % in the first half of the 70s (1976 included). These average percentages are the result of the different situations which for the main economic areas are indicated hereinafter.

The United States, which consumed 68 % of refined copper in 1963, reached about 70 % in 1973 and this figure remained almost unchanged also in the following years, though registering a decrease in 1975. Japan moved up to 74 % in 1973 from the previous 63 % in 1963, it then registered a decline but went up again to 74 % in 1976.

EEC remained on average at around 74 % from 1963 to 1973, registering then an increase to over 78 % in 1975 and declining to 75.5 % in the following year.

GROWTH IN COPPER CONSUMPTION
(Percentage per annum)

	Refined consumption			Total consumption		
	1973/63	1975/73	1976/73	1973/63	1975/73	1976/73
Germany F.R.	+ 4.0	- 6.6	+ 0.8	+ 3.8	- 9.6	- 0.6
France	+ 5.0	- 5.5	- 3.4	+ 4.5	- 9.3	- 2.1
Italy	+ 2.8	- 0.2	+ 2.4	+ 3.7	- 4.7	+ 2.8
Netherlands	+ 4.1	- 1.3	+ 6.4	+ 0.6	- 4.7	+ 1.2
Belgium-Lux.	+ 6.2	+ 2.9	+11.5	+ 5.6	+ 1.2	+ 7.8
United Kingdom	- 0.3	- 8.8	- 5.4	+ 0.3	-10.6	- 5.6
Ireland	(1)	(1)	(1)	(1)	(1)	(1)
Denmark	+ 3.5	-16.9	- 3.8	+ 2.6	+ 5.9	+ 9.9
E E C	+ 2.9	- 5.2	- 0.2	+ 3.0	- 8.0	- 0.8
Other Europe	+ 3.6	- 1.8	+ 0.7	+ 4.7	- 3.2	+ 0.1
EUROPE	+ 3.0	- 4.6	- 0.0	+ 3.3	- 7.1	- 0.6
South Africa	+ 7.7	+ 2.1	- 4.9	+ 7.9	+ 1.5	- 5.2
Other Africa	+ 4.3	- 4.3	- 0.4	+ 5.3	- 6.3	- 1.3
AFRICA	+ 6.6	+ 0.3	- 3.6	+ 7.1	- 0.6	- 4.1
Japan	+13.1	-17.0	- 4.4	+11.2	-15.1	- 4.5
Other Asia	+ 2.7	- 3.2	+ 8.1	+ 4.1	- 6.2	+ 7.2
ASIA	+11.4	-15.5	- 3.0	+10.1	-14.1	- 3.1
Brasil	+12.6	+11.3	+12.7	+12.1	+10.3	+12.1
Canada	+ 4.5	-13.6	- 6.0	+ 3.6	-15.0	- 7.3
USA	+ 3.4	-20.7	- 7.1	+ 3.2	-19.7	- 7.0
Other America	+ 8.4	+ 6.4	+ 7.6	+ 6.4	+ 3.8	+ 7.5
AMERICA	+ 4.0	-16.7	- 5.1	+ 3.6	-16.7	- 5.3
OCEANIA	+ 5.1	-13.1	- 6.1	+ 5.2	-12.0	- 5.4
WESTERN WORLD	+ 4.6	-11.4	- 2.7	+ 4.5	-12.2	- 3.1
EASTERN WORLD	+ 5.2	+ 5.1	+ 4.8			
T O T A L	+ 4.7	- 7.7	- 1.0			

(1) Insignificant data

REFINED COPPER CONSUMPTION

'000 tons

	1963	1968	1973	1974	1975	1976
Germany F.R.	493.5	608.8	727.2	731.0	634.6	744.6
France	250.3	292.9	407.8	414.2	364.5	367.1
Italy	228.0	226.0	300.0	316.0	299.0	322.0
Netherlands	25.5	34.3	38.2	37.7	37.2	46.0
Belgium-Lux.	90.0	122.0	164.4	178.2	174.2	228.1
United Kingdom	557.9	539.2	541.2	496.9	450.5	457.6
Ireland	-	-	0.3	0.3	0.2	0.4
Denmark	3.9	2.9	5.5	4.7	3.8	4.9
E E C	1,649.1	1,826.1	2,184.6	2,179.0	1,964.0	2,170.7
Other Europe	332.6	348.9	472.9	507.4	455.6	483.4
EUROPE	1,981.7	2,175.0	2,657.5	2,686.4	2,419.6	2,654.1
South Africa	30.0	29.0	62.7	67.3	65.3	54.0
Other Africa	15.7	10.7	24.0	24.3	22.0	23.7
AFRICA	45.7	39.7	86.7	91.6	87.3	77.7
Japan	352.1	695.2	1,201.8	880.9	827.4	1,050.3
Other Asia	105.0	66.2	137.7	166.4	129.0	174.1
ASIA	457.1	761.4	1,339.5	1,047.3	956.4	1,224.4
Brasil	38.3	57.0	125.3	162.0	155.2	179.3
Canada	159.9	232.2	248.2	248.0	185.2	206.2
USA	1,590.0	1,701.4	2,221.1	1,994.9	1,396.3	1,778.2
Other America	63.9	105.6	143.8	152.9	162.7	178.9
AMERICA	1,852.1	2,096.2	2,738.4	2,557.8	1,899.4	2,342.6
OCEANIA	83.7	103.5	137.4	123.9	103.7	113.6
WESTERN WORLD	4,420.3	5,175.8	6,959.5	6,507.0	5,466.4	6,412.4
EASTERN WORLD	1,099.0	1,346.3	1,817.3	1,840.3	2,007.7	2,092.2
T O T A L	5,519.3	6,522.1	8,776.8	8,347.3	7,474.1	8,504.6

TOTAL COPPER CONSUMPTION

'000 tons

	1963	1968	1973	1974	1975	1976
Germany F.R.	631.8	766.2	919.0	875.3	751.2	901.9
France	366.9	414.4	568.6	535.3	468.2	533.7
Italy	331.0	382.0	478.0	503.0	434.0	520.0
Netherlands	59.9	77.8	63.6	65.5	57.8	66.0
Belgium-Lux.	117.8	152.5	202.6	211.4	207.3	254.1
United Kingdom	695.5	679.7	718.7	661.0	574.8	604.6
Ireland	-	-	0.3	0.3	0.2	0.4
Denmark	4.5	6.8	5.8	7.5	6.5	7.7
E E C	2,207.4	2,479.4	2,956.6	2,859.3	2,500.0	2,888.4
Other Europe	432.0	485.0	680.9	717.1	637.8	682.6
EUROPE	2,639.4	2,964.4	3,637.5	3,576.4	3,137.8	3,571.0
South Africa	40.0	40.0	85.7	92.3	88.3	73.0
Other Africa	19.7	14.7	33.0	33.3	29.0	31.7
AFRICA	59.7	54.7	118.7	125.6	117.3	104.7
Japan	562.1	1,010.2	1,617.8	1,323.9	1,167.4	1,409.3
Other Asia	125.0	87.2	187.7	216.4	165.0	231.1
ASIA	687.1	1,097.4	1,805.5	1,540.3	1,332.4	1,640.4
Brasil	45.3	69.0	142.3	185.0	173.2	200.3
Canada	199.9	265.2	284.2	285.0	205.2	226.2
USA	2,327.5	2,554.9	3,178.0	2,873.0	2,047.5	2,558.6
Other America	103.9	145.6	193.8	214.9	208.7	240.9
AMERICA	2,676.6	3,034.7	3,798.3	3,557.9	2,634.6	3,226.0
OCEANIA	108.7	140.5	180.4	166.9	139.7	152.6
WESTERN WORLD	6,171.5	7,291.7	9,540.4	8,967.1	7,361.8	8,694.7

7.2.3 Copper and copper alloy semis and castings

The production of copper and copper alloy semis and castings accounts for almost all the copper used by the processing industry of each country.

The evolution of this sector and the various countries' quotas on the world total are therefore practically the same as those of copper consumption as a whole.

Only a small percentage of total consumption of copper, equal to 2 %-3 % of the Western world's demand, is in fact destined for the production of copper sulphate, copper salts and copper powders.

Referring only to the semifinished products, in the USA and Japan nearly 75 % of production expressed in metal content consists of copper semis and the remaining part of copper alloy semis.

In Europe the quota of copper semis is slightly lower and therefore that of copper alloy semis is slightly higher.

Generally the bulk of the semis and castings produced is consumed domestically, and with only few exceptions international trade in semis is limited to a small portion of total production.

Though the USA produces about one third of the Western world's semis production, it is on balance a net importer of both copper and copper alloy semis even if for a very small quota of its apparent consumption. On the contrary Japan is a net exporter. In 1976 Japan's exports were nearly equal to 10 % of its production, but this percentage was usually lower in the previous years.

Also EEC is an exporter of both copper semis and copper alloy semis for a quota of its production which is usually limited to few percentage points.

Amongst the EEC member countries, Belgium is the principal net exporter, while France registered a modest net import during the last few years.

With regard to the breakdown of production by types of products compared with the total amount of copper and copper alloy semis and castings, in the EEC copper semis oscillated between 65 % and 68 % in recent years, copper alloy semis remained between 24 % and 28 % and castings accounted for about 8 %.

Comparing with 1967, the percentage incidence of copper semis registered an increase due mainly to copper wire, while quotas relating to copper alloy semis and castings registered a decrease. Regarding copper semis, the prevailing part consists of wire, while as to copper alloy semis, the main sectors are made up of rods, bars, sections followed by plates, sheets, strips which altogether account for 80 %.

In respect of the average situation, there are however some discrepancies even remarkable among the various countries which are to be related to the pattern of consumption by end-use sector of each country.

In Benelux, mainly because Belgium is so important, production of copper semis is very high surpassing 75 % of the total in some years, due to a higher production of wire but also of tubes.

Production of copper alloy semis (about 20 % of total) and castings (about 4 % of total) results therefore to be more modest.

Also in France the quota of copper semis is higher than average (approx. 70 % of total) due mainly to wire.

Contrary to what registered in the above countries, in Germany F.R. the incidence of copper alloy semis, to the disadvantage of copper semis, is on the total by 3 percentage points higher than the average one, due mainly to tubes and other semifinished products.

In Italy copper semis oscillate around 60 % of the total, due to a low production of tubes and plates, sheets, strips, while copper alloy semis account for 30 % with a high incidence of rods, bars, sections (about two thirds), and castings account for approx. the remaining 10 %.

In the United Kingdom, finally, the quotas by groups of products are like the average ones, however as fas as copper semis are concerned the incidence is 2-3 percentage points lower due to a higher production of plates, sheets, strips and tubes.

PRODUCTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS

Gross weight - '000 tons

		1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
BELGIUM-LUX.	Cu semis	86.2	107.8	124.5	131.7	135.0	} 191.8	} 206.4	} 215.7	} 206.3	} 263.2
	Cu alloy semis	41.3	47.9	51.3	52.6	50.8					
	castings	4.8	5.0	5.5	5.9	5.5	5.2	4.5	5.1	4.1	5.0
	total	132.3	160.7	181.3	190.2	191.3	197.0	210.9	220.8	210.4	268.2
NETHERLANDS	Cu & Cu alloy semis	52.9	68.2	74.8	77.5	80.7	76.1	83.7	78.5	77.1	100.2
	castings	9.0	9.0	9.0	9.9	8.9	8.9	8.7	8.5	8.0	6.7
	total	61.9	77.2	83.8	87.4	89.6	85.0	92.4	87.0	85.1	106.9
BENELUX	Cu semis	} 180.4	223.9	250.6	261.8	266.5	178.2	197.7	206.4	208.7	271.8
	Cu alloy semis						89.7	92.4	87.8	74.7	91.6
	castings	13.8	14.0	14.5	15.8	14.4	14.1	13.2	13.6	12.1	11.7
	total	194.2	237.9	265.1	277.6	280.9	282.0	303.3	307.8	295.5	375.1
FRANCE	Cu semis	232.0	240.3	280.0	285.9	297.6	324.0	373.7	365.1	332.9	368.9
	Cu alloy semis	151.9	158.0	192.9	185.7	180.5	184.6	210.9	194.6	143.5	176.6
	castings	38.3	38.1	40.8	41.7	38.3	38.1	40.1	40.7	35.2	34.7
	total	422.2	436.4	513.7	513.3	516.4	546.7	624.7	600.4	511.6	580.2
GERMANY F.R.	Cu semis	401.3	434.1	508.3	520.3	501.1	502.8	556.0	544.6	489.5	573.1
	Cu alloy semis	285.1	359.9	417.9	394.1	355.6	379.9	444.6	400.2	290.8	398.7
	castings	73.2	88.8	99.2	98.9	91.2	89.1	94.7	85.2	77.3	84.4
	total	759.6	882.8	1,025.4	1,013.3	947.9	971.8	1,095.3	1,030.0	857.6	1,056.2
ITALY	Cu semis	200.0	215.0	226.0	252.0	239.0	243.0	266.0	281.0	256.0	297.0
	Cu alloy semis	155.0	158.0	164.0	181.0	166.5	201.0	219.0	231.0	173.0	228.0
	castings	54.5	58.5	65.0	68.0	63.0	64.0	62.0	61.0	56.0	67.5
	total	409.5	431.5	455.0	501.0	468.5	508.0	547.0	573.0	485.0	592.5
U.K.	Cu semis	413.9	414.8	416.2	416.8	413.2	437.5	464.7	426.8	385.9	403.4
	Cu alloy semis	279.3	314.6	332.5	303.2	266.9	270.3	302.8	271.5	205.9	226.7
	castings	68.9	73.2	74.0	72.5	72.0	68.7	72.2	71.3	67.1	65.7
	total	762.1	802.6	822.7	792.5	752.1	776.5	839.7	769.6	658.9	695.8
SCANDINAVIA	Cu semis	143.4	153.7	143.9	156.1	153.3	162.3	171.2	174.4	156.2	163.4
	Cu alloy semis	61.7	65.5	76.2	78.0	66.2	76.1	83.2	86.3	62.4	69.7
	castings	11.0	13.7	14.2	15.0	12.4	12.6	14.0	14.0	12.0	13.0
	total	216.1	232.9	234.3	249.1	231.9	251.0	268.4	274.7	230.6	246.1
AUSTRIA	Cu semis	21.3	23.0	26.4	31.6	36.9	38.6	33.0	32.6	25.4	21.2
	Cu alloy semis	11.5	12.6	15.5	17.3	15.5	16.3	16.4	16.6	11.3	13.9
	total	32.8	35.6	41.9	48.9	52.4	54.9	49.4	49.2	36.7	35.1
GREECE	Cu & Cu alloy semis					21.0	24.0	25.0	21.3	22.6	25.7
PORTUGAL	Cu semis	4.6	5.2	5.8	6.9	8.6	8.5	9.8	10.1	9.4	10.8
	Cu alloy semis	2.7	3.4	5.1	6.2	5.8	6.6	8.3	8.1	6.5	6.8
	castings	1.8	1.8	1.8	1.7	1.8	3.1	2.6	3.3	3.0	3.0
	total	9.1	10.4	12.7	14.8	16.2	18.2	20.7	21.5	18.9	20.6
SPAIN	Cu semis	37.3	52.3	69.3	64.6	78.1	104.2	118.2	120.1	110.1	114.4
	Cu alloy semis	35.0	41.2	51.1	46.8	46.7	58.3	67.2	61.8	44.0	61.0
	total	72.3	93.5	120.4	111.4	124.8	162.5	185.4	181.9	154.1	175.4
SWITZERLAND	Cu semis	37.3	36.5	39.1	47.2	44.9	45.9	43.8	47.8	42.2	31.2
	Cu alloy semis	37.9	38.6	43.7	46.3	41.6	39.5	45.2	47.0	31.4	38.0
	total	75.2	75.1	82.8	93.5	86.5	85.4	89.0	94.8	73.6	69.2
YUGOSLAVIA	Cu & Cu alloy semis	59.8	58.1	69.1	71.1	61.8	57.6	71.0	91.0	113.7	150.1
INDIA	Cu semis	20.5	24.6	2.7	3.1	3.3	4.0	4.0	3.9	2.9	2.7
	Cu alloy semis	14.1	18.0	21.3	20.3	22.4	22.9	18.8	15.3	12.4	15.4
	total	34.6	42.6	24.0	23.4	25.7	26.9	22.8	19.2	15.3	18.1
JAPAN	Cu semis	573.1	673.9	776.0	787.3	775.1	881.0	1,128.2	861.5	746.5	914.6
	Cu alloy semis	295.9	325.6	369.8	384.3	350.5	412.2	523.2	397.6	334.1	443.0
	castings	94.5	100.5	105.8	117.0	110.0	108.3	122.9	117.3	87.6	
	total	963.5	1,100.0	1,251.6	1,288.6	1,235.6	1,401.5	1,774.3	1,376.4	1,168.2	1,357.6

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PRODUCTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS

Gross weight – '000 tons

		1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
CANADA	Cu semis	139.2	145.3	149.4	133.2	127.4	126.1	194.8	207.9	170.1	191.2
	Cu alloy semis	67.3	78.1	86.0	72.0	68.8	75.8	81.7	83.1	55.7	66.7
	total	206.5	223.4	235.4	205.2	196.2	201.9	276.5	291.0	225.8	257.9
USA	Cu semis	1,518.2	1,447.5	1,677.2	1,496.8	1,559.9	1,705.9	1,945.6	1,653.0	1,269.6	1,503.9
	Cu alloy semis	727.6	806.7	878.2	699.3	729.0	859.8	934.2	823.9	572.0	720.1
	castings	362.5	358.8	386.9	340.4	319.8	346.0	353.8	304.6	232.0	249.1
	total	2,608.3	2,613.0	2,942.3	2,536.5	2,608.7	2,911.7	3,233.6	2,781.5	2,073.6	2,473.1
BRASIL	Cu semis	36.5	48.6	54.4	62.6	74.8	85.2	100.0	126.1	120.5	
	Cu alloy semis	26.5	37.8	36.4	22.4	31.2	41.8	42.7	54.2	65.5	
	castings	10.0	10.0	10.0	10.8	14.3	14.8	17.0	18.8	19.1	
	total	73.0	96.4	100.8	95.8	120.3	141.8	159.7	199.1	205.1	
MEXICO	Cu semis			53.6	49.5	45.3	49.5	58.7	57.7	53.7	
	Cu alloy semis			5.1	5.2	3.5	4.9	8.3	10.3	8.9	
	total			58.7	54.7	48.8	54.4	67.0	68.0	62.6	
CHILE	Cu & Cu alloy semis	8.3	11.5	5.7	11.3	10.2	14.8	18.2	17.3	22.3	39.8
AUSTRALIA	Cu & Cu alloy semis	107.5	113.7	99.1	98.7	102.3	92.4	105.5	106.8	93.8	

PRODUCTION AND APPARENT CONSUMPTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN SOME EEC COUNTRIES (1)

Copper content - '000 tons

	1967		1970		1973		1975		1976	
	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.
COPPER	1,247.2	1,189.9	1,475.0	1,458.6	1,660.4	1,649.6	1,464.3	1,428.2	1,642.4	1,631.1
Wire	905.0	871.9	1,080.1	1,072.5	1,200.0	1,187.3	1,068.5	1,046.4	1,204.5	1,203.8
Rods, bars, sections	55.8	51.3	76.1	76.0	75.1	62.3	68.3	73.4	68.6	69.1
Plates, sheets, strips	115.0	108.2	127.9	123.3	146.4	133.4	130.4	111.2	144.2	123.1
Tubes	170.5	157.6	189.3	185.2	236.3	264.0	195.9	196.0	223.9	233.9
Other	0.9	0.9	1.6	1.6	2.6	2.6	1.2	1.2	1.2	1.2
COPPER ALLOY	567.2	489.2	690.7	637.0	765.5	710.2	529.7	482.5	668.7	621.5
Wire	43.8	37.9	48.9	43.9	53.3	48.3	36.0	33.2	44.1	40.3
Rods, bars, sections	265.6	262.0	327.1	315.2	386.2	376.6	251.9	246.6	332.8	331.1
Plates, sheets, strips	176.4	126.4	208.3	192.6	213.9	193.8	154.1	133.8	196.0	171.2
Tubes	66.9	48.4	86.3	65.2	91.4	70.8	74.6	55.8	81.4	64.5
Other	14.5	14.5	20.1	20.1	20.7	20.7	13.1	13.1	14.4	14.4
CASTINGS	181.1	181.1	216.2	216.2	206.9	206.9	181.2	181.2	193.8	193.8
TOTAL	1,995.5	1,860.2	2,381.9	2,311.8	2,632.8	2,566.7	2,175.2	2,091.9	2,504.9	2,446.4

(1) United Kingdom, France, Italy, Germany F.R.

PRODUCTION AND APPARENT CONSUMPTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN THE EEC COUNTRIES (1)

Copper content – '000 tons

	1967		1970		1973		1975		1976	
	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.
COPPER					1,858.1	1,753.8	1,673.0	1,541.2		
Wire					1,340.7	1,259.8	1,234.6	1,133.5		
Rods, bars, sections					84.0	67.9	76.0	78.5		
Plates, sheets, strips					159.3	139.9	143.6	116.6		
Tubes					271.5	283.6	217.6	211.4		
Other					2.6	2.6	1.2	1.2		
COPPER ALLOY					825.5	760.6	578.2	528.2		
Wire					58.2	51.1	39.4	34.1		
Rods, bars, sections					407.8	393.0	268.0	260.2		
Plates, sheets, strips					246.4	220.2	181.8	159.2		
Tubes					92.4	75.6	75.9	61.6		
Other					20.7	20.7	13.1	13.1		
CASTINGS					216.9	216.9	190.4	190.4		
TOTAL					2,900.5	2,731.3	2,441.6	2,259.8		

(1) Excluding Denmark

PRODUCTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN EEC COUNTRIES

percentage

	1967	1970	1973		1975		1976	
	(1)	(1)	(1)	(2)	(1)	(2)	(1)	(2)
COPPER	62,5	61.9	63.1	64.1	67.3	68.5	65.6	
Wire	45.4	45.3	45.6	46.2	49.1	50.6	48.1	
Rods, bars, sections	2.8	3.2	2.8	2.9	3.1	3.1	2.7	
Plates, sheets, strips	5.8	5.4	5.6	5.5	6.0	5.9	5.8	
Tubes	8.5	7.9	9.0	9.4	9.0	8.9	9.0	
Others		0.1	0.1	0.1	0.1	-	-	
COPPER ALLOY	28.4	29.0	29.1	28.4	24.4	23.7	26.7	
Wire	2.2	2.1	2.0	2.0	1.7	1.6	1.8	
Rods, bars, sections	13.3	13.7	14.7	14.0	11.6	11.0	13.3	
Plates, sheets, strips	8.8	8.7	8.1	8.5	7.1	7.5	7.8	
Tubes	3.4	3.6	3.5	3.2	3.4	3.1	3.2	
Others	0.7	0.9	0.8	0.7	0.6	0.5	0.6	
CASTINGS	9.1	9.1	7.8	7.5	8.3	7.8	7.7	
TOTAL	100.0	100.0	100.0	100.0	100.0	100.0	100.0	

(1) United Kingdom, France, Italy, Germany F.R.
(2) Total EEC countries, excluding Denmark

PRODUCTION AND APPARENT CONSUMPTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN BENELUX

Copper content - '000 tons

	1967		1970		1973		1975		1976	
	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.
COPPER					197.7	104.2	208.7	113.0		
Wire					140.7	72.5	166.1	87.1		
Rods, bars, sections					8.9	5.6	7.7	5.1		
Plates, sheets, strips					12.9	6.5	13.2	5.4		
Tubes					35.2	19.6	21.7	15.4		
COPPER ALLOY					60.0	50.4	48.5	45.7		
Wire					4.9	2.8	3.4	0.9		
Rods, bars, sections					21.6	16.4	16.1	13.6		
Plates, sheets, strips					32.5	26.4	27.7	25.4		
Tubes					1.0	4.8	1.3	5.8		
CASTINGS					10.0	10.0	9.2	9.2	9.0	9.0
TOTAL					267.7	164.6	266.4	167.9		

PRODUCTION AND APPARENT CONSUMPTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN GERMANY F.R.

Copper content - '000 tons

	1967		1970		1973		1975		1976	
	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.
COPPER	401.3	378.1	520.3	531.6	556.0	566.1	489.5	478.7	573.1	543.5
Wire	309.7	291.8	398.8	406.6	406.1	412.8	357.5	348.3	419.2	395.9
Rods, bars, sections	15.9	15.3	25.6	27.1	25.1	22.9	26.6	32.4	29.5	27.1
Plates, sheets, strips	30.5	33.8	40.7	45.8	50.5	48.7	44.5	41.9	52.7	46.4
Tubes	44.3	36.3	53.6	50.5	71.7	79.1	59.7	54.9	70.5	72.9
Other	0.9	0.9	1.6	1.6	2.6	2.6	1.2	1.2	1.2	1.2
COPPER ALLOY	187.5	128.7	259.0	230.7	292.1	251.0	192.0	161.9	262.2	220.5
Wire	20.7	16.2	26.4	21.5	29.9	25.1	18.2	15.3	23.8	19.6
Rods, bars, sections	63.1	59.8	95.2	90.6	107.0	101.2	62.6	62.2	90.6	93.4
Plates, sheets, strips	51.0	15.2	66.1	63.4	74.3	64.5	53.3	40.5	76.5	55.6
Tubes	38.2	23.0	51.2	35.1	60.2	39.5	44.8	30.8	56.9	37.5
Other	14.5	14.5	20.1	20.1	20.7	20.7	13.1	13.1	14.4	14.4
CASTINGS	55.6	55.6	75.2	75.2	72.0	72.0	58.7	58.7	64.1	64.1
TOTAL	644.4	562.4	854.5	837.5	920.1	889.1	740.2	699.3	899.4	828.1

PRODUCTION AND APPARENT CONSUMPTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN FRANCE

Copper content - '000 tons

	1967		1970		1973		1975		1976	
	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.
COPPER	232.0	227.7	285.9	279.2	373.7	364.0	332.9	337.0	368.9	408.3
Wire	166.6	162.2	206.2	198.2	282.0	274.8	260.4	258.5	287.6	316.6
Rods, bars, sections	12.5	11.5	15.9	15.9	14.2	6.1	11.1	12.1	11.2	15.6
Plates, sheets, strips	14.0	13.1	16.9	14.7	20.0	18.6	14.0	15.4	17.6	19.6
Tubes	38.9	40.9	46.9	50.4	57.5	64.5	47.4	51.0	52.5	56.5
COPPER ALLOY	102.9	99.7	125.8	118.3	142.8	146.2	97.1	102.1	119.4	129.4
Wire	3.5	3.1	4.0	4.4	4.6	5.1	6.2	7.0	7.3	8.1
Rods, bars, sections	64.7	65.7	78.1	75.0	89.0	93.3	56.7	59.0	71.8	77.2
Plates, sheets, strips	29.9	25.3	36.8	32.0	43.9	39.5	28.8	28.8	36.0	36.2
Tubes	4.8	5.6	6.9	6.9	5.3	8.3	5.4	7.3	4.3	7.9
CASTINGS	31.7	31.7	34.2	34.2	32.9	32.9	28.9	28.9	28.5	28.5
TOTAL	366.6	359.1	445.9	431.7	549.4	543.1	458.9	468.0	516.8	566.2

PRODUCTION AND APPARENT CONSUMPTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN ITALY

Copper content - '000 tons

	1967		1970		1973		1975		1976	
	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.
COPPER	200.0	192.9	252.0	248.0	266.0	262.6	256.0	242.6	297.0	291.6
Wire	156.6	157.1	194.1	194.0	201.0	201.3	193.0	190.1	231.5	231.5
Rods, bars, sections	12.9	12.4	21.6	21.1	20.0	18.8	18.3	17.7	16.0	15.4
Plates, sheets, strips	13.8	11.1	16.3	14.4	17.6	15.3	20.5	14.7	22.0	16.5
Tubes	16.7	12.3	20.0	18.5	27.4	27.2	24.2	20.1	27.5	28.2
COPPER ALLOY	99.0	97.6	115.6	114.8	138.2	135.4	109.1	100.6	142.7	134.9
Wire	7.7	8.0	7.8	8.6	8.1	8.3	5.2	4.9	5.6	5.6
Rods, bars, sections	52.8	53.8	60.0	60.0	85.5	82.6	66.2	60.9	95.8	86.5
Plates, sheets, strips	28.3	26.8	35.8	33.8	31.6	31.0	28.0	25.6	31.4	31.6
Tubes	10.2	9.0	12.0	12.4	13.0	13.5	9.7	9.2	9.9	11.2
CASTINGS	41.4	41.4	51.7	51.7	47.1	47.1	42.6	42.6	51.3	51.3
TOTAL	340.4	331.9	419.3	414.5	451.3	445.1	407.7	385.8	491.0	477.8

PRODUCTION AND APPARENT CONSUMPTION OF COPPER AND COPPER ALLOY SEMIS AND CASTINGS IN THE UNITED KINGDOM

Copper content - '000 tons

	1967		1970		1973		1975		1976	
	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.	Prod.	App.cons.
COPPER	413.9	391.2	416.8	399.8	464.7	456.9	385.9	369.9	403.4	387.7
Wire	272.1	260.8	281.0	273.7	310.9	298.4	257.6	249.5	266.2	259.8
Rods, bars, sections	14.5	12.1	13.0	11.9	15.8	14.5	12.3	11.2	11.9	11.0
Plates, sheets, strips	56.7	50.2	54.0	48.4	58.3	50.8	51.4	39.2	51.9	40.6
Tubes	70.6	68.1	68.8	65.8	79.7	93.2	64.6	70.0	73.4	76.3
COPPER ALLOY	177.8	163.2	190.3	173.2	192.4	177.6	131.5	117.9	144.4	136.7
Wire	11.9	10.6	10.7	9.4	10.7	9.8	6.4	6.0	7.4	7.0
Rods, bars, sections	85.0	82.7	93.8	89.6	104.7	99.5	66.4	64.5	74.6	74.0
Plates, sheets, strips	67.2	59.1	69.6	63.4	64.1	58.8	44.0	38.9	52.1	47.8
Tubes	13.7	10.8	16.2	10.8	12.9	9.5	14.7	8.5	10.3	7.9
CASTINGS (1)	52.4	52.4	55.1	55.1	54.9	54.9	51.0	51.0	49.9	49.9
TOTAL	644.1	606.8	662.2	628.1	712.0	689.4	568.4	538.8	597.7	574.3

(1) Include miscellaneous products but exclude copper sulphate

7.2.4 Copper consumption by end-use sector

Statistical information on the end-use pattern of copper consumption is limited to the major consumers among the developed market-economy countries. Even where such information is available, it is comparable over time and by country only at a broad level of aggregation. Much of the detailed information is the product of *ad hoc* surveys and lacks uniformity in classification. Although data problems are not unique to copper —end-use information is scanty for many metals— their persistence in an important commodity such as copper merits concern.

Data shown in the following tables were obtained from the publication "Copper Trends 1970-1978" by Amalgamated Metal Trading Ltd.

Assessments made by Amalgamated Metal Trading allow to indicate the dimensions of end-use sectors and their evolution during the first half of the seventies.

In Western Europe and Japan the electrical sector accounts for about 50 % of consumption. In the United States the electrical industry is rather less important, although it has increased its share somewhat in recent years. Construction, transport and general engineering are important sectors everywhere. The use of copper in domestic goods is more extensive in the United States than in Western Europe and Japan, partly because of the more active marketing of copper in the former. In all countries, one fact is of overriding importance and lends a common pattern to consumption : its concentration on the electrical and construction industries, which together account for 60 %-70 % of total copper consumption. Indeed, the importance of these two sectors has been a feature of the copper market for more than 50 years.

End-use sectors and products alternative to copper will be analysed in the chapter regarding forecasts.

COPPER CONSUMPTION BY END-USE SECTOR IN USA, JAPAN AND WESTERN EUROPE

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	3,203.3	3,102.4	3,360.5	3,660.2	3,328.0	2,770.0
Construction	994.8	1,025.1	1,125.3	1,206.0	1,053.0	860.0
Transport	737.9	784.7	827.8	910.9	773.0	670.0
General engineering	1,086.4	1,051.4	1,179.8	1,413.7	1,207.7	955.0
Domestic goods and misc.	553.0	503.4	530.3	613.1	532.1	400.0
Total copper use	6,575.4	6,467.0	7,023.7	7,803.9	6,893.8	5,655.0
To add: net exports	80.7	129.3	100.3	215.9	293.3	320.0
stocks at US mills	424.3	385.2	415.6	317.2	411.5	352.0
Total copper consumption	7,080.4	6,981.5	7,539.6	8,337.0	7,598.6	6,327.0
percentage						
Electrical	48.7	48.0	47.8	46.9	48.3	49.0
Construction	15.1	15.8	16.0	15.4	15.3	15.2
Transport	11.2	12.1	11.8	11.7	11.2	11.8
General engineering	16.6	16.3	16.8	18.1	17.5	16.9
Domestic goods and misc.	8.4	7.8	7.6	7.9	7.7	7.1
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper consumption	92.9	92.6	93.2	93.6	90.7	89.4
To add: net exports	1.1	1.9	1.3	2.6	3.9	5.0
stocks at US mills	6.0	5.5	5.5	3.8	5.4	5.6
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

COPPER CONSUMPTION BY END-USE SECTOR IN WESTERN EUROPE

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	1,555.1	1,480.3	1,544.4	1,538.0	1,548.3	1,390.0
Construction	504.2	498.5	528.1	590.6	550.5	510.0
Transport	316.7	319.5	327.6	344.3	314.6	270.0
General engineering	524.8	478.7	523.3	610.2	559.6	485.0
Domestic goods and misc.	244.9	226.9	224.1	238.0	226.7	185.0
Total copper use	3,145.7	3,003.9	3,147.5	3,321.1	3,199.7	2,840.0
To add: net exports	112.3	154.0	143.0	258.0	297.7	260.0
Total copper consumption	3,258.0	3,157.9	3,290.5	3,579.1	3,497.4	3,100.0
percentage						
Electrical	49.4	49.3	49.1	46.3	48.4	48.9
Construction	16.0	16.6	16.8	17.8	17.2	18.0
Transport	10.1	10.6	10.4	10.4	9.8	9.5
General engineering	16.7	15.9	16.6	18.4	17.5	17.1
Domestic goods and misc.	7.8	7.6	7.1	7.1	7.1	6.5
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper consumption	96.6	95.1	95.7	92.8	91.5	91.6
To add: net export	3.4	4.9	4.3	7.2	8.5	8.4
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

COPPER CONSUMPTION BY END-USE SECTOR IN GERMANY F.R.

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	447.9	440.3	436.0	433.2	414.0	370.0
Construction	114.0	117.2	124.0	139.0	118.0	110.0
Transport	85.0	84.1	88.7	91.4	81.6	70.0
General engineering	149.9	123.5	132.1	153.5	107.0	95.0
Domestic goods and misc.	47.6	43.1	45.4	46.9	41.5	35.0
Total copper use	844.4	808.2	826.2	864.0	762.1	680.0
To add: net exports	14.6	7.9	6.2	51.5	111.3	57.5
Total copper consumption	859.0	816.1	832.4	915.5	873.4	737.5
percentage						
Electrical	53.0	54.5	52.8	50.1	54.3	54.4
Construction	13.5	14.5	15.0	16.1	15.5	16.2
Transport	10.1	10.4	10.7	10.6	10.7	10.3
General engineering	17.8	15.3	16.0	17.8	14.0	14.0
Domestic goods and misc.	5.6	5.3	5.5	5.4	5.5	5.1
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper consumption	98.3	99.0	99.3	94.4	87.3	92.2
To add: net exports	1.7	1.0	0.7	5.6	12.7	7.8
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

COPPER CONSUMPTION BY END-USE SECTOR IN FRANCE

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	205.7	203.0	212.4	240.2	237.1	220.0
Construction	87.6	86.9	92.4	105.4	98.6	90.0
Transport	47.4	48.2	50.1	55.7	51.4	46.0
General engineering	61.4	62.3	70.8	93.0	81.0	70.0
Domestic goods and misc.	32.1	33.6	36.2	38.4	36.7	30.0
Total copper use	434.2	434.0	461.9	532.7	504.8	456.0
To add: net exports	15.9	23.4	22.1	18.2	9.8	10.0
Total copper consumption	450.1	457.4	484.0	550.9	514.6	466.0
percentage						
Electrical	47.4	46.8	46.0	45.1	47.0	48.2
Construction	20.2	20.0	20.0	19.8	19.5	19.7
Transport	10.9	11.1	10.9	10.4	10.2	10.1
General engineering	14.1	14.4	15.3	17.5	16.0	15.4
Domestic goods and misc.	7.4	7.7	7.8	7.2	7.3	6.6
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper consumption	96.5	94.9	95.4	96.7	98.1	97.9
To add: net exports	3.5	5.1	4.6	3.3	1.9	2.1
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

COPPER CONSUMPTION BY END-USE SECTOR IN ITALY

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	185.5	161.0	170.0	188.6	201.2	180.0
Construction	74.6	66.3	70.0	76.2	78.9	75.0
Transport	49.3	49.6	51.2	54.1	51.6	40.0
General engineering	69.3	65.8	73.8	83.4	89.5	80.0
Domestic goods and misc.	50.2	48.2	51.8	54.9	52.1	40.0
Total copper use	428.9	390.9	416.8	457.2	473.3	415.0
To add: net exports	2.6	14.1	12.8	3.4	29.7	6.5
Total copper consumption	431.5	405.0	429.6	460.6	503.0	421.5
percentage						
Electrical	43.2	41.2	40.8	41.3	42.5	43.4
Construction	17.4	17.0	16.8	16.7	16.7	18.1
Transport	11.5	12.7	12.3	11.8	10.9	9.6
General engineering	16.2	16.8	17.7	18.2	18.9	19.3
Domestic goods and misc.	11.7	12.3	12.4	12.0	11.0	9.6
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper cnsumption	99.4	96.5	97.0	99.3	94.1	98.5
To add: net exports	0.6	3.5	3.0	0.7	5.9	1.5
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

COPPER CONSUMPTION BY END-USE SECTOR IN THE UNITED KINGDOM

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	254.0	228.0	258.0	254.0	248.0	220.0
Construction	102.0	98.1	108.6	123.0	105.0	100.0
Transport	67.0	68.4	66.2	69.1	63.0	50.0
General engineering	121.2	117.7	131.0	149.9	140.7	125.0
Domestic goods and misc.	57.0	47.0	34.0	38.8	36.0	30.0
Total copper use	601.2	559.2	597.8	634.8	592.7	525.0
To add: net exports	81.6	89.5	73.4	85.2	69.5	48.0
Total copper consumption	682.8	648.7	671.2	720.0	662.2	573.0
percentage						
Electrical	42.2	40.8	43.1	40.0	41.9	41.9
Construction	17.0	17.5	18.2	19.4	17.7	19.1
Transport	11.1	12.2	11.1	10.9	10.6	9.5
General engineering	20.2	21.1	21.9	23.6	23.7	23.8
Domestic goods and misc.	9.5	8.4	5.7	6.1	6.1	5.7
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper consumption	88.0	86.2	89.1	88.2	89.5	91.6
To add: net exports	12.0	13.8	10.9	11.8	10.5	8.4
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

COPPER CONSUMPTION BY END-USE SECTOR IN JAPAN

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	588.2	562.1	646.1	802.2	629.4	580.0
Construction	94.6	96.6	112.2	147.4	107.0	80.0
Transport	201.2	205.2	216.2	242.6	206.1	230.0
General engineering	176.6	159.7	186.6	276.5	180.6	140.0
Domestic goods and misc.	71.3	68.5	74.2	96.2	82.3	65.0
Total copper use	1,131.9	1,092.1	1,235.3	1,564.9	1,205.4	1,095.0
To add: net exports	33.2	49.1	51.2	34.4	41.1	85.0
Total copper consumption	1,165.1	1,141.2	1,286.5	1,599.3	1,246.5	1,180.0
percentage						
Electrical	52.0	51.5	52.3	51.3	52.2	53.0
Construction	8.3	8.8	9.1	9.4	8.9	7.3
Transport	17.8	18.8	17.5	15.5	17.1	21.0
General engineering	15.6	14.6	15.1	17.7	15.0	12.8
Domestic goods and misc.	6.3	6.3	6.0	6.1	6.8	5.9
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper consumption	97.2	95.7	96.0	97.8	96.7	92.8
To add: net exports	2.8	4.3	4.0	2.2	3.3	7.2
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

COPPER CONSUMPTION BY END-USE SECTOR IN USA

	1970	1971	1972	1973	1974	1975
'000 tons						
Electrical	1,060.0	1,060.0	1,170.0	1,320.0	1,150.3	800.0
Construction	396.0	430.0	485.0	468.0	395.5	270.0
Transport	220.0	260.0	284.0	324.0	252.3	170.0
General engineering	385.0	413.0	470.0	527.0	467.5	330.0
Domestic goods and misc.	236.8	208.0	232.0	279.0	223.1	150.0
Total copper use	2,297.8	2,371.0	2,641.0	2,918.0	2,488.7	1,720.0
To add: stocks at mills etc	424.3	385.2	415.6	317.2	411.7	352.0
To deduct: net imports	64.8	73.8	93.9	76.5	45.5	25.0
Total copper consumption	2,657.3	2,682.4	2,962.7	3,158.7	2,854.9	2,047.0
percentage						
Electrical	46.1	44.7	44.3	45.2	46.2	46.5
Construction	17.2	18.1	18.4	16.0	15.9	15.7
Transport	9.6	11.0	10.7	11.1	10.1	9.9
General engineering	16.8	17.4	17.8	18.1	18.8	19.2
Domestic goods and misc.	10.3	8.8	8.8	9.6	9.0	8.7
Total copper use	100.0	100.0	100.0	100.0	100.0	100.0
Total copper use as %age of total copper consumtion	86.5	88.4	89.2	92.4	87.2	84.0
To add: stocks at mills etc	15.9	14.4	14.0	10.0	14.4	17.2
To deduct: net imports	2.4	2.8	3.2	2.4	1.6	1.2
Total copper consumption	100.0	100.0	100.0	100.0	100.0	100.0

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8. INTERNATIONAL TRADE

8 INTERNATIONAL TRADE

8.1 Copper prices

8.1.1 Metal prices

1 - Copper free market prices are fixed on the London Metal Exchange. Prices in Europe, Japan, Australia and South Africa are linked on various bases to LME quotations.

Practically the whole of copper production which is treated internationally is sold at prices based on LME.

In the USA also copper free market prices are fixed : export price ; merchant price ; Comex price ; but the most of the copper consumed is negotiated at a "producer" price established by the major companies. The Canadian "producer" price usually follows the US "producer" price considering the closeness of these two markets. Copper prices are characterised by a high instability, especially the prices established by LME. Stability for copper does not appear to be obtainable as long as the metal is quoted on a commodity exchange and the price of so much of its production is related to ruling prices on such an exchange. On the other hand it must be acknowledged that attempts to force the industry away from commodity exchanges have so far failed. For example a producer price similar to that operated in the United States has proved to be impracticable in the international market despite numerous attempts to adapt such a system. Such attempts —and other efforts which have been made from time to time to influence the international market in different ways— must be considered in the context in which they occurred.

In the United States it is possible to obtain a certain stability for the producer price because of the structure of the market : almost of equilibrium between production and consumption of refined copper, advanced integration of the copper industry ; this price changes periodically as supply and demand conditions dictate —but occasionally an upper limit is imposed by the US government.

Variations in these prices do not follow the erratic fluctuations of the LME quotations, where also speculative factors occur, but they reflect them in an alternative way.

2 - The main reasons for price instability and the difficulty to introduce a system for price regulating consist in the lack of flexibility on the part of the copper industry due, especially, to investments being so heavy during the ore mining and concentration phase, which makes supply difficult to adjust to demand. Price instability is the result of a number of decisions taken independently by producers and governments concerning level of production and investments, as they see fit under prevailing economic circumstances. Price instability can have harmful effects on the copper industry : it encourages substitution by other materials.

Copper prices fluctuate far more than prices of substitutes such as aluminium and plastics. This poses problems for fabricating companies and large consumers, who look for more stable alternatives such as aluminium.

Replacement by aluminium is of extreme importance for copper considering that, because of the characteristics proper of aluminium and the results achieved by research studies, aluminium can already to-day potentially substitute 60 % of copper in the electrical sector, which accounts for over 60 % of copper total consumption. It discourages the advancement of technology.

It can have an adverse effect on costs of production.

It discourages investment in exploration and new mines.

Price fluctuations certainly affect the timing of new projects but rarely prevent them. The instability has contributed to prevent the creation of new mine capacity as needed. The margin of availability spare capacity has at time been inadequate to cope with peak demand.

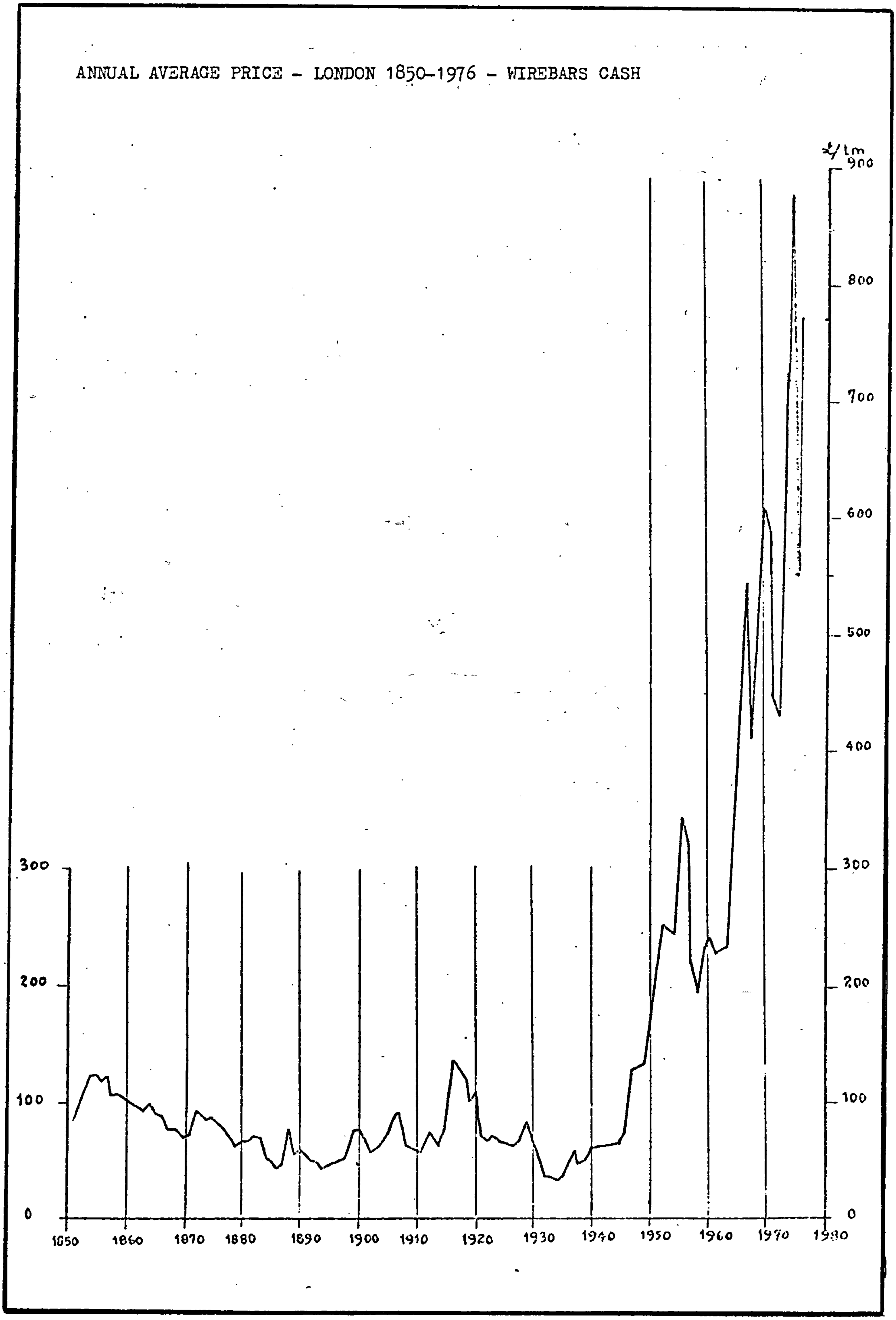

ANNUAL AVERAGE PRICE - LONDON 1850-1976 - WIREBARS CASH
£/tm
900
800
700
600
500
400
300
200
100
0
300
200
100
0
1850
1860
1870
1880
1890
1900
1910
1920
1930
1940
1950
1960
1970
1980

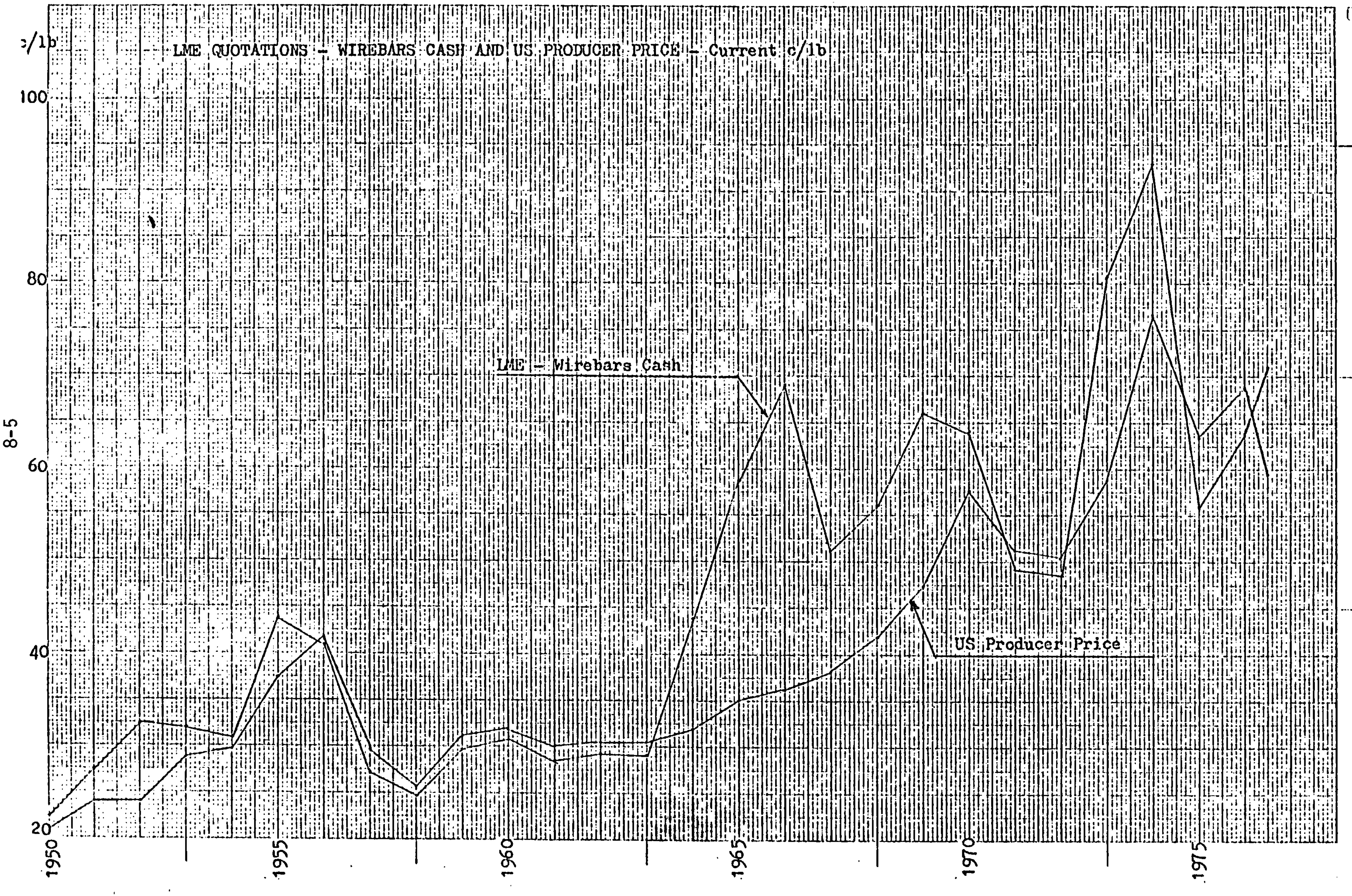
Table 25 Copper Prices 1950–1976

	1950	1951	1952	1953	1954	1955	1956	1957	1958	1959	1960	1961	1962	1963
London Metal Exchange[1] (£ per metric ton) Refined Copper – Wirebars														
Cash Settlement Average	175·98	216·88	255·25	265·08	244·66	315·87	323·73	216·02	194·27	234·00	242·07	225·94	230·29	230·57
High	198·81	230·30	282·47	280·50	305·10	398·60	430·10	267·80	256·88	262·88	274·69	245·07	233·26	232·27
Low	150·58	198·81	223·41	248·02	211·60	284·44	257·86	173·22	157·47	206·19	213·72	213·08	223·56	227·35
Three Months Price	NQ	NQ	NQ	206·03	236·09	335·62	319·52	217·57	194·09	231·20	235·04	227·63	228·84	230·85
LME Cash Average (U.S. cents per lb.)	22·35	27·55	32·40	32·03	31·06	43·72	40·99	27·26	24·73	29·73	30·75	28·59	29·28	29·32
United States Prices[2] (U.S. cents per lb.) Refined Copper – Wirebars														
Producer Price Average	21·24	24·20	24·20	28·80	29·69	37·49	41·82	29·58	25·76	31·18	32·05	29·92	30·60	30·60
High	24·20	24·20	24·20	30·75	29·70	45·30	47·00	35·60	28·83	35·18	34·18	30·68	30·60	30·60
Low	18·20	24·20	24·20	24·20	29·60	29·70	35·37	25·43	23·53	28·60	29·60	28·60	30·60	30·60
Yellow Brass Ingot	19·25	24·68	23·25	21·77	22·96	31·28	30·63	24·23	22·32	24·35	23·94	20·23	27·50	27·50
No. 2 Heavy Copper Scrap Dealers Buying Price[2 3]	17·67	21·33	19·00	22·43	24·54	33·62	31·57	20·09	17·58	22·55	21·16	21·78	21·58	22·19
Refinery Buying Price	...	...	...	..	...	...	...	...	...	...	24·55	25·04	24·92	25·13
Copper Wire Price	25·82	29·49	30·79	35·85	35·48	43·05	47·37	32·53	32·29	37·08	38·32	36·30	37·07	37·08
Prices Adjusted for Inflation in 1975 Values (U.S. cents per lb.)														
LME Cash Average	47·8	52·9	63·9	64·1	62·0	87·0	79·0	51·1	45·7	64·8	56·6	52·9	54·0	54·2
U.S. Producer Average	45·4	46·4	47·8	57·6	59·3	74·6	80·6	55·4	47·6	57·5	59·0	55·3	56·4	55·6

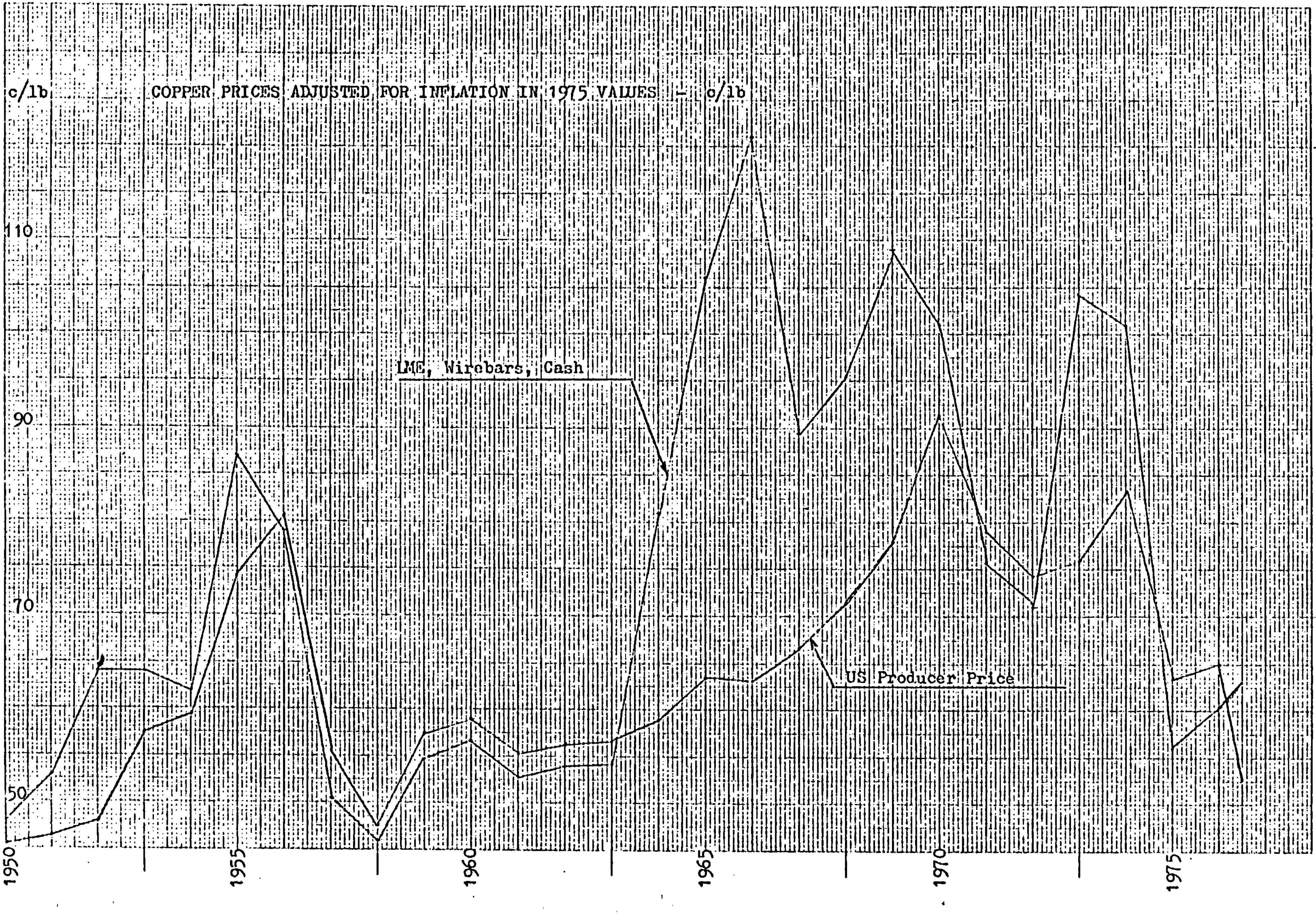
1964	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976	
													London Metal Exchange[1] (£ per metric ton) Refined Copper – Wirebars
345·52	462·46	545·71	410·75	515·70	611·44	587·90	441·43	427·96	726·82	877·00	556·81	782·40	Cash Settlement Average
522·51	561·00	777·52	597·90	804·59	734·72	749·00	536·50	451·00	1 135·00	1 400·00	626·00	936·50	High
231·29	325·77	350·38	341·52	425·67	501·94	422·00	394·00	407·00	447·00	529·00	498·00	576·50	Low
325·60	423·88	529·91	399·20	487·37	590·13	588·27	453·27	436·81	701·57	865·01	576·35	810·72	Three Months Price
43·86	58·73	69·13	51·24	56·01	66·29	63·89	49·27	48·55	80·81	93·10	56·11	53.56	LME Cash Average (U.S. cents per lb.)
													United States Prices[2] (U.S. cents per lb.) Refined Copper – Wirebars
31·96	35·02	36·17	38·23	41·85	47·53	57·71	51·43	50·62	58·87	76·65	63·54	68·82	Producer Price Average
33·76	37·05	47·18	39·58	42·34	52·91	59·70	52·60	51·95	68·81	85·97	68·70		High
30·60	33·60	35·60	36·21	41·65	41·69	52·60	49·70	49·70	49·99	67·95	61·65		Low
29·65	33·49	40·28	35·24	35·62	44·88	46·51	44·08	42·19	56·28	72·64	56·44		Yellow Brass Ingot
25·98	34·49	44·66	33·15	32·76	42·88	39·45	27·57	39·02	50·22	54·88	33·94		No. 2 Heavy Copper Scrap Dealers Buying Price[2 3]
30·53	38·91	50·09	37·28	38·73	49·13	49·19	38·43	31·38	60·20	67·75	41·12		Refinery Buying Price
38·66	43·71	48·58	52·98	57·35	69·13	83·16	73·99	73·52	81·83	98·91	05·46	..	Copper Wire Price
													Prices Adjusted for Inflation in 1975 Values (U.S. cents per lb.)
81·0	106·3	121·1	89·6	95·6	108·8	101·2	75·7	71·2	104·3	101·1	56·1	60.4	LME Cash Average
59·0	63·4	63·2	66·8	71·4	78·0	91·4	79·0	74·3	76·0	83·3	63·5	65.0	U.S. Producer Average

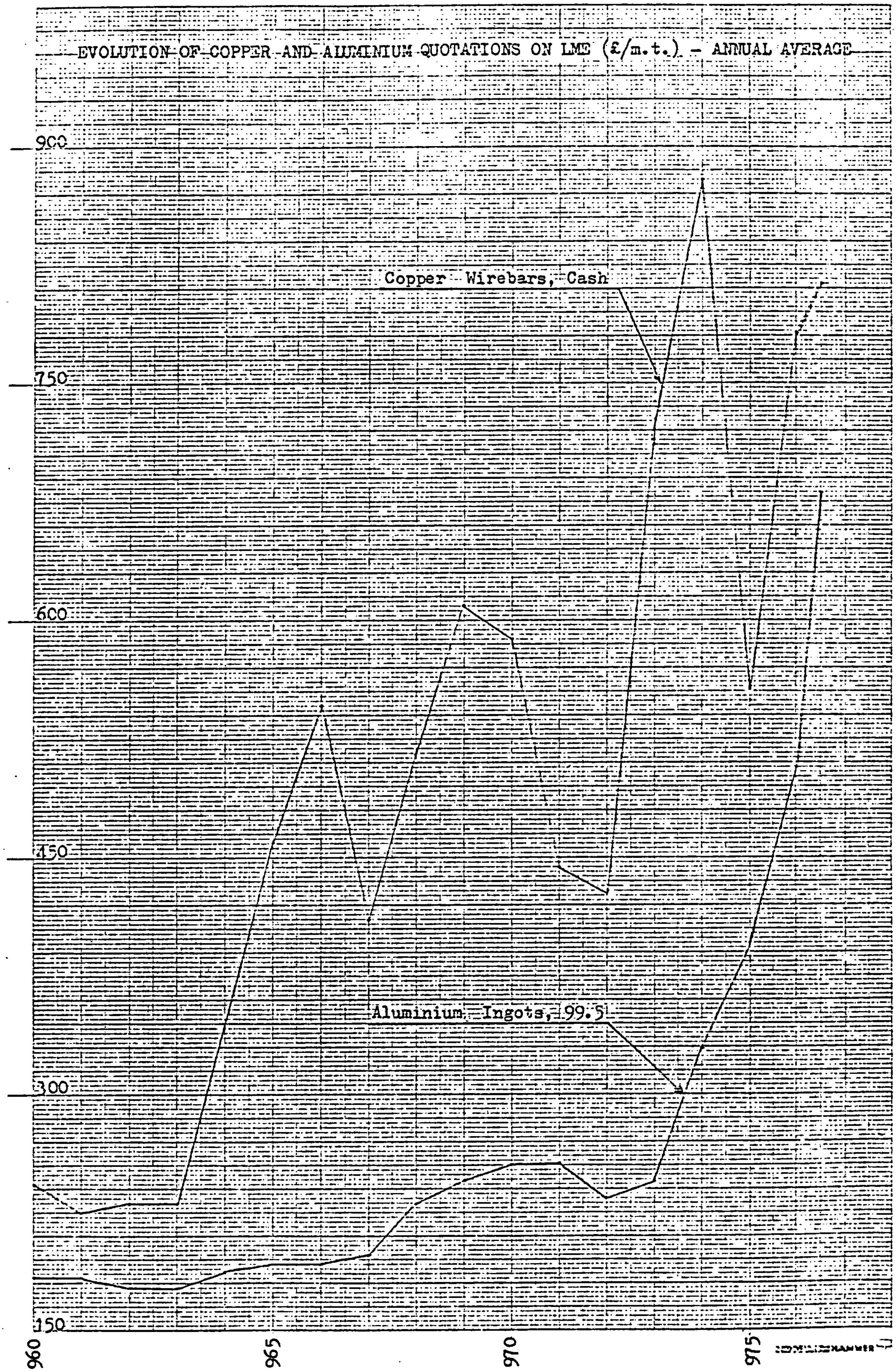
Notes: [1] Until August 4th 1953 selling price of U.K. Ministry of Supply
[2] Source Metal Statistics published by American Metal Market
[3] 1950–1955 – No. 1 Copper Scrap Prices quoted

Source: World Copper Statistics Since 1950 - Published by World Bureau of Metal Statistics


c/lb
LME QUOTATIONS - WIREBARS CASH AND US PRODUCER PRICE - Current c/lb
100
80
60
40
20
LME - Wirebars Cash
US Producer Price
1950
1955
1960
1965
1970
1975

8-5


c/lb
COPPER PRICES ADJUSTED FOR INFLATION IN 1975 VALUES - c/lb
LME, Wirebars, Cash
US Producer Price
110
90
70
50
1950
1955
1960
1965
1970
1975

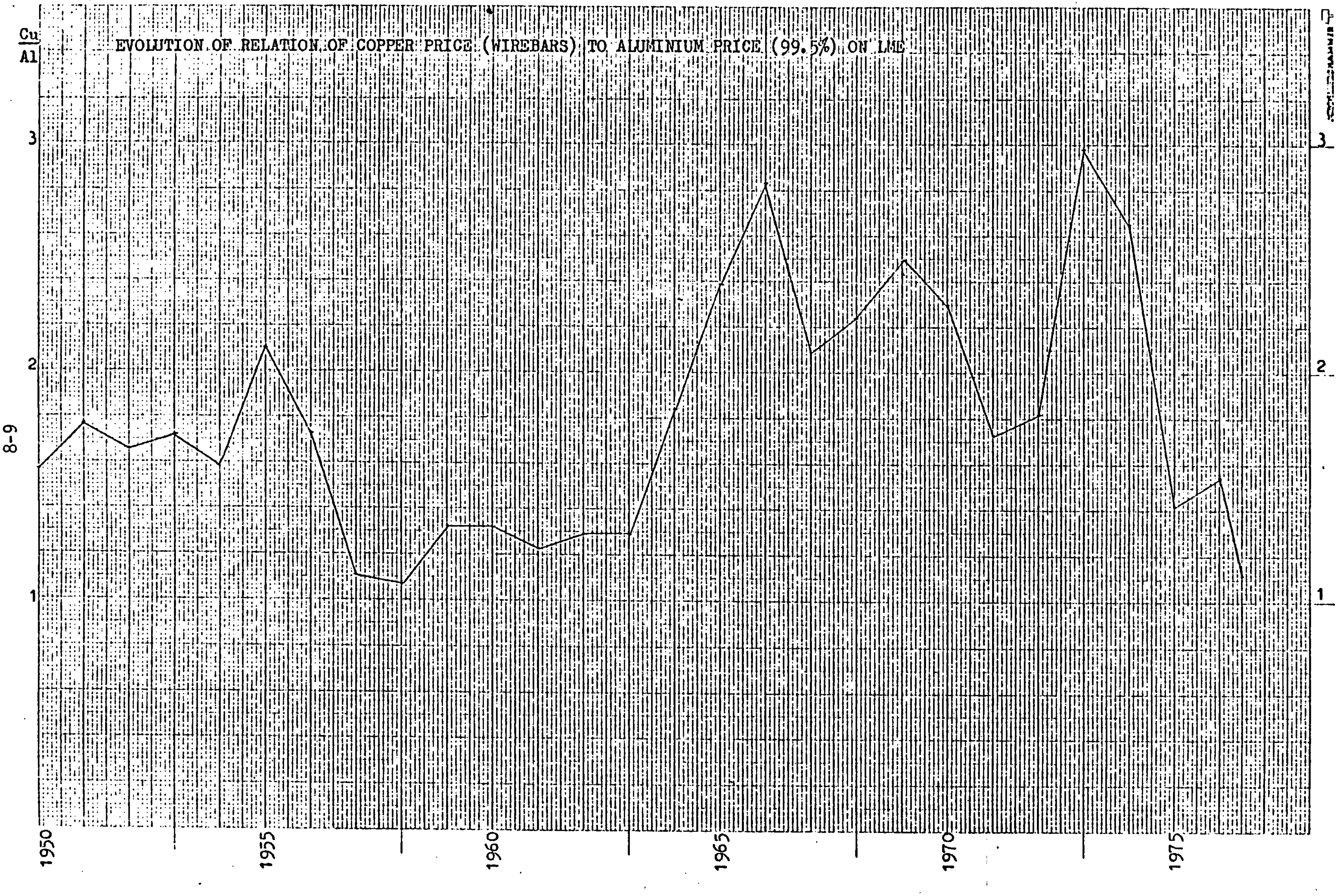

EVOLUTION OF COPPER AND ALUMINIUM QUOTATIONS ON LME (£/m.t.) - ANNUAL AVERAGE
900
750
600
450
300
150
Copper Wirebars, Cash
Aluminium Ingots, 99.5
960
965
970
975

EVOLUTION OF COPPER AND ALUMINIUM QUOTATIONS FROM 1946
LONDON METAL EXCHANGE

Annual average - £/long ton (1)

	Electrolytic Copper (wirebars)(2)	Aluminium 99.5% (ingots)(3)	Ratio Cu Price / Al Price
1946	77.17	73.74	1.05
1947	130.56	78.34	1.66
1948	134.00	81.75	1.64
1949	133.04	95.69	1.39
1950	178.80	113.85	1.57
1951	220.36	124.00	1.77
1952	259.35	155.81	1.66
1953	269.33	156.75	1.72
1954	248.59	156.00	1.59
1955	351.42	167.00	2.10
1956	328.92	190.34	1.73
1957	219.49	197.00	1.11
1958	197.38	184.25	1.07
1959	237.76	180.25	1.32
1960	245.96	186.00	1.32
1961	229.57	186.00	1.23
1962	233.99	180.70	1.29
1963	234.27	180.99	1.29
1964	351.07	190.88	1.84
1965	468.07	196.00	2.39
1966	555.00	196.00	2.83
1967	418.04	199.74	2.09
1968	525.30	234.11	2.24
1969	620.75	249.10	2.49
1970	589.45	255.86	2.30
1971	444.36	257.20	1.73
1972	427.82	235.28	1.82
1973	727.10	244.00	2.98
1974	876.16	329.34	2.66
1975	556.51	392.20	1.42
1976	782.09	504.13	1.55

(1) From 1970, £/m.t.
(2) Until 4 August 1953, Ministry of Supply selling price, cif delivery price. From August 1953, LME official quotation, cash.
(3) Until 30 June 1953, Ministry of Supply selling price.
Source: Metallgesellschaft


Cu/Al
EVOLUTION OF RELATION OF COPPER PRICE (WIREBARS) TO ALUMINIUM PRICE (99.5%) ON LME
3
2
1
1950
1955
1960
1965
1970
1975

Price instability caused by speculation limits the effectiveness of the price mechanism as a method of allocating resources. It has a relevant importance to the problems of the less developed countries.

Copper highly contributes to the economies of a limited number of developing countries. As weak prices coincide with low demand, these countries are unduly affected by changes in world economic activity.

These considerations justify the interest and the pressure from the developing countries, through CIPEC to introduce measures for the copper market regulating.

3 - Stability in copper price quotations represents a necessity for the relations between producing countries and industrialized countries. The solution of this problem has however not yet received that character of priority it deserves because, although the share of production accounted for by the developing countries bears such a weight and is destined to become even more important in the future, the majority of production, considering also the amount of scrap, is accounted for by the industrialized countries.

There are some basic considerations to be taken into account concerning proposals of schemes or measures for price regulating :

- it is considered to be outside the political reality to rely on market forces unimpeded by any schemes ;
- a producer price system appears impracticable in to-day's political conditions ; the fabricators and final consumers are resolutely opposed to any dual pricing system for copper ;
- necessity of improvements in statistics on production, consumption, trade and stocks and comparison on long term capacity estimates ;
- a combination of the various schemes is possibly the correct approach, blending aspects of all ;
- the necessity to ensure earnings support for less developed countries.

Essentially the measures proposed fall within the following types :

A - Improvement in market mechanisms : changes in pricing terms and other contract conditions and income support for less developed countries with the aim to act as a buffer between variable copper industries receipts and steady government spending programmes.

B - International agreements

B_1 An International Copper Agreement along the lines proposed by the International Wrought Copper Council, which in turn are closely modelled on the International Tin Agreement, which is based on combined operations of a buffer stock and production of export controls.

A variant might be based on an understanding that voluntary restraint might accompany the operations of a buffer stock.

B_2 A large international buffer stock would alone be sufficient to minimise price fluctuations. The econometric studies have indicated that a support based on a five year moving average with a plus or minus 15 % fluctuation would be manageable.

One of the main studies acknowledges that production cuts may still be essential if the buffer stock gets things wrong and builds up an unmanageable excess.

B_3 National stockpiles are an alternative approach. The point about such stocks is that they would be more securely held than privately financed material and therefore less liable to influence prices. Their existence would also reduce the need for large private stockpiles.

B_4 An alternative approach is to build a scheme around the price mechanism included in many long term sales contracts.

It might be necessary for major producing and consuming companies to discuss and agree on a scheme on these lines.

In Geneva at the beginning of November, within UNCTAD, talks were started on the possibility to create a common fund for financing mechanisms for the stability of raw materials. The 98 delegations participating in these works were agreed upon the necessity to avoid erratic fluctuations in raw materials quotations in the interest of both producers and consumers.

The common fund should therefore finance a system of stockpiles destined to regulate markets wavings through the creation —according to the opinion of the nations of the Third World— of a real "raw materials bank". On the contrary, for the industrialized countries, the common fund should be just a sort of "compensation fund" by means of which the products temporarily on the rise would pay for the products for which demand was at the moment downward.

Compared to the first one, the second proposal has the advantage of being more rapidly set into operation : as a matter of fact it should not be difficult to find the investments required amongst the countries interested in stabilizing prices and checking inflation.

5 - Owing to a slowing of world economic growth and continued substitution particularly by aluminium, world consumption of refined copper, including scrap, is likely to grow at less than the average trend rate of the past two decades.

According to "The outlook for the world copper industry 1975-1985" by CRU Press Release Immediate - 24 August 1976 - CRU research has revealed that capital spending in the crucial electrical generation and transmission sector of the market will be very low throughout the industrialized world over the next four-to-five years. This factor, plus the expectation that the period of spectacular growth in the Japanese economy has passed, has led CRU to forecast a copper consumption growth trend of only 2.5 % per annum through 1980. 1972 was used as the base year for the forecast period since it was the last "normal" year before the boom-bust cycle of 1973-75.

However, CRU anticipates a stronger growth rate in the 1981-85 period once the slack in the electrical generating system has been absorbed and new capital investment projects have been launched.

Unfortunately for the world's copper producers, they are entering this low-growth period with a considerable amount of overcapacity, plus large inventories.

World Stocks of Unwrought Copper

AND METRIC TONS

	1972	1973	1974	1975	1976	1977 May	June	July	Aug
BLISTER[1]									
Australia	3·0	3·2	6·5	9·7	6·0	6·3	8·6		
Canada	4·3	8·3	10·8	12·0	11·7	28·2	21·5	21·4	
S. & S.W. Africa	5·8	6·2	7·1	5·5	4·3	0·6[6]			
U.S.A.	255·8	240·4	293·9	233·0	291·2	254·9	232·2		
Zambia	40·5	43·5	42·8	42·1	33·3	37·9	30·6	69·8	
Total Blister Stocks	309·4	301·6	361·1	352·3	346·5				
REFINED COPPER									
METAL EXCHANGE STOCKS									
Comex	52·4	5·3	39·2	90·7	182·3	191·2	191·4	186·3	182·3
London Metals Exchange									
Wirebars	121·4	27·2	99·4	403·5	503·0	512·7	503·0	503·2	502·8
Cathodes	61·6	7·6	26·5	93·5	100·5	93·8	95·3	102·3	106·6
Total	183·0	34·8	125·9	497·0	603·5	606·5	599·3	605·5	609·4
of which:									
Belgium	2·2	1·2	2·4	31·3	43·7	44·2	39·1	39·8	41·4
Germany	58·7	11·0	36·3	159·9	232·3	251·1	249·9	255·4	262·2
Netherlands	75·8	11·3	76·6	232·2	233·6	222·8	221·4	219·3	216·1
United Kingdom	46·3	11·3	10·6	73·6	93·9	83·4	88·9	89·5	89·7
Total Metal Exchange Stocks	235·4	40·1	165·1	587·7	785·8	797·7	790·7	789·5	791·7
Country Series									
Australia[2]	5·6	5·2	10·0	12·1	6·2	9·2	6·2		
Canada[2]	20·6	9·4	27·2	56·1	34·8	23·8	30·3	37·3	45·2
France[3]	48·7	52·0	64·3	57·3	53·0	56·5	57·0	59·5	61·0
Germany, Federal Republic									
Producers	22·8	15·8	27·5	39·4	32·4		44·1		
Merchants	6·2	5·2	5·0	7·7	7·6		8·2		
Consumers	53·8	36·2	50·8	61·4	64·8		57·6		
Total	82·8	57·2	83·3	108·5	104·8		109·9		
Japan									
Producers	27·1	53·1	106·7	187·8	156·1	194·0	198·8		
Merchants	16·0	15·8	24·0	48·0	50·6	50·0	51·8		
Consumers	22·8	33·5	41·1	62·4	50·4	75·6	73·6		
Total	65·9	102·4	171·8	298·2	257·1	319·6	324·2		
S. & S.W. Africa[2]	2·8	1·2	5·8	1·5	4·4	4·8	7·9		
United Kingdom[3]									
Wirebars	19·3	5·6	10·9	6·6	7·1	5·1	6·6	4·2	
Other Shapes	10·4	9·8	15·3	13·8	15·7	17·2	21·1	19·4	
Total	29·7	15·4	26·2	20·4	22·8	22·3	27·7	23·6	
United States									
Producers[5]	90·3	39·2	137·6	236·5	247·5	214·8	203·2	164·6	153·9
Wire Mills	44·9	38·5	97·9	108·1	103·8	122·1			
Brass Mills	25·1	27·2	33·1	28·1	32·1	54·4			
Total	160·3	104·9	268·6	372·7	383·4	391·3			
Zambia[2]	5·0	5·8	9·7	21·6	19·6	10·9	11·8	9·0	
Total – Country Series	421·4	353·5	666·9	948·4	885·1				
of which:									
Producers	174·2	129·7	324·5	555·0	501·0				
Merchants	22·2	21·0	29·0	55·7	58·2				
Consumers	225·0	202·8	313·4	337·7	325·9				
Total Commercial Stocks of Refined Copper	656·8	393·6	832·0	1 536·1	1 671·9				
Strategic Stockpiles:									
United States	228·2	228·2	32·0	23·7	38·2	26·7[6]			
Japan	–	–	–	–	50·4	50·4	50·4	50·4	50·4
PRODUCER STOCKS[4] (as reported by A.B.M.S.)									
U.S.A.[5]	90·3	39·2	137·6	236·5	247·5	214·8	203·2	164·6	153·9
Outside U.S.A.	189·2	281·5	403·6	525·8	409·6	449·3	463·5	471·9	
Total	279·5	320·7	541·2	702·3	657·1	664·1	666·7	636·5	

This table summarises all available information on stocks of unwrought copper. Details of stocks of semi-manufactures are only available for the U.S.A. and will be found on page 65. All figures indicate stocks at the end of period.

Notes (1) Including anodes (2) At producers (3) At consumers
(4) Includes stocks at refineries and in transit. New series for "Outside U.S.A." reported from 1973
(5) This does not include Comex (6) As at end of March

Source: World Metal Statistics October 1977 - Published by World Bureau of Metal Statistics

Table 9 World Stocks of Refined Copper at Year End 1950–1976

Thousands Metric Tons

	1950	1951	1952	1953	1954	1955	1956	1957	1958	1959	1960	1961	1962	1963
Metal Exchange Stocks														
L.M.E.	–	–	–	2·1	1·1	3·7	4·3	20·6	5·6	5·5	14·6	17·4	13·1	14·4
Comex	0·1	–	–	0·1	0·4	0·3	0·8	0·7	9·5	11·7	2·5	7·8	3·6	1·5
Total Metal Exchange Stocks	0·1	–	–	2·2	1·6	4·0	5·1	21·3	15·1	17·2	17·1	25·2	16·7	15·9
Country Series														
Australia[1]	4·1	3·2	7·8	4·4	1·8	3·1	2·5	1·4	1·4	1·3	4·7	1·8	5·3	4·1
Canada[1]	7·2	10·9	9·0	8·3	7·2	6·0	11·6	20·3	10·1	14·0	21·3	30·6	35·4	22·2
France[2]	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Germany														
Producers	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Merchants	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Consumers	..	..	..	..	..	..	..	..	..	..	..	..	..	..
Total	27·0	27·1	22·5	18·0	23·7	28·6	30·3	35·9	39·8	29·8	53·7	60·3	51·9	44·4
Japan														
Producers			3·2	0·4	3·8	0·9	1·2	11·5	5·0	1·6	2·5	5·1	15·2	3·2
Merchants } Consumers			6·7	5·7	9·5	6·1	8·6	14·3	13·3	7·3	8·9	14·0	0·8	13·1
Total	6·4	7·5	9·9	6·1	13·3	7·0	8·8	25·8	18·3	8·9	12·4	19·1	25·0	16·3
Peru[1]	..	..	..	..	..	..	..	..	0·8	0·5	1·4	3·2	2·4	3·0
S. & S.W. Africa[1]	0·3	0·2	0·7	0·1	0·8	0·3	3·2	1·5	1·9	0·7	0·3	0·8	0·2	1·4
U.K.[2]														
Wirebars	7·3	5·2	1·8	7·2	7·7	12·9	10·6	15·8	11·5	6·5	16·2	18·9	21·2	14·5
Other	21·6	24·5	18·2	13·8	15·4	20·2	15·6	12·9	13·3	16·4	17·8	16·3	15·2	12·4
Total	28·9	29·7	20·0	21·0	23·1	33·1	26·2	28·7	24·8	22·9	34·0	35·2	36·4	26·9
U.S.A.														
Producers	44·4	64·9	53·4	80·8	42·3	55·5	108·6	163·5	63·7	47·1	123·8	64·6	102·9	68·3
Wire Mills } Brass Mill	..	..	..	..	..	57·1	82·8	93·5	80·4	45·2	61·0	60·2	64·7	44·5
Total	44·4	64·9	53·4	80·8	42·3	112·6	191·4	257·0	144·1	92·3	184·8	124·8	167·6	112·8
Zambia[1]	2·6	2·9	3·9	1·8	3·1	3·6	4·6	9·8	9·0	12·8	19·4	13·6	15·2	17·3
Total Country Series	120·9	146·4	127·2	140·5	115·4	194·3	279·6	380·4	250·2	183·2	332·0	289·2	339·4	248·4
Total Commercial Stocks	121·0	146·4	127·2	142·7	116·9	198·3	284·7	401·7	265·3	200·4	349·1	314·4	356·1	264·3
U.S. Strategic Stockpile	482·3	543·3	501·7	600·0	780·8	802·2	845·4	817·5	1 030·7	1 034·7	1 010·2	1 035·6	1 028·1	1 018·6

	1964	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Metal Exchange Stocks													
L.M.E.	5·6	7·6	14·3	11·5	19·4	18·6	72·2	140·3	183·1	34·8	125·9	497·0	603·5
Comex	2·8	8·4	3·7	12·5	10·6	3·6	16·2	18·4	52·4	5·3	39·2	90·7	182·3
Total Metal Exchange Stocks	8·4	17·0	18·0	24·0	30·0	22·2	80·4	158·7	235·5	40·1	165·1	587·7	785·8
Country Series													
Australia[1]	2·4	2·6	2·3	4·2	6·7	6·0	8·1	3·0	5·6	5·2	10·0	12·1	6·2
Canada[1]	9·9	21·2	11·6	11·4	11·1	11·8	16·8	22·8	20·6	9·4	27·2	56·1	34·8
France[2]	..	..	..	46·9	33·5	52·7	62·8	46·6	48·7	52·0	64·3	57·3	53·0
Germany													
Producers	..	11·7	15·7	11·7	12·3	12·8	15·0	17·3	22·8	15·8	27·5	39·4	32·4
Merchants	..	3·3	4·6	4·6	3·8	4·7	7·0	6·1	6·2	5·2	5·0	7·7	7·6
Consumers	..	39·5	47·5	30·7	38·5	37·3	47·3	50·2	53·8	36·2	50·8	61·4	64·8
Total	50·2	54·5	67·8	[illegible]	54·6	54·8	69·3	73·6	82·8	57·2	83·3	108·5	104·8
Japan													
Producers	2·6	5·0	3·7	6·4	6·4	17·9	22·5	49·8	27·1	53·1	106·7	187·8	156·1
Merchants	12·0	13·5	1·4	3·4	3·0	5·7	4·1	5·1	16·0	15·8	24·0	48·0	50·6
Consumers			12·7	13·1	14·7	15·5	11·8	20·1	22·8	33·5	41·1	62·4	50·4
Total	14·6	18·5	17·8	22·9	24·1	39·1	38·4	75·0	65·9	102·4	171·8	298·2	257·1
Peru[1]	0·9	1·0	0·6	0·9	2·9	2·5	–	0·8	2·2	..	..	..	..
S. & S.W. Africa[1]	0·2	1·5	1·0	2·0	2·0	0·9	2·5	3·1	2·8	1·2	5·8	1·5	4·4
U.K.[2]													
Wirebars	9·5	9·6	14·6	10·6	9·5	6·4	11·8	12·0	19·3	5·6	10·9	6·6	7·1
Other	10·3	13·5	15·0	10·8	15·6	9·4	11·2	12·0	10·4	9·8	15·3	13·8	15·7
Total	19·8	23·1	29·6	21·4	25·1	15·8	23·0	24·0	29·7	15·4	26·2	20·4	22·0
U.S.A.													
Producers	30·6	45·8	55·9	37·7	40·8	38·1	129·5	75·0	90·3	39·2	137·6	236·5	247·5
Wire Mills	49·9	55·4	41·3	20·1	24·2	33·9	95·4	84·4	44·9	38·5	97·9	108·1	103·8
Brass Mill			60·7	35·2	36·4	35·9	38·0	37·6	25·1	27·2	33·1	28·1	32·1
Total	88·5	101·2	157·9	93·0	101·4	107·9	262·9	197·0	160·3	104·9	260·6	372·7	383·4
Zambia[1]	7·6	7·0	8·0	8·8	8·0	5·1	9·5	7·3	5·0	5·8	9·7	21·6	19·6
Total Country Series	194·1	230·6	296·6	258·5	268·4	296·6	493·3	453·2	423·6	353·5	666·9	948·4	886·1
Total Commercial Stocks	202·5	247·6	314·6	282·5	298·4	318·8	581·7	611·9	659·1	393·6	832·0	1 536·1	1 671·9
U.S. Strategic Stockpile	993·8	814·1	410·4	249·6	237·1	229·9	229·9	228·3	228·2	228·2	32·0	23·7	38·2

Notes: [1] At Producers
[2] At Consumers

Source: World Copper Statistics Since 1950
Published by World Bureau of Metal Statistics

CRU feels this combination will hold copper prices at relatively low levels, with real strength only beginning to emerge towards the mid 1980's. Copper prices prior to that time will be sufficient to generate profits for almost all existing mines —particularly those which are fully amortized— but will be insufficient to encourage development of new, large, low-grade deposits.

CRU has calculated that a price of over $ 1.00 would be needed for profitable development of such low grade porphyries. This is based on capital costs for mine-smelter-refinery of about $ 6,500 per annual metric ton of capacity. Under financial terms now prevailing, capital costs for most new projects would exceeed operating costs. The study details the future pattern of capital and operating costs and forecasts that operating costs will increase by 2.8 % per annum in real terms.

While the CRU price forecasts would not be high enough to stimulate development of these large low-grade deposits, there will be increases in capacity over the forecast period. These will stem from the handful of large projects already under development, plus expansions at existing properties and the exploitation of smaller, high-grade deposits and those with rich by-product values.

The CRU study makes the point that while its trend-line price forecasts remains below $ 1.00 over the period, there could well be times during the peaks in the industrial cycle when significantly higher prices could be reached.

8.1.2 Ore prices

The price of ore is in function of the price of metal, treatment charge, refining charge and freight rates.

Treatment charges and refining charges are not published. For an accurate estimate of the evolution of the above charges, it would be necessary to know supply conditions practised to independent smelters of importing countries.

On the basis of the following data in hand, it results a constant increase in both the charges from 1960 to 1977, particularly with regard to the treatment charges.

Treatment charges refer to concentrates with 25 %-26 % copper content.

Years	Treatment charge c/lb Cu paid	Refining charge c/lb Cu paid
1960	2.5	2.5/3
1971	5/7	3/4
1972	8	5/6
1973	8.1	6
1974	9.25	6.8
1976	9.5	7
1977	11.6	7.7

Taking no account of the inflation rates of the American market from 1960 to 1974, treatment charges registered a growth rate in real terms of 5 %-6 %, and refining charges of 4 %-5 %.

CRU anticipates production costs to increase by 2.8 % per annum in real terms during the period 1975 to 1985.

8.1.3 Evolution of freights from 1970 to 1976

Survey of the trend of freights for the period under review was based on annual averages of medium tonnage freights calculated for the most significant trade of the market.

To better understand the phenomenon, we have indexed the freights so that average freight for 1970 equals 100 :

Years	Freight index
1970	100.0
1971	45.1
1972	45.6
1973	144.8
1974	168.3
1975	71.9
1976	74.7

From the above we can see that freights suffered a sharp decline in 1971 compared with 1970, and the same occured in the following year.

Throughout 1973 a net rise was registered followed by a further increase in 1974 ; a remarkable decline occurred in 1975 and at this level freight rates then settled throughout 1976.

Short and medium term forecasts

The first six months of 1977 confirmed the rates registered in 1976. The fail in recovery in world cereals and steel trading allow to anticipate that the current situation will be maintained during all the present year.

It is hazardous to make forecasts on medium term freights, however it may be reasonable to suppose that a real recovery in the market will occur only towards the second half of 1978, with the exception of the usual season oscillations.

It can however be stated with certainty that freights especially for major tonnages will not be able to move below the present average levels, without incurring laying up.

Relationship between bunker price and freight rates

There is certainly a relationship between the price of the bunker and levels of freights.

Oil represents in fact the most substantial, variable factor of the "ship cost" which accounts for about 30 %-40 % of the combined costs consisting of the running cost and bunker cost.

The incidence of the variation in bunker price on freights is higher when freight rates are —as at present— very low (that is when navigation pays just for the ship costs), while it is less sensible when freights are at good levels or indeed high levels.

8.2 Structure of the copper market

The primary copper industry is characterized by two types of producers, the vertically integrated companies and the custom smelters and refiners. The first group is active in every stage of copper production from exploration and mining through output of refined copper. This group includes the large government mining companies as well as the traditional copper producers. The second group relies on purchased raw materials and toll processing of customer materials. Some of the largest refineries in the world fall into the second class of producers. Copper is useful only in its final refined form, but the intermediate products, concentrate and blister, do move in international commerce. However, the actual trade in these two crude forms is limited to contracts between producers. Refined copper is freely sold on the world market.

Broadly speaking the excess of copper-smelting capacity of Western Europe is met by imports of copper concentrates chiefly from Asia, Oceania and South America. In Japan, where copper-smelting capacity far exceeds copper-mining capacity, the difference is offset by very large imports of copper concentrates from within the Asia and Oceania region (the Philippines principally) and from regions with an excess mining capacity, mainly North America.

A somewhat similar situation is registered in the international trade of blister, where the excess of refining capacity over smelting capacity of the industrialized regions is met by imports from developing producer countries.

Finally, in Western Europe the excess of semi-fabricating capacity over refining capacity is met by imports of refined copper which originate in all regions, but especially in Africa, which has a huge excess refining capacity.

Although Asia and Oceania maintain an approximate over-all balance between the two kinds of copper capacity, Japan (a country with an excess semi-fabricating capacity) receives its refined copper imports mainly from Africa, while Australia (a country with an excess refining capacity) ships its exports largely to Western Europe.

8.2.1 Copper concentrate

The international trade of copper concentrates expressed in metal content, which was of about 300,000 tons in 1967, registered a continuous increase (1.2 M tons- up to 1973-74, it then suffered from the economic crisis of 1975 and finally showed a partial recovery in 1976 reaching 1.1 M tons (about 18 % of the Western world's production).

The major exporters (Canada, Oceania, Philippines and Chile) provide for approx. 80 % of world exports.

Indonesia, Zaïre and Norway are amongst the minor countries of a certain weigh. Japan imports about 70 % of international trade and Germany F.R. about 20 %. The United States and, more modestly, Spain account for almost all the remaining part.

8.2.2 Copper blister

The international trade of copper blister (including leach cathodes shipped from Zaïre to Belgium), contrary to what happened for the copper concentrates, registered a decrease in 1967-76 period moving down from 800-900,000 tons to 700-800,000 tons, equivalent to slightly over 10 % of Western world's production.

Zaïre and Chile are currently accounting for 70 % of exports of blister, while South and South West Africa for just over 10 %.

Up to 1974 Peru held a remarkable quota on total exports (over 15 %), but this decreased to 6 % in 1976 due to the events occured in the refining sector of this country.

With regard to the major importers, during the decade under consideration, a sharp decrease was registered in imports of Japan and United States which accounted on the whole for 40 % in 1967 and fell to 10 % in 1976.

In the most recent years, EEC member countries (especially Belgium and more modestly Germany F.R. and United Kingdom) absorbed 60 to 70 % of the world trade of blister with a peak (85 %) in 1976 due to the exceptional imports of leach cathodes to Belgium from Zaïre.

8.2.3 Refined copper

International trade of refined copper increased to 2.6 M tons in 1976 from 2.0 M tons in 1967 reaching the highest point in 1974. In the most recent years international trade was equivalent to roughly 40 % of the Western world's production.

In the seventies the East/West trade was limited within 100,000 to 130,000 tons with some peaks which however never reached 200,000 tons.

The Western world was a net exporter up to 1971 for quantities oscillating in general between 40,000 and 70,000 tons, and a net importer thereafter, with a negligible exception in 1974, for quantities variable through the years (a peak was registered in 1973 with 69,300 tons).

Contrary to what is registered for concentrates and blister, international trade of refined copper is rather steady and a great number of countries who often operate in the movements in imports as well as exports is involved.

The major part of exports is accounted for by Zambia, Chile, Belgium, Canada, Zaïre (with the exception of 1976 for the well-known events occured) and Peru from 1976 onwards.

These countries represent altogether about 80 % of the total. The remaining part is divided amongst several countries, the principal ones of which are the United States, Australia and Germany F.R. The chief importers are the mainEEC member countries (including Germany F.R., the imports of which are four times higher than its exports, and Belgium which in general imports less than half of what it exports), Japan, the United States who have however in the long term an even trade balance, and in recent years Brazil.

8.2.4 Net imports of EEC

The EEC is a strong importer of copper in its various forms. Through the last 10 years net imports increased to around 1.9 M tons from 1.4 M tons.

Usually imports of copper as a whole meet approx. 65 % of the processing industry's demand, with the exception of a peak registered in 1975 when the economic recession altered the percentage incidence because of stocks piling up.

In recent years 60 %-65 % of net imports consisted of refined copper, 20 %-24 % of blister, 8 %-12 % of concentrates and 5 %-6 % of secondary materials (mainly scrap).

Regarding inter-EEC trade, it is to be noted that this is practically non-existent as to concentrates, it is modest as to blister (11 % of total gross inter-EEC imports in 1974), and it becomes more important in the case of refined copper (18 % of total imports in 1974).

Inter-EEC trade of scrap and residues is on the contrary rather steady accounting for about two thirds of the total for scrap and over 50 % for residues.

WORLD TRADE OF COPPER CONCENTRATES - METAL CONTENT - '000 TONS

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
IMPORTS										
Belgium-Lux.	2.7	3.6	3.0	5.3	9.4	6.4	7.7	8.1	11.7	6.0
Germany F.R.	72.1	94.3	91.5	83.2	81.6	149.7	238.2	173.6	166.1	196.9
Sweden	10.7	7.9	10.2	11.1	11.2	13.0	16.5	11.6	17.4	17.0
Spain	14.3	24.0	25.9	19.9	28.8	36.2	28.9	36.8	28.9	43.5
Japan	190.7	260.8	291.5	376.7	461.4	607.7	832.4	874.6	729.5	724.5
S. Korea								5.5	7.3	17.0
USA	29.9	25.0	35.4	30.7	28.4	49.9	38.9	50.8	67.1	81.6
Total *	320.4	415.6	457.5	526.9	620.8	862.9	1,162.6	1,161.0	1,028.0	1,086.5
EXPORTS										
Belgium-Lux.	0.9	3.1	0.4	0.3	-	0.6	-	0.3	1.3	1.0
France	-	-	-	-	-	-	0.2	0.8	0.5	0.5
Germany F.R.	0.1	-	-	0.1	0.3	-	0.2	1.3	-	-
Ireland					11.0	13.6	4.4	12.6	16.5	7.4
Italy	1.4	1.9	1.9	2.1	1.2	0.7	1.3	0.6	0.6	0.6
Norway	3.2	5.0	9.0	8.4	10.1	10.0	16.0	13.2	20.8	26.4
Sweden	0.8	0.3	0.3	0.5	0.9	3.5	9.2	8.5	5.2	8.0
Zaire	-	-	-	-	-	6.0	28.0	32.0	29.0	36.4
Other Africa	2.2	21.3	33.6	9.7	10.8	14.7	38.2	45.1	28.9	27.0
Philippines	85.8	106.4	131.4	160.3	197.4	201.3	210.0	221.0	210.0	236.0
Indonesia	-	-	-	-	-	-	37.9	63.7	61.2	66.8
Other Asia	24.2	34.4	31.9	31.0	31.5	27.5	25.7	22.3	27.5	25.0
Canada	117.0	146.8	143.2	162.5	204.1	270.3	346.4	343.8	314.6	308.9
Chile	29.8	35.1	40.2	38.7	78.9	74.9	109.2	155.9	103.8	156.2
Peru	30.8	23.0	30.2	30.2	45.1	35.0	38.5	62.7	24.4	12.1
USA	35.1	59.0	1.1	55.8	7.4	17.8	21.3	11.3	7.5	13.5
Oceania	12.2	10.5	10.7	27.2	36.1	155.1	225.6	241.0	216.7	228.1
Total *	343.5	446.8	433.9	526.8	634.8	831.0	1,112.1	1,236.1	1,068.5	1,153.9

* Total shown concerns the reported countries only

WORLD TRADE OF COPPER BLISTER

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
IMPORTS										
Belgium-Lux. (1)	318.0	285.8	209.4	260.0	273.5	279.3	312.9	270.5	262.9	413.0
France	11.6	16.4	15.8	16.5	16.4	15.7	18.3	23.7	26.8	23.9
Germany F.R.	164.8	146.4	144.8	133.3	155.7	117.9	121.2	98.2	119.8	130.8
United Kingdom	31.6	42.3	44.6	40.8	33.2	53.2	66.3	86.2	82.6	66.2
Austria	4.8	5.3	5.1	5.5	6.6	8.6	5.9	6.6	8.3	9.7
Portugal	1.5	1.7	1.3	1.9	1.5	1.0	1.0	1.5	1.3	0.3
Spain	19.9	12.3	12.2	15.0	7.4	2.9	4.4	15.6	37.4	20.6
Sweden	-	8.9	8.8	4.1	1.9	2.8	-	0.7	-	-
Yugoslavia	0.6	2.6	10.8	10.8	18.9	25.6	33.7	25.1	18.8	16.9
Japan	115.8	127.3	149.7	136.6	115.7	120.2	80.6	62.2	40.0	29.4
USA	244.3	245.6	215.9	203.6	141.2	142.8	139.8	188.5	79.2	40.4
Total *	912.9	894.1	818.4	828.1	772.0	770.6	784.1	778.8	677.1	751.2

(1) Including leach cathodes from Zaire

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
EXPORTS										
Belgium-Lux.	-	-	0.2	3.2	1.8	0.1	2.1	5.9	1.2	1.4
France	12.5	12.7	10.0	8.6	6.7	6.3	6.8	7.8	2.2	4.4
Germany F.R.	3.0	4.0	1.2	0.9	0.6	20.9	32.8	50.2	38.5	38.9
Norway	6.6	5.4	5.7	6.4	5.4	7.3	7.6	6.3	7.0	5.9
Spain	-	-	1.8	0.2	0.4	2.5	1.5	-	-	0.4
Sweden	-	-	-	-	8.5	5.6	2.2	-	0.5	2.8
S. and S.W. Africa	146.6	111.0	94.9	102.0	95.1	105.2	91.8	90.3	93.0	87.6
Uganda	14.6	15.4	16.6	15.9	16.8	14.1	9.6	8.8	5.0	5.0
Zaire	160.5	159.5	181.5	189.7	207.0	210.8	219.3	199.3	240.3	315.5
Zambia	79.9	91.6	108.1	103.7	98.3	87.6	42.7	31.9	19.0	21.1
Turkey	15.3	15.1	6.5	4.6	2.7	2.3	10.0	5.4	7.3	5.0
USA	19.0	14.3	3.9	7.1	26.0	7.8	6.8	2.4	1.4	2.5
Chile	240.7	224.5	188.0	190.1	170.3	149.9	159.5	214.5	179.8	231.0
Mexico	6.4	6.4	7.2	5.0	9.7	17.8	14.9	8.5	12.0	12.0
Peru	123.2	148.2	135.8	138.4	113.9	137.6	141.2	134.4	88.7	45.8
Australia	7.0	7.1	8.5	7.0	5.9	7.1	8.4	12.3	12.4	9.9
Total *	835.3	815.2	769.9	782.8	769.1	782.9	757.2	778.0	708.3	789.2

* Total shown concerns the reported countries only

WORLD TRADE OF REFINED COPPER

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
IMPORTS										
Belgium-Lux. (1)	58.1	117.7	134.9	121.5	119.1	115.3	124.9	118.5	139.5	119.1
Denmark	3.7	4.0	3.7	4.5	3.8	5.9	4.0	4.1	3.8	3.9
France	235.3	257.9	323.1	315.4	301.6	366.4	377.3	375.3	350.2	338.1
Germany F.R.	286.3	352.8	362.3	399.7	375.4	404.5	414.4	449.7	404.9	409.7
Ireland	-	-	-	0.1	0.1	0.1	0.3	0.3	0.2	0.4
Italy (2)	202.9	200.7	237.2	273.1	261.8	286.5	271.6	306.7	284.1	293.4
Netherlands	31.0	36.9	40.1	39.7	44.2	41.9	48.6	47.3	48.0	60.0
United Kingdom	419.6	416.3	416.7	409.5	366.1	395.5	399.4	380.7	369.0	367.9
Austria	15.8	16.5	21.2	26.4	27.7	28.9	25.2	26.5	15.9	10.4
Finland	6.5	8.1	7.7	14.2	10.3	9.0	8.8	14.5	11.8	16.3
Greece	8.4	8.4	12.5	9.9	13.5	15.6	18.0	15.2	24.0	23.7
Norway	3.2	3.1	2.3	0.6	0.3	0.3	0.4	1.2	1.4	2.5
Portugal	2.0	2.4	3.8	5.8	5.8	6.5	6.3	6.8	4.6	7.6
Spain	24.0	29.0	36.0	40.2	31.7	32.7	46.0	28.7	20.0	24.2
Sweden	64.3	68.7	62.3	61.7	64.1	62.5	59.5	65.4	68.5	65.6
Switzerland	41.1	37.4	37.9	48.6	40.5	30.3	27.1	31.2	33.9	21.3
Yugoslavia	13.5	15.2	21.0	30.6	41.2	47.9	25.6	31.0	14.2	42.0
South Africa	32.7	6.3	3.3	2.3	4.7	6.5	5.9	9.4	4.1	2.6
India	33.4	25.4	40.9	45.6	52.1	49.3	50.4	34.9	19.9	29.0
Japan	158.5	157.1	199.6	165.4	152.7	173.4	314.0	230.2	168.0	200.5
Taiwan	2.5	2.8	2.2	5.3	9.1	13.6	14.3	18.8	17.9	24.4
South Korea	1.5	1.9	0.9	4.7	1.9	3.8	4.3	11.1	6.3	8.4
USA	297.8	367.8	119.0	119.8	147.0	172.3	181.4	275.7	132.4	346.1
Brazil	36.4	49.0	49.7	52.9	70.4	84.8	94.0	136.6	126.3	148.2
Canada	4.8	5.3	16.5	13.2	19.9	16.2	17.2	22.1	10.9	9.1
Total *	1,983.3	2,190.7	2,154.8	2,210.7	2,165.0	2,369.7	2,538.9	2,641.9	2,279.8	2,574.4

(1) Excluding leach cathodes from Zaire
(2) Including some unrefined copper

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
EXPORTS										
Belgium-Lux.	286.4	309.8	254.3	295.2	273.4	265.8	319.4	288.9	248.6	307.5
Denmark	-	-	-	-	0.6	0.9	-	-	-	-
France	9.3	10.6	6.2	3.0	3.6	1.4	3.6	2.8	4.4	6.5
Germany F.R.	161.5	143.8	108.5	93.5	140.7	121.7	119.5	115.9	97.3	66.3
Italy (1)	5.5	5.5	2.8	5.4	4.4	3.7	7.1	8.0	3.3	2.3
Netherlands	1.6	4.2	1.5	1.4	0.9	5.2	7.2	16.8	10.5	7.0
United Kingdom	62.4	56.3	67.6	44.9	28.1	18.8	66.1	35.0	15.7	12.3
Austria	3.9	4.3	4.7	5.8	6.0	5.5	9.2	9.9	9.8	12.2
Finland	11.5	10.8	6.3	3.6	7.2	7.3	22.2	11.9	14.3	23.1
Norway	14.2	17.0	20.8	26.1	26.0	23.9	23.9	25.8	17.9	14.8
Spain	35.2	30.7	15.8	15.0	5.4	0.3	4.6	5.1	16.5	29.4
Sweden	32.3	27.4	33.8	27.2	23.1	17.9	10.2	16.6	19.0	30.4
Yugoslavia	1.4	9.4	17.5	27.6	46.2	90.3	77.9	71.7	45.9	35.0
South Africa	11.9	36.6	26.4	27.9	31.4	28.6	27.5	15.2	26.6	34.6
Zaire	161.0	166.0	183.3	180.0	198.7	214.7	229.2	252.0	224.0	74.0
Zambia	527.0	544.5	615.0	578.4	531.1	622.9	627.1	649.8	616.1	712.4
Japan	1.4	11.9	15.0	47.1	10.7	25.5	24.1	278.5	21.7	29.3
USA	146.6	227.7	195.0	201.4	170.3	165.7	173.3	113.3	156.2	103.5
Canada	250.3	250.9	190.5	265.3	285.1	293.4	290.0	282.8	319.6	313.2
Chile	361.3	377.8	428.3	440.0	434.8	406.0	387.8	487.8	504.2	594.7
Peru	33.6	33.1	32.0	32.6	28.5	27.4	27.0	38.5	36.9	122.1
Australia	9.5	16.8	32.7	31.4	49.2	59.8	48.2	70.5	90.4	68.9
Total *	2,127.8	2,295.1	2,258.0	2,352.8	2,305.4	2,406.7	2,505.1	2,796.8	2,498.9	2,599.5

* Total shown concerns the reported countries only
(1)Including unrefined copper

NET IMPORTS OF EEC - METAL CONTENT

'000 tons

	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976
Copper concentrate	72.4	92.9	92.2	86.2	78.5	141.2	239.8	166.1	158.9	193.4
Copper blister (1)	510.5	474.2	403.2	439.7	469.7	438.8	477.0	414.7	450.2	589.2
Refined copper	710.2	856.1	1,077.1	1,120.1	1,020.4	1,198.6	1,117.6	1,215.2	1,219.9	1,190.6
Copper alloy ingots	5.3	14.4	19.1	20.5	17.3	20.2	12.8	15.1	8.8	
Copper ashes and residues	-	3.0	6.2	7.7	3.0	9.3	16.3	26.8	23.6	
Copper scrap	51.1	128.2	126.6	140.6	72.5	40.2	85.4	74.7	55.1	
T o t a l	1,349.5	1,568.8	1,724.4	1,813.0	1,661.4	1,848.3	1,948.9	1,912.6	1,916.5	

(1) Including leach cathodes from Zaire

EEC FOREIGN TRADE OF COPPER ORE - GROSS WEIGHT - '000 TONS - YEAR 1974

Export \ Import	Germany F.R.	Belgium-Lux	EEC	Extra E E C: Sweden	Finland	Spain	Jugosl.	other	Total	World total	According to NIMEXE Export: E E C	World total
Germany F.R.	.	0.8	0.8	-	-	-	5.0	-	5.0	5.8	-	5.0
France	-	-	-	-	-	-	-	1.7	1.7	1.7	1.0	2.7
Italy	-	3.0	3.0	-	-	2.9	-	0.2	3.1	6.1	-	3.1
Netherlands	-	-	-	-	-	-	-	-	-	-	-	-
Belgium-Lux.	-	.	-	-	1.0	-	-	-	1.0	1.0	-	1.0
U.K.	-	-	-	-	-	-	-	0.1	0.1	0.1	0.2	0.3
Ireland	-	-	-	16.1	-	16.1	-	-	32.2	32.2	-	32.2
Denmark	-	-	-	-	-	-	-	-	-	-	-	-
E E C	-	3.8	3.8	16.1	1.0	19.0	5.0	2.0	43.1	46.9	1.2	44.3
Extra EEC												
Norway	24.9	-	24.9									
Spain	24.8	-	24.8									
Yugoslavia	10.0	-	10.0									
Other Europe	1.0	3.1	4.1									
Morocco	7.3	3.4	10.7									
South Africa	68.1	0.8	68.9									
Other Africa	2.3	7.5	9.8									
Canada	24.8	3.1	27.9									
Chile	91.4	1.6	93.0									
Other America	7.8	4.0	11.8									
Cyprus	17.4	-	17.4									
Indonesia	67.9	-	67.9									
Other Asia	-	0.6	0.6									
Australia	81.7	3.9	85.6									
Papua N. Guinea	141.7	-	141.7									
East. Countries	-	0.6	0.6									
Total	571.1	28.6	599.7									
WORLD TOTAL	571.1	32.4	603.5									

Source: Eurostat - Analytical tables of foreign trade - NIMEXE

EEC FOREIGN TRADE OF UNREFINED COPPER - '000 TONS - YEAR 1974

Export \ Import	Germany F.R.	France	Italy	Belgium Lux.	U.K.	EEC	Extra EEC: Spain	Extra EEC: Others	Extra EEC: Total	World total	According to NIMEXE Export: E E C	According to NIMEXE Export: World total
Germany F.R.	•	-	0.1	3.1	26.6	29.8	6.9	0.2	7.1	36.9	43.1	50.2
France	-	•	-	6.3	-	6.3	1.2	0.1	1.3	7.6	6.5	7.8
Italy	2.7	-	•	0.7	-	3.4	-	1.3	1.3	4.7	1.5	2.8
Netherlands	-	-	-	-	-	-	-	-	-	-	-	-
Belgium-Lux.	0.7	6.6	0.1	•	0.5	7.9	-	-	-	7.9	5.9	5.9
U.K.	1.2	0.7	-	1.9	•	3.8	-	-	-	3.8	0.7	0.7
Ireland	-	-	-	-	-	-	10.7	2.3	13.0	13.0	-	13.0
Denmark	-	-	-	-	-	-	-	-	-	-	-	-
E E C	4.6	7.3	0.2	12.0	27.1	51.2	18.8	3.9	22.7	73.9	57.7	80.4
Extra EEC												
Other Europe	5.4		-	8.1	-	13.5						
Zaire	0.1	10.6	0.2	210.1	-	221.0						
South Africa	51.5	1.5	-	24.7	1.7	79.4						
Other Africa	4.6	0.4	0.2	2.3	14.9	22.4						
Peru	16.1	-	-	10.9	2.1	29.1						
Chile	11.1	2.4	1.2	0.7	40.4	55.8						
Other America	0.4		1.3	1.2	-	2.9						
Asia	2.5		0.1	-	-	2.6						
Australia	0.5		-	-	-	0.5						
East. Countries	1.4		0.2	0.5	-	2.1						
Unspecified	-	1.5	-	-	-	1.5						
Total	93.6	16.4	3.2	258.5	59.1	430.8						
WORLD TOTAL	98.2	23.7	3.4	270.5	86.2	482.0						

Source: Eurostat - Analytical tables of foreign trade - NIMEXE
with modifications for the imports of France, UK and Belgium

EEC FOREIGN TRADE OF REFINED COPPER - '000 TONS - YEAR 1974

Export \ Import	Germany F.R. (1)	France	Italy	Netherlands	Belgium-Lux	U.K.	Ireland	Denmark	EEC	Extra E E C									World total	According to NIMEXE Export	
										Sweden	Switz.	Austria	Spain	other Europe	USA	Brazil	other	Total		E E C	World total
Germany F.R. (1)	.	17,1	2,9	4,3	18,9	8,7	-	-	51,9	0,4	7,5	16,3	0,9	0,9	4,8	8,8	4,7	44,3	96,2	38,1	82,4
France	2,7	.	1,0	-	0,1	-	-	-	3,8	0,1	-	-	-	-	-	-	0,1	0,2	4,0	2,6	2,8
Italy	1,9	-	.	-	0,2	-	-	-	2,1	-	-	0,3	-	0,5	-	-	-	0,8	2,9	4,4	5,2
Netherlands	0,9	2,0	2,4	.	3,4	3,4	-	-	12,1	-	-	-	-	0,4	8,3	0,2	-	8,9	21,0	7,9	16,8
Belgium-Lux.	62,6	110,3	21,2	8,7	.	2,1	-	4,0	208,9	13,6	11,5	0,7	8,3	3,7	1,9	5,9	1,4	47,0	255,9	241,9	288,9
U.K.	12,2	5,0	3,4	1,3	0,8	.	-	-	22,7	0,1	0,1	-	3,3	3,6	2,1	0,5	8,3	18,0	40,7	17,0	35,0
Ireland	-	-	-	-	-	-	.	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Denmark	-	-	-	0,2	-	-	-	.	0,2	-	-	-	-	-	-	-	-	-	0,2	-	-
E E C	80,3	134,4	30,9	14,5	23,4	14,2	-	4,0	301,7	14,2	19,1	17,3	12,5	9,1	17,1	15,4	14,5	119,2	420,9	311,9	431,1
Extra EEC																					
Norway	7,8	3,9	1,1	-	-	5,4	-	-	18,2												
Yugoslavia	11,3	8,3	8,4	0,4	0,2	-	-	-	28,6												
Other Europe	11,9	6,5	1,2	0,3	0,4	4,4	-	0,1	24,8												
Zaire	21,7	36,2	73,3	14,3	50,3	11,8	-	-	207,6												
Angola	6,0	2,2	0,1	0,9	3,4	-	-	-	12,6												
Zambia	80,3	73,4	75,8	-	2,1	134,3	-	-	365,9												
South Africa	3,5	2,6	8,7	-	3,0	18,1	-	-	35,9												
Other Africa	3,0	0,2	1,3	0,1	3,9	-	-	-	8,5												
USA	9,8	17,9	25,6	1,2	5,0	18,4	-	-	77,9												
Canada	23,6	21,2	7,0	1,4	6,1	94,4	-	-	153,7												
Chile	87,1	33,5	60,2	4,2	3,4	50,4	-	-	238,8												
Other America	1,2	0,8	0,4	-	0,8	-	-	-	3,2												
Japan	17,9	0,4	4,7	0,9	3,6	-	-	-	27,5												
Other Asia	0,3	-	-	-	-	-	-	-	0,3												
Australia	11,9	16,7	0,9	0,1	3,2	14,2	-	-	47,0												
East. Countries	62,4	17,1	3,7	9,0	9,7	15,1	-	-	117,0												
Unspecified	-	-	-	-	-	-	0,3	-	0,3												
Total	359,7	240,9	272,4	32,8	95,1	366,5	0,3	0,1	1.367,8												
WORLD TOTAL	440,0	375,3	303,3	47,3	118,5	380,7	0,3	4,1	1.669,5												

(1) Excluding trade with Germany D.R.
Source: Eurostat - Analytical tables of foreign trade - Nimexe, and other organizations

EEC FOREIGN TRADE OF COPPER SCRAP - METAL CONTENT - '000 TONS - YEAR 1974

Export \ Import	Germany F.R.	France	Italy	Netherlands	Belgium-Lux	U.K.	Ireland	Denmark	EEC	Extra E E C: Sweden	Switz.	Austria	Spain	Other Europe	USA	other	Total	World total	According to NIMEXE Export: E E C	World total
Germany F.R.	.	1.7	13.9	3.4	7.9	0.5	-	0.2	27.0	0.1	0.6	4.3	2.2	-	0.9	0.4	8.5	36.1	46.1	54.6
France	19.6	.	11.0	0.8	25.9	0.3	-	-	57.6	1.0	0.1	-	2.7	-	0.3	-	4.1	61.7	43.8	47.9
Italy	1.5	0.7	.	-	0.3	1.7	-	-	4.2	-	-	0.2	-	-	-	-	0.2	4.4	7.1	7.3
Netherlands	13.7	3.0	1.5	.	11.9	1.0	-	-	31.1	0.5	-	-	-	-	-	0.1	0.6	31.7	20.3	20.9
Belgium-Lux.	5.1	4.3	1.8	2.0	.	0.7	-	-	13.9	-	-	-	0.6	-	0.2	0.2	1.0	14.9	33.1	34.1
U.K.	10.5	1.0	3.1	1.4	10.9	.	-	0.1	27.0	0.1	0.5	-	0.5	0.1	0.3	0.6	2.1	29.1	8.7	10.8
Ireland	0.5	-	-	0.1	1.0	1.8	.	-	3.4	-	-	-	0.1	-	-	-	0.1	3.5	0.7	0.8
Denmark	7.8	-	-	0.2	0.4	0.4	-	.	8.8	0.1	-	-	-	-	-	-	0.1	8.9	4.7	4.8
E E C	58.7	10.7	31.3	7.9	58.3	6.4	-	0.3	173.6	1.8	1.2	4.5	6.1	0.1	1.7	1.3	16.7	190.3	164.5	181.2
Extra EEC																				
Norway	1.1	0.1	-	-	0.6	0.1	-	-	1.9											
Switzerland	5.9	0.2	1.6	-	2.0	-	-	-	9.7											
Spain	0.2	0.8	0.4	-	0.1	-	-	-	1.5											
Other Europe	1.3	0.6	0.7	0.1	0.3	0.4	-	-	3.4											
Zambia	-	-	-	-	-	3.8	-	-	3.8											
Algeria	-	1.2	-	-	-	-	-	-	1.2											
Angola	0.1	-	-	-	0.1	2.3	-	-	2.5											
Other Africa	3.2	1.0	0.7	0.5	2.2	0.5	-	-	8.1											
USA	8.8	0.3	11.4	0.6	15.5	3.5	-	-	40.1											
Canada	1.1	0.1	1.8	0.1	4.3	0.4	-	-	7.8											
Chile	0.6	-	-	-	-	4.0	-	-	4.6											
Other America	0.5	0.1	0.1	0.1	0.1	0.2	-	-	1.1											
Lebanon	0.6	-	0.3	-	1.4	0.1	-	-	2.4											
Israel	0.8	0.6	0.1	-	0.2	-	-	-	1.7											
Other Asia	1.6	-	0.4	0.1	0.9	0.3	-	-	3.3											
Oceania	0.2	-	-	-	-	0.3	-	-	0.5											
East. Countries	2.6	-	0.1	-	0.1	0.2	-	-	3.0											
Unspecified	-	0.1	-	-	-	0.1	-	-	0.2											
Total	28.6	5.1	17.6	1.5	27.8	16.2	-	-	96.8											
WORLD TOTAL	87.3	15.8	48.9	9.4	86.1	22.6	-	0.3	270.4											

Source: Eurostat - Analytical tables of foreign trade - NIMEXE

EEC FOREIGN TRADE OF COPPER RESIDUES - GROSS WEIGHT - '000 TONS - YEAR 1974

Export \ Import	Germany F.R.	France	Italy	Nether-lands	Belgium Lux.	U.K.	Denmark	EEC	Extra E E C					World total	According to NIMEXE Export	
									Austria	Spain	Sweden	USA	Total		E E C	World total
Germany F.R.	•	0.1	0.1	0.1	0.5	0.1	1.0	1.9	5.7	0.6	0.8	0.1	7.2	9.1	0.8	8.0
France	5.5	•	1.3	–	21.3	0.8	–	28.9	–	0.9	–	–	0.9	29.8	9.8	10.7
Italy	5.3	–	•	–	0.9	0.5	–	6.7	–	0.3	–	–	0.3	7.0	5.4	5.7
Netherlands	2.0	–	–	•	0.7	0.5	–	3.2	–	–	0.3	–	0.3	3.5	3.3	3.6
Belgium-Lux.	2.8	0.5	–	0.3	•	2.5	–	6.1	–	0.4	–	–	0.4	6.5	6.0	6.4
U.K.	0.3	–	–	0.1	0.1	•	–	0.5					n.a.	0.5	n.a.	n.a.
Ireland	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Denmark	0.4	–	–	–	–	–	•	0.4	–	–	0.2	–	0.2	0.6	0.5	0.7
E E C	16.3	0.6	1.4	0.5	23.5	4.4	1.0	47.7	5.7	2.2	1.3	0.1	9.3	57.0	25.8	35.1
Extra EEC																
Austria	0.3	–	2.0	–	–		–	2.3								
Norway	0.4	–	–	–	0.3		–	0.7								
Switzerland	4.2	–	0.3	–	–		–	4.5								
Yugoslavia	–	–	–	–	3.3		–	3.3								
Other Europe	0.4	–	–	–	0.6		0.3	1.3								
Africa	–	–	–	–	0.9		–	0.9								
Canada	1.1	–	–	–	0.7		–	1.8								
Chile	0.4	–	–	–	5.8		–	6.2								
USA	0.7	–	–	–	8.9		–	9.6								
Other America	0.1	–	–	–	0.6		–	0.7								
Japan	–	–	–	–	10.1		–	10.1								
Other Asia	0.1	–	–	–	0.1		–	0.2								
Australia	–	–	–	–	0.1		–	0.1								
East. Countries	1.5	0.1	–	–	–		–	1.6								
Total	9.2	0.1	2.3	–	31.4	n.a.	0.3	43.3								
WORLD TOTAL	25.5	0.7	3.7	0.5	54.9	4.4	1.3	91.0								

Source: Eurostat - Analytical tables of foreign trade - NIMEXE

8.3 Outlook for copper consumption

- Forecast by sectors of utilization
- Overall forecast by world areas
- Forecast for the Community

Review of the copper forecasts available in the literature

There is a general consensus that future developments of copper will be slower than in the past.

In fact, it is likely that historic growth of national economies in the developed countries will be tempered by the redistribution of factor income to the sources of raw materials mostly to the less developed countries. The required investment for pollution control equipment represents a deversion of capital toward nonproductive uses, effectively reducing industrial output. Also, the development of new consciousness toward work and economic growth may affect the productivity. The sum of such effects and the policy of Governments to control the inflation are expected to be a reduction in the rates of economic growth in the developed countries. However, the measures that it will be possible to take to check the phenomenon of unemployment should in some measure moderate the decrease in development of economic activity.

The combined effects of miniaturization and product saturation will determine lower rates of usage of copper in the metal's traditional applications.

Lastly, substitution by other materials will serve to displace copper from some applications and reduce its usage in others, while new applications of a considerable commercial development are not foreseen.

Forecasts on copper consumption to be found in literature usually refer to refined metal with some considerations on scrap recycling. Considering the relationship existing in the various economic areas between total consumption and refined copper consumption, and after framing some hypotheses on the recovery of old scrap, the above forecasts appear to be indicative also with regard to total consumption development.

COPPER CONSUMPTION FORECASTS - GROWTH RATES PER ANNUM

		Developed market	Developing market	Western world	World	EEC	USA	Japan	Other Western world
United Nations refined copper	1976-1990	2.9	10.3	4.5	4.3				
CRU refined copper	1972-1980			2.5					
	1981-1985			>2.5					
Informal Mining Group refined copper	1974-1985			3.5					
SRI refined copper	1974-1990			3.3	3.2	2.0	2.5	4.7	4.9
US Bureau of Mines primary copper	1973-2000				3.9		2.9		
secondary copper	1973-2000				4.1		5.0		
total	1973-2000				3.9		3.4		
US Dept. of Commerce total copper	1976-1985						4.8		

United Nations, Economic and Social Council : Future demand and the development of the raw materials base for the copper industry, March 1977. Forecasts were made using an econometric model. Equations for each of the current principal consuming areas were employed. These equations incorporated relations between copper consumption manufacturing output levels in the major copper-using sectors, relative prices, and a variable representing a copper saturation constraint.

According to United Nations world consumption of refined copper is expected to grow at the rate of about 4.3. % per year between 1976 and 1990. This rate of growth, which is about equal to the historic rate, is strongly influenced by the partially offsetting factors of continued sustituion for copper by other materials and accelerated consumption of copper in the developing economies.

Projected consumption of refined copper in selected future years

	Quantity (thousands of tons)				Share (percentage)		
	World total	Developed market economies	Developing market economies	Centrally planned economies	Developed market economies	Developing market economies	Centrally planned economies
1980	10,900	7,290	859	2,730	67.2	7.9	25.1
1985	13,500	8,450	1,450	3,560	62.8	10.8	26.5
1990	16,900	9,790	2,440	4,660	58.0	14.4	27.6

Principally because of the high rate of future industrial growth assumed for developing countries, the share of copper consumption of these countries is expected roughly to double over the period.

The average annual growth rates of copper consumtion between 1976 and 1990 are : 2.9 % for the developed market economies ; 10.3 % for the developing market economies ; 5.5.% for the centrally planned economies.

Projected refined copper consumtion in the developed market economies by end-use in selected years.

	Total	Electrical and electronic equipment	Building products	Transportation equipment	Industrial equipment	Consumer products
		Quantity (thousands of tons)				
1980	7,290	3,610	1,100	860	1,200	539
1985	8,450	4,080	1,290	1,020	1,420	650
1990	9,790	4,610	1,530	1,220	1,680	783
		Share (percentage)				
1980	100.0	49.5	15.0	11.8	16.4	7.4
1985	100.0	48.3	15.3	12.1	16.8	7.7
1990	100.0	47.1	15.6	12.5	17.2	8.0

It will be noted that the share of refined copper consumption within the developed market economies represented by consumption specifically for electrical equipement is expected to decline between 1977 and 1990. The decline reflects the slowdown in the growth of electricity consumption.

Each of the other end-use categories is expected to increase its share of total copper consumption.

<u>Commodities Research Unit</u> : <u>Outlook for the world copper industry 1975-1985, May 1976.</u>

The forecasts were developed with the aid of a long-run econometric model of the copper industry. CRU's assumptions about long-run rates of industrial production are that growth in the late 1970's and early 1980's will be roughly on the long run historical average of 4 % in the USA and 5 % in Europe. The growth rate foreseen for Japan is about half of the past trend rate of 11 %.

According to the press release of CRU the copper industry is moving into a period of below-trend consumption growth.

CRU research has revealed that capital spending in the crucial electrical generation and transmission sector of the market will be very low troughout the industrialized world over the next four-to-five years. This factor, plus the expectation that the period of spectacular growth in the Japanese economy has passed, has led CRU to forecast a copper consumption growth trend of only 2.5 % per annum through 1980.

1972 was used as the base year for the forecast period since it was the last "normal" year before the boom-bust cycle of 1973-1975. However, CRU anticipates a stronger growth rate in the 1981-1985 period once the slack in the electrical generating system has been absorbed and new capital investment projects have been launched.

With regard to the role of scrap in the copper market, it is believed that despite governmental recycling campaigns throughout the industrialized world, there is likely to be very little expansion of the secondary industry in Europe and USA.

<u>Informal Mining Group</u> : <u>Outlook for copper in the EEC to 1985- Second half 1976.</u>

According to the Informal Mining Group a simple consideration of the end-uses for refined copper suggests that a slower rate of growth compared with that registered in the 60's and early 70's, is likely to be experienced after the recession in some of the chief sectors of consumption if only because of the slower rate of economic growth in the Western World which most authorities expect. Thus 3.5% per annum seems of more likely average trend in the Western World.

Estimates shown below were made using a RTZ copper model, and projected industrial production growth rates for the relevant countries as the main exogenous factors. "Real" price is also included in the equation. These estimates are concerned solely with the underlying trend in demand, after the world economy has picked up from the 1974-1975 recession. No attempt has been made to superimpose any economic cycle around this trend. Suggestions of a tailing off in economic recovery in 1978-1979, or a further severe recession are ignored. It is assumed that any cyclical variations will be around the trend shown.

'000 tons

	Demand	Less net adjustment to end-uses	Demand considered
	(a)	(b)	
1976	6,250	-	6,250
1977	7,227	50	7,177
1978	7,429	100	7,329
1979	7,630	100	7,530
1980	7,894	150	7,744
1981	8,225	150	8,075
1982	8,605	200	8,405
1983	9,012	200	8,812
1984	9,430	250	9,180
1985	9,858	300	9,558

a) With medium rate economic recovery assumptions (see below), demand derived from copper model but with 1976 separately estimated.

b) Estimated net new developments in end-uses not reflected in historical data (1956-1974) on which model is based, mostly the phasing out of copper radiators and substituion of other materials in wiring cables.

The extimates for industrial production used, in terms of % increase on the previous year, are given below :

INDUSTRIAL PRODUCTION

% p.a.

	USA	JAPAN	EEC	REST OF THE WORLD
1977	6	8	5	5
1978-85	3 3/4	7	4	5

Alternative runs were made with slower rates of recovery (e.g. 3 % p.a. for the USA from 1979 onwards), and also faster rates (e.g. 4.5 % p.a. for the USA) ; these produce total demand (a) above in 1985 of 9,290 and 10,496 respectively, i.e. about 600 either side of the value for the medium run adopted.

As far as scraps are concerned, this study consider only the secondary materials which pass through smelters and refineries and it is believed that they will reach in the Western world 1.2-1.3 M. tons in the future compared with 1-1.1 M. tons registered in the last decade.

Stanford Research Institute : World Minerals Availability, 1975-2000- Copper, April 1976.

The intensity of use approach was employed by SRI to make forecasts : this method correlates the logarithm of GNP per capita to the logarithm of consumption per unit of GNP.

The principal assumptions implicit in the projections technique are :

- No major disruption will occur in world economy
- Population growth rates will continue to decline
- Domestic economic product will increase at rates somewhat below recent historic levels.
- No sudden changes in copper consumption are anticipated
- Copper consumption in less developed countries will follow the trends established in more developed economies.

According to SRI estimates given in the table, a decline in copper growth rates will occur compared with those registered in the last 20 years.

WORLD CONSUMPTION OF REFINED COPPER

'000 tons

	1980	1985	1990	2000
Western Europe				
Belgium/Luxembourg	192	219	255	359
Denmark	3	2	2	3
France	431	467	531	616
Germany, West	801	909	1,059	1,381
Italy	346	391	440	568
Netherlands	49	57	67	97
Norway	5	6	6	6
Portugal	21	27	32	35
Spain	234	312	370	460
Sweden	113	117	120	140
Switzerland	22	17	13	13
United Kingdom	500	500	507	550
All other	235	252	270	352
Subtotal	2,952	3,276	3,672	4,580
Eastern Europe	526	569	625	757
USSR	1,345	1,509	1,708	2,390
Noth America				
Canada	264	288	317	420
United States	2,288	2,570	2,919	3,827
Subtotal	2,552	2,858	3,236	4,247
Latin America				
Brazil	264	426	634	1,341
Mexico	115	186	282	499
All other	131	205	317	625
Subtotal	510	817	1,233	2,465
Africa and Middle East				
South Africa	99	136	186	349
All other	35	49	68	132
Subtotal	134	185	254	481
Asia and Far East				
Australia	117	100	151	196
China, People's Republic	393	504	649	1,077
India	79	99	124	192
Japan	1,180	1,419	1,720	2,617
All other	56	73	99	188
Subtotal	1,825	2,225	2,743	4,270
World total	9,844	11,439	13,471	19,190

ANNUAL GROWTH RATE IN REFINED COPPER CONSUMPTION
(percent)

	Actual 1953-1974	Projected 1975-1980	1980-1985	1985-1990	1990-2000	1975-2000
Western Europe	4.8	3.7	2.1	2.3	2.3	2.5
Eastern Europe and USSR	5.7	3.3	2.0	2.3	3.0	2.8
North America	2.1	1.9	2.3	2.5	3.0	2.5
Latin America	7.5	11.0	9.9	8.5	7.2	8.7
Africa and Middle East	7.5	6.6	6.6	6.6	6.6	6.6
Asia and Far East	10.7	2.6	4.1	4.3	4.5	4.0
World	4.6	3.3	3.1	3.3	3.6	3.4

With regard to scrap SRI emphasise that the historic ratio of scrap to copper consumption has been relatively stable. The ratio may increase somewhat during the projection period as improved recovery methods and favourable legislation make secondary copper economically attractive. However, there have been no recent developments or trends to indicate a significant departure from the historic relationship.

US Bureau Of Mines : Mineral Facts and Problems, 1975 edition

Total US demand for copper in 2000 is forecast to be between 4.1 and 7.5. million short tons. The most probable demand within the range is established at 6.0 million tons, representing an annual growth rate of 3.4 % between 1973 to 2000.

Demand for each of the major end-uses was projected to 2000 by means of a forecast base obtained by relating demand to the most appropirate economic indicator selected from electrical energy, gross national product (GNP), new construction, productivity - output per civilian, or total population. The forecast base was modified by contingency assumptions for technology and other factors, resulting in the creation of a demand range for each end-use in 2000. Analysis, based largely on knowledge and judgement of emerging trends of utilization, was applied to each end use to determine the most probable point within the range. The sum of the most probable demand for each use established the total probable demand for copper in 2000.

In consideration of the anticipated increased emphasis on and incentives for recycling, the portion of supply furnished by old scrap was increased from the approximate 20 % of recent years to 25 % for 1985 and 30 % for 2000. This resulted in an annual probable growth rate from 1973-2000 of 5.0 % for secondary supply. The remainder, supplied by primary material, will grow at an annual rate of 2.9 %.

Total demand for copper in the rest of the world is forecast to range from 16.9 million tons to 28.5 million tons with a probable demand of 24.0 million tons in 2000. Rates of growth were a composite based on the assumption that demand in industrialized, developed countries will increase at rates similar to the growth in US demand, while in developing nations the rates of growth will be considerably higher.

The distribution between supply from secondary and primary sources was derived using a rationale similar to that described for the United States.

SUMMARY OF FORECASTS OF US. AND REST-OF-WORLD COPPER DEMAND, 1973-2000

'000 short tons

	1973	2000 Forecast range Low	2000 Forecast range High	Probable 1985	Probable 2000	Probable average annual growth rate 1973-2000 (%)
UNITED STATES						
Primary (1)	1,942	2,900	5,300	2,700	4,200	2.9
Secondary	486	1,200	2,200	900	1,800	5.0
Total	2,428	4,100	7,500	3,600	6,000	3.4
Cumulative (primary)	...	65,000	92,000	28,000	80,000	...
REST OF WORLD						
Primary (1)	6,058	12,700	21,400	10,000	18,000	4.1
Secondary	2,158	4,200	7,100	3,000	6,000	3.9
Total	8,216	16,900	28,500	13,000	24,000	4.0
Cumulative (primary)	...	246,000	337,000	97,000	301,000	...
WORLD						
Primary (1)	8,000	15,600	26,700	12,700	22,200	3.9
Secondary	2,644	5,400	9,300	3,900	7,800	4.1
Total	10,644	21,000	36,000	16,600	30,000	3.9
Cumulative (primary)	...	311,000	429,000	125,000	381,000	...

(1) Copper use by industry less old scrap

US Copper demand by end-use

Electrical

The forecast base of 3.6. million tons for demand in 2000 was obtained by relating electrical apparatus and electrical transmission sectors to the economic indicator for electrical energy, and the communications and household appliance sectors to GNP. Increased emphasis on safety, comfort, recreation, and a pollution-free environment will probably be reflected in great demand for electrical equipment. Automation, including use of computers, also should spur use of electrical equipment. The trend of underground power distribution systems is likely to boost the use of copper over that of competitive materials for technological reasons. Because of these strong factors, the high was set at 5.1. million tons.

A number of contingencies inhibiting the use of copper were considered in deriving the low of 3.0 million tons. These include substitution of aluminum and copper-clad aluminum for copper, advanced power generation systems not requiring generators, cryogenic techniques in power transmission, microminiaturization of communication circuitry, and use of satellites for communications.

The factors of relative price advantages of substitute materials and technological developments reducing unit material requirements are expected to partly offset the large increase in the demand of copper for electrical purposes, and, on balance, the probable use in 2000 is placed at 4.3. million tons.

Construction

A forecast base of 0.8 million tons for 2000 was obtained by relating the demand for copper in this category to the forecast growth of new constructions. A plus factor in use of copper for construction is the prestige image often desired in housing. Technological improvements in the copper cladding of building materials should promote this decorative use. Another factor may be insistence on superior performance in materials to combat future high maintenance costs.

Factors curtailing the use of copper in construction include the trend to multiple housing units, which reduces material needs per unit, and the use of substitute material to replace the relatively high-priced copper.

In consideration of the preceding contingencies, the demand was set from 0.4 million to 0.8 million tons, and the probable demand in 2000 was placed at 0.6. million tons.

Machinery

Projection of this end-use category at the anticipated productivity- output per civilian- growth rate yields a forecast base of 0.4 m. tons for 2000. Anticipated strong growth in the areas of air-conditioning and desalination, where copper alloys are a preferred material, should enhance copper demand. A supplementary consideration is the relative ease of copper fabrication, which will become more important as labor costs rise faster than raw material costs. The principal deterrent to use of copper for industrial machinery is its high cost relative to that of alternate materials.

Owing to the cited strong growth factors, the high was established at 0.6 million tons, and because of the substitutability pressure the low was set at 0.35 million tons. The probable demand in 2000 was set at 0.4. million tons.

Transportation

Relating the growth in demand for copper in transportation to population growth resulted in a forecast base of 0.23 million tons. A high of 0.4. million tons could be attained through increased number of cars per family, continued popularity of house trailers for recreation, more leisure time, installation of rapide transit systems for major cities, and expected growth in aircraft passenger miles travelled. On the other hand, possible environmental restrictions excluding automobiles from metropolitan commuting as an antipollution measure, a shift from truck to rail haulage, and loss of copper markets to aluminum and plastics could reduce demand to a low of 0.15 million tons. Energy considerations and public attitudes favour a movement toward improved mass transportation systems, particularly for short-distance travelling, and expansion in railroad transportation. This resulted in a probable demand of 0.25 million tons of copper in 2000.

Ordnance

Growth in demand for copper used in ordnance applications was projected at the growth rate for total population and resulted in a forecast base of 0.07 million tons in 2000.

Political instability in foreign nations requiring military expenditures for arms exports could create a demand for copper of 0.2 million tons. Universal disarmament, however, could limit the use of copper to a low of 0.05 million tons.

Expectation of a significant peacetime military strength indicates a probable demand of 0.15 million tons of copper in 2000.

Other (including Jewelry, Pigments, and Coinage)

Demand for the many miscellaneous uses for copper was forecast to grow at the same rate as GNP to a forecast base of 0.3 million tons of copper. The popularity of copper Jewelry for aesthetic purposes together with continued research in other areas may result in increased requirements for copper chemicals and inorganic pigments. Copper as a trace element is needed to sustain the life of plants, animals, and humans. Copper in coinage has extended copper's usefulness and the technique of laminating or cladding copper with other materials may result in its use in diverse applications. These contingencies could result in a copper demand as high as 0.4. million tons in 2000. On the other hand, the use of alternate materials and the use of credit cards in place of coinage could lessen the demand for copper to 0.15 million tons.

It is likely that the copper demand for the many miscellaneous items in which it is used will continue to increase ; hence the probable demand is set at 0.3 million tons in 2000.

PROJECTIONS AND FORECASTS FOR US COPPER DEMAND BY END USE, 1973 AND 2000

'000 short tons

End use	1973	2000			
		Contingency forecasts for United States			
		Forecast base	Forecast range Low	Forecast range High	Probable
Electrical	1,445	3,600	3,000	5,100	4,300
Construction	355	800	400	800	600
Machinery	253	400	350	600	400
Transportation	198	230	150	400	250
Ordnance	57	70	50	200	150
Other (including jewelry, pigments, and coinage)	120	300	150	400	300
Total	2,428	. . .	4,100	7,500	6,000

Outlook for copper consumption

Forecasts concern the long period up to 1990.

1972 was taken as the base year being this the last "normal" year available.

Forecasts for the EEC were developed by end-use sectors and considered together with those for the other economic areas (USA, Japan, rest of the Western world) which were made on the basis of the literature available.

In general terms, while the long-term prospects for electricity and telecommunications are encouraging, the medium-term outlook is not as favourable. In the United States for example the immediate outlook is for a tapering off of investment in electrical generation and transmission. Plant and equipment expenditures in this sector are unlikely to return to their peak 1974 level until 1980, and only modest growth is likely tereafter. A similarly cautious assessment must be made of the market for electrical equipment, including switchgear, motors and generators, and transformers. The telecommunications market, too, offers little chance of rapid growth, since there are many signs of near-saturation in the market for domestic telephones.

In Japan, telecommunications have the worst prospects, the domestic market is virtually saturated, since the level of telephone ownership is very high. The outlook for cables and for generating equipment is somewhat better, mainly because the market is not so close to saturation, but a much slower growth is expected in demand for electricity. The demand for electric motors is likely to be among the least bouyant in the electrical sector and this will have an adverse effect on the demand for enamelled wire.

In most industrialized countries, both the short-term and long-term prospects in construction, transportation, industrial engineering and consumer products are better than those in the electrical sector. The registration of new cars will follow the general business cycle in the developed market economies. The demand for copper in industrial machinery is likely to be stimulated by expecially rapid growth in two areas : mining and oilfield machinery, where the energy situation has led to major increases in the demand for both domestic and export markets ; and valves and pipe fittings, which have been relatively strong markets even during the recession. Continued expansion in the demand for copper in consumer products is expected in all industrialized countries.

The main alternative products to copper are aluminium, stainless steel and PVC.

The displacement of copper has occured because of the superior physical properties of alternative materials in particular applications. In addition, relative prices have played a major role also because of the instability of copper prices.

According to Commodities Research Unit the impetus given to substitution by certain European public utilities can be traced directly to fears of uncertain supply and fluctuations in copper prices.

Forecast for the Community by end use sectors

Demand for each of the major end use has been projected by means of a forecast base by relating demand to the most appropriate economic indicator selected from electrical energy consumption, GNP, GNP per civilian and population. The forecast base was modified by contingency assumptions and other factors based largely on knowledge and judgement of engineering trends of utilization.

Demand is expressed in copper total consumption and forecasts have veen carried out on the portion of supply furnished by old scrap.

Economic indicators

As to GNP, for the period 1972-80 it was considered a growth rate per annum of 3.5 %, and of 4.0 % for the following decade. As to GNP per civilian, a 3.5 % growth rate to 1980 was considered, equal to the GNP rate. Such 3.5. % rate was maintained constant also for the successive period having taken into consideration possible policies of governments to cut the rate of unemployment.

As to electric energy consumption, a 5.3 % growth rate per annum was considered, as it resulted from the "World Energy Outlook, 1977" by OECD. As to population, it was considered a growth rate along the line of that one registered in the most recent years, equal to 0.6 % per annum.

End use sectors-

Electical

This sector comprises copper for generation, transformation, large distribution and cabling of electric energy in houses and industrial plants as well as for telecommunications and appliances. The electric part referring to the transport and domestic goods sectors is excluded.

This sector is almost completely covered by copper wire, with some small portions of other semifinished products.

Construction

This sector covers all those semifinished products, with the exception of the electrical part, which are used in civil and industrial building. It will therefore include water piping, air conditioning and heating systems, roofing and all decorative and architectural elements made of copper and copper alloys.

Transport

This sector considers all consumption regarding transport by road vehicles, ships, aircraft, railways, including that part relating to the electrical sector comprising motors and contacts of any kinds.

General engineering

It covers a wide and diverse field including : mechanical engineering equipment ; water turbines ; power station equipment ; machine tools and other heavy industrial process plant ; heat exchangers ; precision instruments, and a wide use of fasteners. It also includes the chemical industry in all its ramifications.

Domestic goods

Domestic ornamental and decorative parts and household appliances are here included.

Correlations between economic indicators and end use sectors were set as follows :

75 % Electrical	- Electric energy consumption
25 % Electrical	- GNP
Construction	- GNP
Transport	- Population
General engineering	- GNP per civilian
Domestic goods	- GNP

Consumption in the electrical sector was divided into two homogeneous subsectors : the first one, including generation, transformation, large distribution and cabling in houses and plants, equal to 75 % of the electrical sector consumption ; the second one, including telecommunications equipment and appliances, equal to 25 %.

Electrical sector

In the overhead power lines, copper has already been substituted by aluminium, and it maintains some specific uses, for which it has been regarded as unreplaceable.

In the underground power cables, copper is being gradually replaced by aluminium.

In the period 1969 to 1974, in Germany the growth in copper consumption for this specific use accounted for just 0.6 % per annum, while aluminium consumption rose by 7.5 %. If aluminium were to be given general application, it is assumed that copper consumption will decrease, or at least no increases will be registered in the most technologically advanced countries.

In France, copper will be replaced shortly by aluminium in the wiring cable ; 15,000 tons of copper are estimated to be replaced in a few years' time.

A sector susceptible of replacement is the magnet wire one. Following the example of the USA, where about 60 % of small motors is currently produced with aluminium windings, a partial substitution may be assumed.

In the telecommunications sector, the communication wire presents no problems regarding substitution. In the cabling sector, on the contrary, it is foreseen for 1985 an almost total substitution by aluminium in the United Kingdom, where at present aluminium registers a 25 % penetration rate. Besides, it is to be mentioned the possibility of replacement by optical fibres, already being tested on a semi-industrial scale, and wave guides.

No information is available on the possibility of replacement as far as appliances are concerned.

Finally, while for the electical sector the projection showed a total consumption of $1,720.10^3$ tons for 1990, under hypotheses A and B this consumption has been indicated as $1,520.10^3$ and $1,420.10^3$ respectively ; on the contrary, the figures for 1985 are : $1,540.10^3$; $1,410.10^3$; $1,340.10^3$.

Construction

In this sector, substitution by PVC and stainless steel may occur. In France it is deemed that 60 % of copper used in this sector is potentially replaceable.

On the other hand, in Europe, contrary to the USA, changes caused by the multiple house are not being considered, and some countries, such as Italy for example, are going to use copper instead of galvanized steel for hot water pipes. It is therefore deemed that the Italian situation could counterbalance possible reductions taking place in France, so that the forecast base : 550.10^3 tons for 1985 and 600.10^3 tons for 1990 was maintained.

Transport

In this sector, and especially the road transport one which accounts for 60 % of it, it is not deemed advisable to modify the forecast base of 370.10^3 tons for 1985 and 410.10^3 tons for 1990, as if the designing of the copper radiator were to be modified, this could still be regarded as competitive and preferable to others, above all for its higher corrosion resistance performance. Though aluminium radiators account at present for 20 % of European consumption, it was assumed that a possible decrease in copper consumption could be compensated by a higher development in the established industrial truck market and the areas where emissions are unacceptable and the new market will be for postal delivery and possibly bus system.

General engineering

The use of copper in this field is sufficiently diversified to make it relatively secure for some time to come. In traditional copper products such as valves and bearings, the metal faces competition from numerous materials. The forecast base was assumed as follows : 570.10^3 tons for 1985 and 630.10^3 for 1990.

Domestic goods

There are no elements to modify the results of the correlation : 220.10^3 tons for 1985 and 240.10^3 tons for 1990.

On the basis of what previously stated, it is now possible to proceed and develop the following table to which have been added 250.10^3 tons for 1985 and 300.10^3 tons for 1990 referring to direct exports of semifinished products, considered as constant in terms of percent, compared with total consumption. The total consumption of the Community will then amount to a tonnage varying between $3,300.10^3$ tons and $3,500.10^3$ tons for 1985. For 1990, on the contrary, the tonnage will vary between $3,600.10^3$ tons and $3,900.10^3$ tons, with a growth rate, compared with 1972, of 1.6 % and 2.1 % respectively.

In order to calculate the demand for primary metal in the Community, the new and old scrap must be deducted from total consumption. In line with the pattern of consumption assumed, the percentage of new scrap at the fabricating stage is 21 % compared with total consumption. This figure must be increased by 10 % plus 5 % of secondary production for the reasons stated in Appendix IV. The old scrap has been indicated under two hypotheses of recycling at pages 5-52 and 5-53.

PROJECTIONS AND FORECASTS FOR EEC COPPER DEMAND BY END USE - '000 TONS

	1972	1976	1985			1990		
			Forecast base	Hypothesis A	Hypothesis B	Forecast base	Hypothesis A	Hypothesis B
Electrical	1,161	1,291	1,540	1,410	1,340	1,720	1,520	1,420
Construction	428	462	550	550	550	600	600	600
Transport	276	300	370	370	370	410	410	410
General engineering	440	462	570	570	570	630	630	630
Domestic goods	181	200	220	220	220	240	240	240
Total copper use	2,486	2,715	3,250	3,120	3,050	3,600	3,400	3,300
To add: estimated net exports	198	173	250	250	250	300	300	300
Total copper consumption	2,684	2,888	3,500	3,370	3,300	3,900	3,700	3,600

Overall forecast

On the basis of the previously mentioned reports and literature available, the following growth rates have been assumed :

3.0 % for the USA ; 4.5 % for Japan ; 1.7 % for developed countries ; 8.4 % for developing countries.

The following table shows the final data for 1972 and 1976 and forecast for 1985 and 1990 of total consumption of copper.

'000 tons

	1972	1976	1985	1990
EEC	2,684	2,888	3,370	3,700
USA	2,976	2,559	4,400	5,100
Japan	1,311	1,409	2,330	2,900
Other developed countries	1,034	1,062	1,300	1,400
Developing countries	563	777	1,600	2,400
Western World	8,568	8,695	13,000	15,500

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World Metal Statistics

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9. SUPPLY AND DEMAND EQUILIBRIUM AND ITS EFFECTS UPON THE SITUATION IN THE COMMUNITY

9. Supply and Demand Equilibrium and its Effects upon the Situation in the Community

9.1 World Supply and Demand Equilibrium

The supply and demand equilibrium of the refined copper market has been established for the Western World alone since the socialist countries take part only in terms of the final import-export balance with the western countries.

Copper supply (Table n°36) is determined on the basis of several hypotheses :

. Mining production capacity is obtained by adding to 1976 mining capacity the new capacity which is to come into service in the next few years and substracting the capacity of mines closed because of the exhaustion of reserves or other technical and economic reasons. This capacity which is closed each year is estimated at 100 kt of copper per year but obviously in reality this factor is closely linked to the economic situation ;

. Mines are supposed to operate at 85 % of capacity ;

. Refinery projects have not been taken into account. Since the time necessary for developing a refinery is much shorter than that necessary for a mine it is supposed that refined copper production capacity will easily adapt itself to mine production capacity although the political aspects of the location choice may sometimes delay projects.

. Refined copper production accounts for 95 % of concentrates production with 5 % of the copper being lost during the various operations ;

. Scrap used in the western refineries represents an average of 1 Mt of copper per year and during the last ten years between 895 kt (in 1975) and 1,182 kt (in 1974) were used with recovery depending to a great extent upon the economic situation. It is obvious that this assumption represents a conservative one not taking into account the projected efforts to increase the recycling, notably in the EEC (see chapter 5). These efforts will mainly have an effect in long term.

. The credit balance for exports of the socialist countries continues to increase regularly and should reach 200 kt of copper in the short term.

TABLE N° 36

TREND FOR REFINED COPPER SUPPLY FROM PRESENT TO 1982

(in kt Cu)

	Mine production capacity	Actual Mine production	Refined copper Production from ore	Scrap reused by refineries	Export Surplus of Socialist Countries	Available Refined Copper
1976	7 366					
1977	7 769	6 604	6 274	1 000	80	7 354
1978	8 053	6 845	6 503	1 000	100	7 603
1979	8 382	7 125	6 769	1 000	140	7 909
1980	8 747	7 435	7 063	1 000	170	8 233
1981	8 893	7 559	7 181	1 000	200	8 381
1982	9 598	8 158	7 750	1 000	210	8 960

Taking into account all these hypothesis some of which may be considered doubtful it is possible to draw up a schedule of quantities of refined copper available in the Western World from now up to 1982. It did not appear possible to make forecasts for after 1982 since we do not have a "realistic" sufficiently complete schedule of mining projects after that date.

Refined copper demand in the Western World has been established on the basis of two hypothesis (Table n°37) :

. Low hypothesis : copper demand will increase by 6 % in 1977 and then by 3 % per year during the following ten years (on the basis of 1972 demand) ;

. High hypothesis : copper demand will increase by 6 % in 1977 and then by 4 % per year during the next ten years.

These hypotheses are based upon the following results :

Refined copper consumption in the Western World increased on an average of 4.5 % per year between 1955 and 1966 and by an average of 3 % per year between 1967 and 1976.

In view of this, there seems to be little chance that the high hypothesis will be verified since real refined copper consumption in the Western World should be quite close to the low hypothesis forecast.

By comparing the refined copper supply and demand forecasts for the Western World obtained in this way (Table n°37) it is possible to draw the following conclusions in spite of the relative uncertainty of some hypothesis :

- Whether the economic situation remains depressed or improves, our forecasts indicate the risk of very large surplus capacity in the future. According to the low hypothesis 1985 demand will only be at the level of 1982 supply.

- The supply and demand balance will not improve during the next few years. In the case of the hypothesis, annual surplus production could be 500 kt of metal per year while in the case of the low hypothesis this surplus production could be even higher and increase from now until 1982.

TABLE N° 37

REFINED COPPER SUPPLY AND DEMAND IN WESTERN WORLD

FROM PRESENT TO 1985

in kt Cu

	Copper Supply	Copper Demand	
		Low Hypothesis	High Hypothesis
1976		6 467	6 467
1977	7 354	6 880	6 880
1978	7 603	7 086	7 155
1979	7 909	7 299	7 441
1980	8 233	7 518	7 739
1981	8 381	7 744	8 049
1982	8 960	7 976	8 371
1983		8 125	8 706
1984		8 369	9 054
1985		8 620	9 416

It appears that severe restrictions should be placed upon copper mine development projects. This self-regulation could however be limited by the fact that the governments are to an increasing extent taking control of mines away from private companies so that if there is a fall in prices (due to an excess of supply over demand) there is no reduction in production but rather an increase so that the structural deficit in the balance of payments will not become too great. Most of the large copper exporting countries are developing countries whose revenues are basically linked to copper exports. Even under present market conditions it appears that while there are several large projects which have either recently started up or are on the point of doing so, others which have been planned for 1978 and following years will be delayed.

Thus the new mining production capacity which is to come into production between now and 1982 should be enough to satisfy demand up to 1984.

In the short term, it therefore appears that it will be difficult to restore refined copper supply and demand equilibrium in the Western World but this equilibrium might possibly be approached between 1980 and 1985 if some mining projects are delayed.

Let us take as an example a restrictive decision by the producer countries consisting in "freezing" several projects for a certain number of years in order to modify market terms. A scenario of this kind is not simply utopic and could become a possible policy for CIPEC or other producers.

In our scenario we suppose that the following projects are delayed for 5 years : La Caridad (Mexico), Andacollo (Chile), Santa Rosa (Peru), Gecamines Projects (Zaire). In view of their size, this amounts to reducing mining capacity by 140 kt in 1979, 430 kt in 1980, 430 kt in 1981, and 430 kt in 1982.

Supply and demand under these conditions are given in Table n° 38 and in the case of high concumption hypothesis there will be an equilibrium in 1980 and 1981.

It should thus be kept in mind that these forecasts are based on several hypothesis (forecasts of production, consumption, surplus of

TABLE N° 38

REFINED COPPER

SUPPLY AND DEMAND IN THE WESTERN WORLD

ONE OR SEVERAL PROJECTS ARE FROZEN

(in kt Cu)

	Copper Supply	Copper Demand	
		Low Hypothesis	High Hypothesis
1976		6 467	6 467
1977	7 354	6 880	6 880
1978	7 603	7 086	7 155
1979	7 796	7 299	7 441
1980	7 886	7 518	7 739
1981	8 034	7 744	8 049
1982	8 613	7 976	8 371
1983		8 125	8 706
1984		8 369	9 054
1985		8 620	9 416

exports for the socialist countries, etc.) some of which could be modified in the next few years and which will therefore have to be updated at the necessary time.

Thus an analysis of the political, technical and financial risks affecting the projects could suggest a certain degree of prudence in the middle and long term as concerns European supply.

9.2 Past Evolution and Present Situation of the Community

EEC mining production which comes essentially from Ireland is very low (< 13.1 kt in 1975) and covers less than 1 % of European needs for primary copper. Almost all primary copper is therefore imported in the form of concentrates, blister or refined copper (Table n°39).

In addition, scrap recovery makes it possible to cover 18 % of refined copper consumption and 35 % of copper consumption in all forms.

Thus more than 80 % of the copper consumption of the EEC must be imported. These importations are in the form of refined copper (about 72 %), blister (20 %) or concentrates (8 %).

The concentrates are used mainly in the German smelters and come from Papua New Guinea (almost half German imports of concentrates in 1975) as well as from Chile, South Africa, Norway, Indonesia and Canada (Table n°40). Before 1972 Chile was the principal exporter of concentrates to West Germany while Papua New Guinea was not at that time a copper producer.

Blister is imported by Belgium, West Germany, the United Kingdom and France from Zaire, South Africa, Chile and Peru.

All the EEC countries except for Belgium and to a lesser degree West Germany are highly dependent upon refined copper imports from Zambia, Chile, Zaire and Canada.

EEC refined copper consumption was 1954 kt in 1975 (against 2,171 kt in 1974) to which should be added 542 kt of scrap reused directly at the processing level i.e. 2,496 kt of copper in all.

The supply structure of the EEC has changed little during recent years. Refined copper imports continue to come from the same source (but the increasing size of imports from the socialist countries : USSR, Poland and Yugoslavia, should be mentionned. On the other hand, West Germany has diversified ist concentrate imports by buying from

TABLE N° 39

EEC COPPER SUPPLY SITUATION IN 1975

Imports and exports means imports to and exports form outside the EEC
in kt Cu

	EEC	W.Germany	Belgium-Luxembourg	Denmark	France	Ireland	Italy	Netherlands	United Kingdom
Mining Production	13.1	2.0			0.5	9.8	0.8		
Imports of concentrates*	607.5	563.1	44.0						0.3
Exports of concentrates*	45.8		4.1		1.7	37.4	2.6		
Blister Production	188.1	168.1	20.0						
Imports of Blister	392.8	117.7	195.7		19.2		2.0		58.2
Exports of Blister	2.8	1.1	1.1				0.2		0.3
Refined Copper Production	972.9	422.2	346.4		39.6		13.2		151.5
Refined Copper Imports	1 430.7	349.6	166.6		252.3	0.1	267.8	34.3	360.0
Exports of Refined Copper	131.3	72.1	48.1		0.1			4.3	6.6
Scrap reused in refineries	345.4								
Scrap reused directly	541.6	116.6	33.1		103.7		144.0	20.8	123.4
Refined Copper Consumption	1 954.0	634.6	174.2	2.7	365.4	0.5	290.0	37.0	450.5

* in kt of concentrates

TABLE N° 40

STRUCTURE OF EEC COPPER IMPORTS FROM OUTSIDE

Concentrates

	Imports of the Community	
	in kt of concentrates	in %
Norway	54.5	9.0
Canada	44.2	7.3
South Africa	65.0	10.7
Chile	66.6	11.0
Indonesia	52.0	8.6
Papua New Guinea	271.0	44.6
Others	54.2	8.8
TOTAL	607.5	100

Blister

	Imports of the Community	
	in kt of copper	in %
South Africa	84.4	21.5
Zaire	149.0	37.9
Peru	26.2	6.7
Chile	85.6	21.8
Others	47.6	12.1
TOTAL	392.8	100

Refined Copper

	Imports of the Community	
	in kt of Cu	in %
United States	94.1	6.6
Canada	159.8	11.2
Socialist Countries	114.2	8.0
Zaire	214.3	15.0
Zambia	360.9	25.2
Chile	281.1	19.6
Others	206.3	14.1
TOTAL	1 430.7	100

new producer countries such as Papua New Guinea which do not have smelters or refineries instead of the traditional raw material exporting countries which have developed processing industries.

The characteristics of the EEC supply is its high degree of dependence upon the developing countries of Africa and South America and this has not become less marked in recent years. The trend for imports from these countries has remained more or less constant while the mining production of the Community remained at a level which was low in relation to needs.

9.3 Future Perspectives for Community Supplies

Since the fall in prices in the second half of 1974, the market has favoured copper buyers. In addition to overproduction there also exist world copper stocks which were recently estimated at 2.2 Mt amounting to almost six months of world trade at the 1975 level.

On the hypothesis that economic growth will recover slowly, the forecasts indicate the possibility of a million additional tonnes from now to 1982. This figure may be reduced, but only in part, if consumption grows more rapidly.

We have seen that the EEC was principally an importer of refined copper and it therefore benefits from the present economic situation. Moreover, the West German smelters might develop supply problems in the near future as a result of the reduction in world trade in concentrates.

In this case, it may also be possible to find solutions at the company level by means of inter-European exchanges with the East-block countries, especially Poland. This point, however, indicates the importance of attempting to utilise copper deposits within the frontiers of the Community or of the European countries associated with it and of promoting the activities of European mining companies overseas. The effect could be an additional guarantee for the maintenance of certain currents of supply.

It can be supposed that the surplus situation of the copper market will necessarily lead to self-regulation measures. International discussions have already begun on this point and should theoretically be completed at the end of 1978 (UNCTAD). Whatever may be the character of the measures actually discussed and whichever may finally be adopted, the agreements will involve financial aid intended to encourage the producer countries to reduce exports and, temporarily at least, their production.

The effect of the low price levels is to discourage investments in new projects which are hardly attractive from the profitability point of view under a copper price estimated at 95 or $ 1.00 per pound of metal (value at beginning of 1977).

The combined effect of these factors could lead to a rapid modification in supply and demand relations and the reduction of stocks, a part of which might be incorporated into an international buffer stock or a series of coordinated national stocks.

The Community should therefore prepare itself for these changes. In addition to the purely quantitative aspect of supply and demand balance trends there is also concern as to the protection of the European copper processing industry from the point of view of international competition.

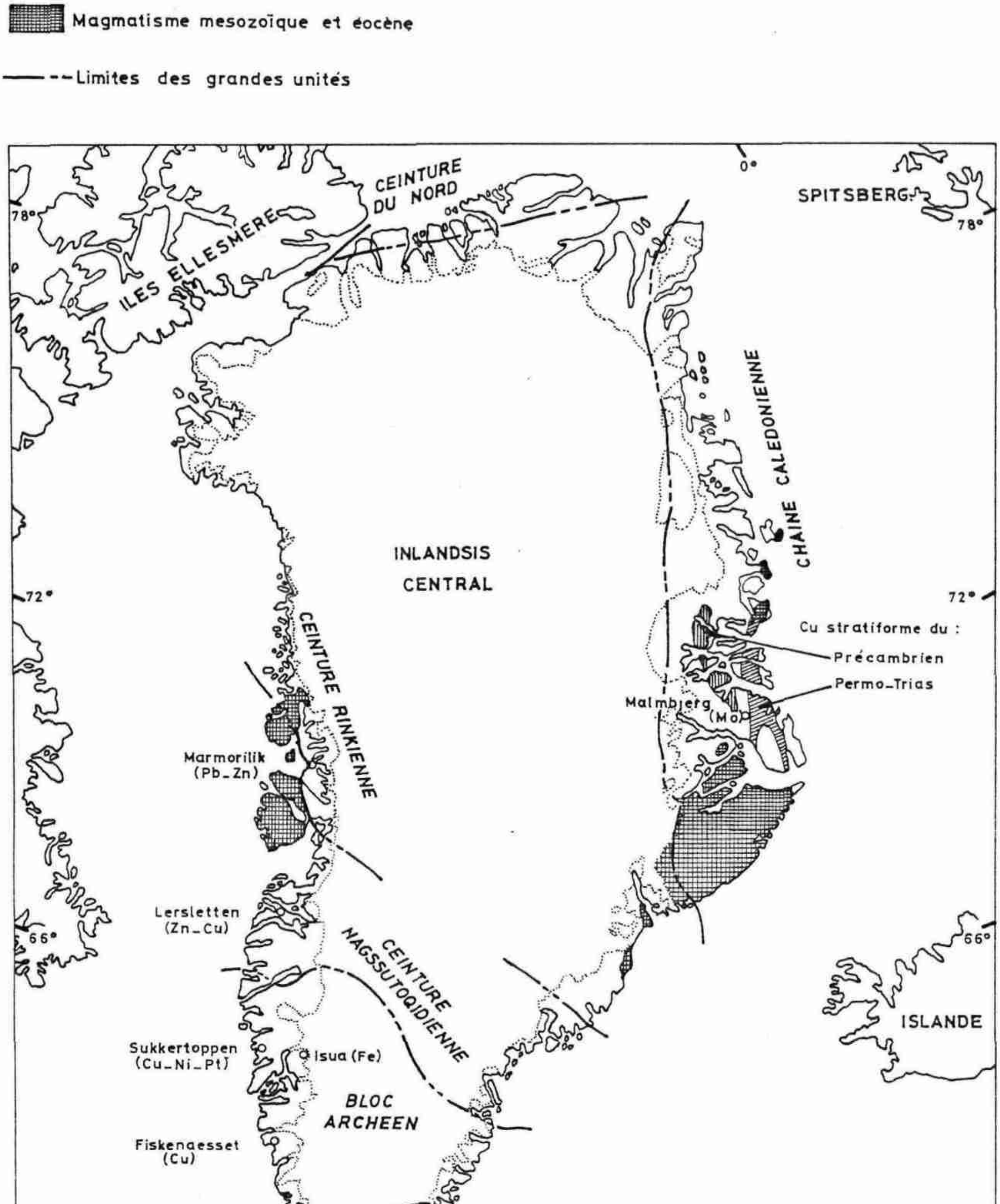
In no case can European industry remain exclusively dependent upon market conditions. Its direct competitors, that is North American and Japanese companies and the industries of the Socialist countries, under all circumstances will have good access to raw materials either on their own territory or in their zones of political and economic influence. This situation does not present any problems when the market is depressed but it is a heavy handicap when the market is tight.

It is therefore indispensable that the European Community take advantage of the present period of market surpluses and that it attempts to improve its positions at all stages of resource development.

Fig.1

GROENLAND

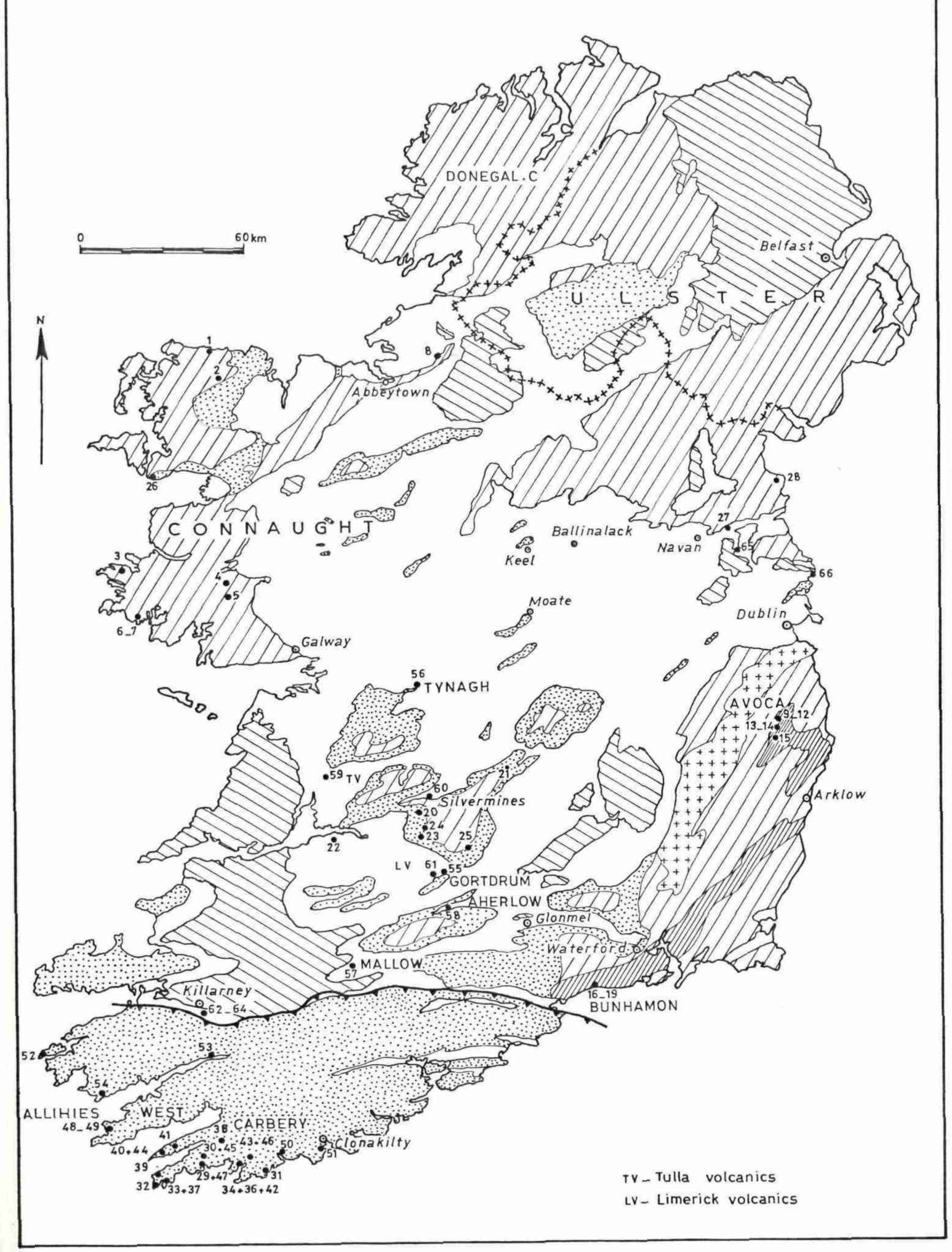
GRANDES UNITES GEOLOGIQUES ET PRINCIPALES MINERALISATIONS



D'après "Geologie of Greenland" _ (1976)

Fig.2

MINERALISATIONS CUPRIFERES D'IRLANDE



LEGENDE

"Mallow Thrust Front"

Formations postérieures au "Calcaire Carbonifère"

Formations du "Calcaire Carbonifère"

Dévonien et Carbonifère inférieur (ant^r au calcaire)

Formations antérieures au Dévonien

Granite de Leinster } du SE. de l'Irlande
Ceintures volcaniques }

AVOCA : Principaux districts ou mines de cuivre.

Arklow : Localités citées (dont principales mines Pb-Zn)

Liste des mines et indices de cuivre

1. Geevraun
2. Sralaghy
3. Cleggan
4. Dooros
5. Glawn
6. Truska
7. Errisbeg
8. Pollboy
9. Kilmacoo
10. Connary
11. Cronebane
12. Tigroney
13. Ballygahan
14. Ballymurtagh
15. Moneyteigue

(9.–15.) AVOCA

16. Knockmahon
17. Kilduane
18. Bunhamon
19. Tankardstown
20. Ballyhourigan
21. Rathnaveoge
22. Pallaskenry
23. Lackamore
24. Killeen
25. Hollyford
26. Bolinglanna
27. Beauparc
28. Salterstown
29. Ballycummisk
30. Ballydehob
31. Bawnishall
32. Boulysallagh
33. Brow Head
34. Cappagh
35. Coney Island
36. Coosheen
37. Crookhaven
38. Derrycarhoon
39. Dhurode
40. Glanallan
41. Gortavallig
42. Horse Island
43. Kilcoe
44. Kilcrohane
45. Mount Gabriel
46. Roaring Water
47. Skull
48. Allihies
49. Urhin
50. Glandore
51. Duneen
52. Valencia
53. Ardtully
54. Carrigcrohane
55. Gortdrum
56. Tynagh
57. Mallow
58. Aherlow
59. Ballyvergin
60. Silvermines
61. Oola
62. Ross Island
63. Crow Island
64. Muckross
65. Brownstown
66. Loughshinny.

D'APRES UN DOCUMENT INEDIT DU GEOLOGICAL SURVEY OF IRELAND.

Fig.2

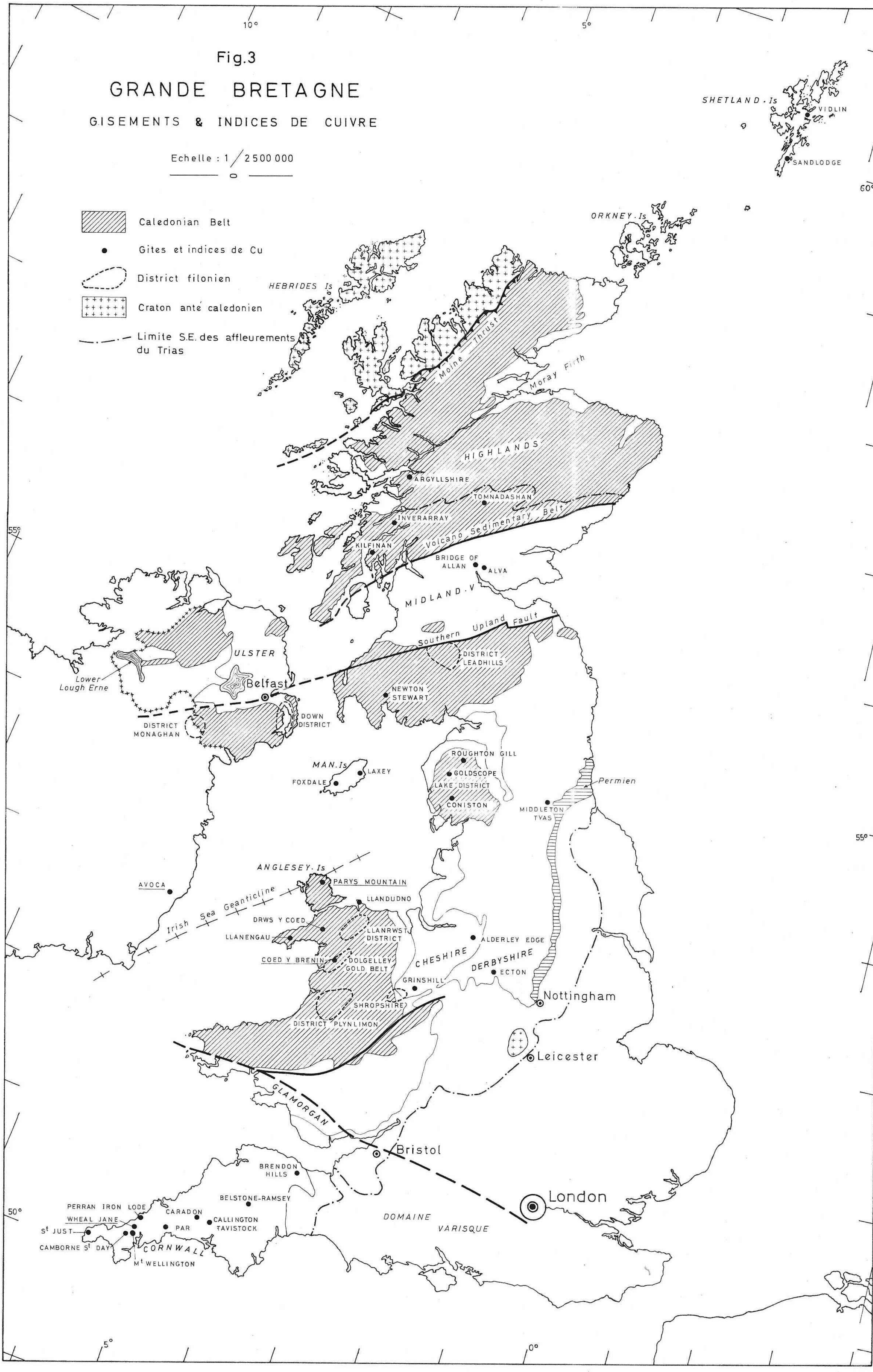
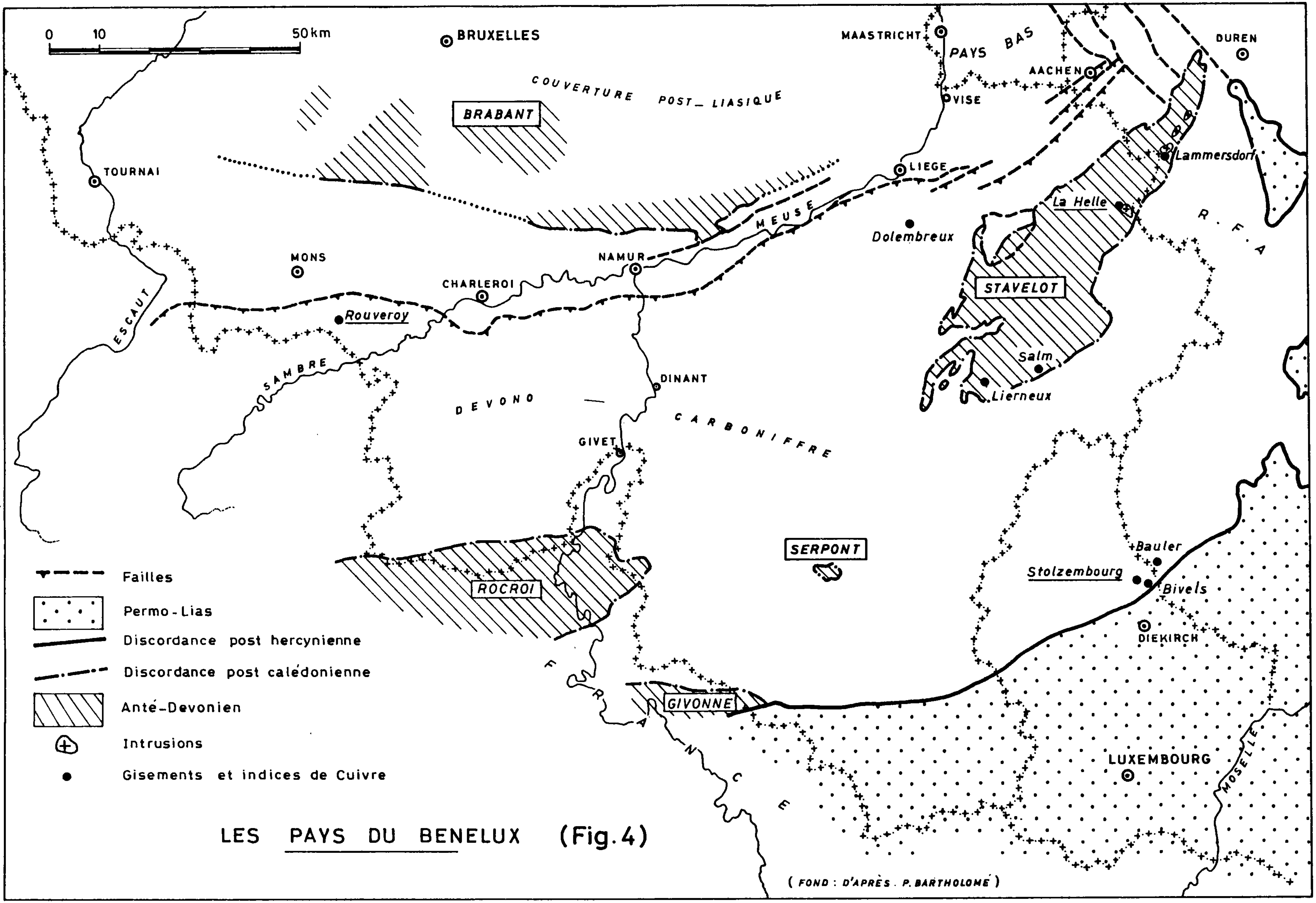


Fig.3
GRANDE BRETAGNE
GISEMENTS & INDICES DE CUIVRE
Echelle : 1/2 500 000
0
Caledonian Belt
Gites et indices de Cu
District filonien
Craton ante caledonien
Limite S.E. des affleurements du Trias
10°
5°
60°
55°
50°
0°
SHETLAND. Is
VIDLIN
SANDLODGE
ORKNEY. Is
HEBRIDES Is
Moine Thrust
Moray Firth
HIGHLANDS
ARGYLLSHIRE
TOMNADASHAN
INVERARRAY
Volcano Sedimentary Belt
KILFINAN
BRIDGE OF ALLAN
ALVA
MIDLAND. V
Southern Upland Fault
ULSTER
Lower Lough Erne
Belfast
DISTRICT LEADHILLS
NEWTON STEWART
DOWN DISTRICT
DISTRICT MONAGHAN
MAN. Is
LAXEY
FOXDALE
ROUGHTON GILL
GOLDSCOPE
LAKE DISTRICT
CONISTON
Permien
MIDDLETON TYAS
ANGLESEY. Is
PARYS MOUNTAIN
LLANDUDNO
AVOCA
Irish Sea Geanticline
DRWS Y COED
LLANRWST DISTRICT
LLANENGAU
ALDERLEY EDGE
COED Y BRENIN
DOLGELLEY GOLD BELT
CHESHIRE
DERBYSHIRE
ECTON
GRINSHILL
Nottingham
SHROPSHIRE
DISTRICT PLYNLIMON
Leicester
GLAMORGAN
Bristol
BRENDON HILLS
BELSTONE-RAMSEY
London
PERRAN IRON LODE
CARADON
WHEAL JANE
CALLINGTON
TAVISTOCK
S^t JUST
PAR
CAMBORNE S^t DAY
CORNWALL
M^t WELLINGTON
DOMAINE VARISQUE



LES PAYS DU BENELUX (Fig. 4)

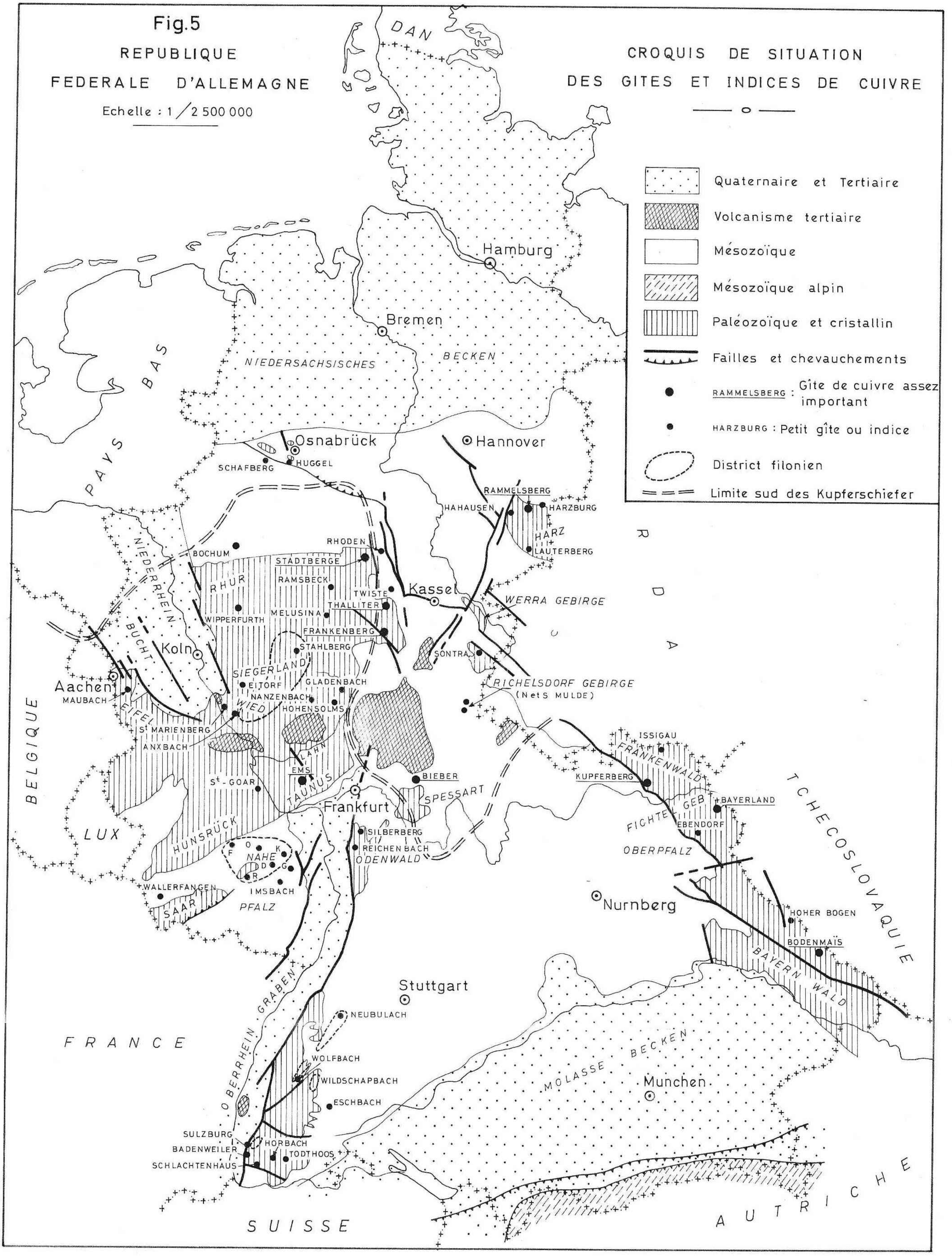


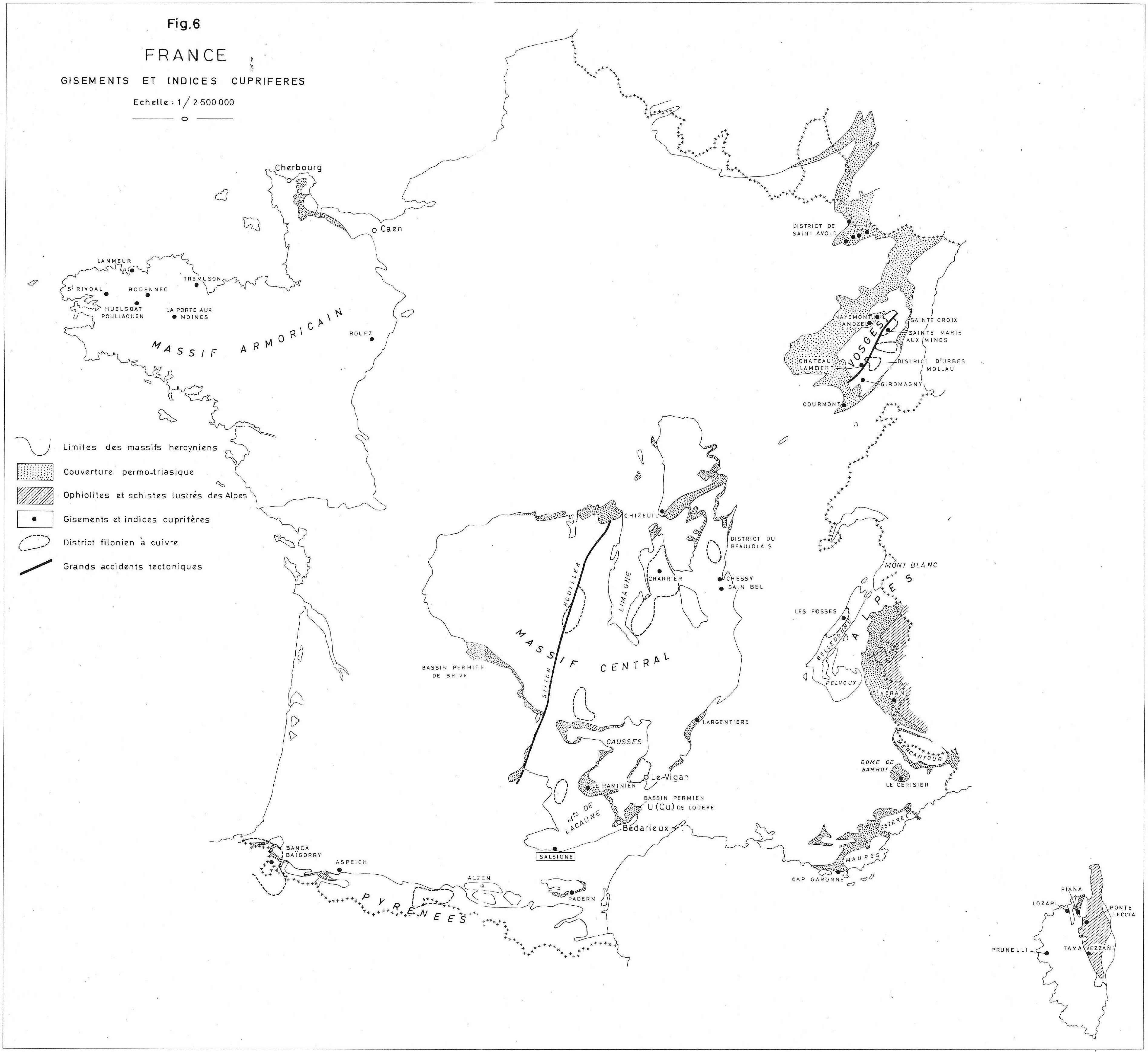
Fig.5
REPUBLIQUE
FEDERALE D'ALLEMAGNE
Echelle : 1/2 500 000
CROQUIS DE SITUATION
DES GITES ET INDICES DE CUIVRE
Quaternaire et Tertiaire
Volcanisme tertiaire
Mésozoïque
Mésozoïque alpin
Paléozoïque et cristallin
Failles et chevauchements
RAMMELSBERG : Gîte de cuivre assez important
HARZBURG : Petit gîte ou indice
District filonien
Limite sud des Kupferschiefer
DAN
Hamburg
Bremen
NIEDERSACHSISCHES
BECKEN
PAYS BAS
Osnabrück
SCHAFBERG
HUGGEL
Hannover
RAMMELSBERG
HAHAUSEN
HARZBURG
HARZ
LAUTERBERG
R D A
NIEDERRHEIN BUCHT
BOCHUM
RHODEN
STADTBERGE
RHUR
RAMSBECK
TWISTE
Kassel
THALLITER
WERRA GEBIRGE
WIPPERFURTH
MELUSINA
FRANKENBERG
Koln
STAHLBERG
SONTRA
SIEGERLAND
Aachen
MAUBACH
EIFEL
EITORF
GLADENBACH
NANZENBACH
HOHENSOLMS
WIED
RICHELSDORF GEBIRGE
(N et S MULDE)
St MARIENBERG
ANXBACH
LAHN
ISSIGAU
FRANKENWALD
BELGIQUE
EMS
BIEBER
KUPFERBERG
TAUNUS
St-GOAR
Frankfurt
SPESSART
FICHTE GEB
BAYERLAND
EBENDORF
TCHECOSLOVAQUIE
LUX
HUNSRÜCK
SILBERBERG
REICHENBACH
ODENWALD
OBERPFALZ
NAHE
WALLERFANGEN
IMSBACH
SAAR
PFALZ
Nurnberg
HOHER BOGEN
BODENMAÏS
BAYERN WALD
Stuttgart
NEUBULACH
OBERRHEIN GRABEN
FRANCE
WOLFBACH
WILDSCHAPBACH
MOLASSE BECKEN
Munchen
ESCHBACH
SULZBURG
BADENWEILER
HORBACH
TODTHOOS
SCHLACHTENHAUS
SUISSE
AUTRICHE

Fig.6

FRANCE

GISEMENTS ET INDICES CUPRIFERES

Echelle : 1/2 500 000



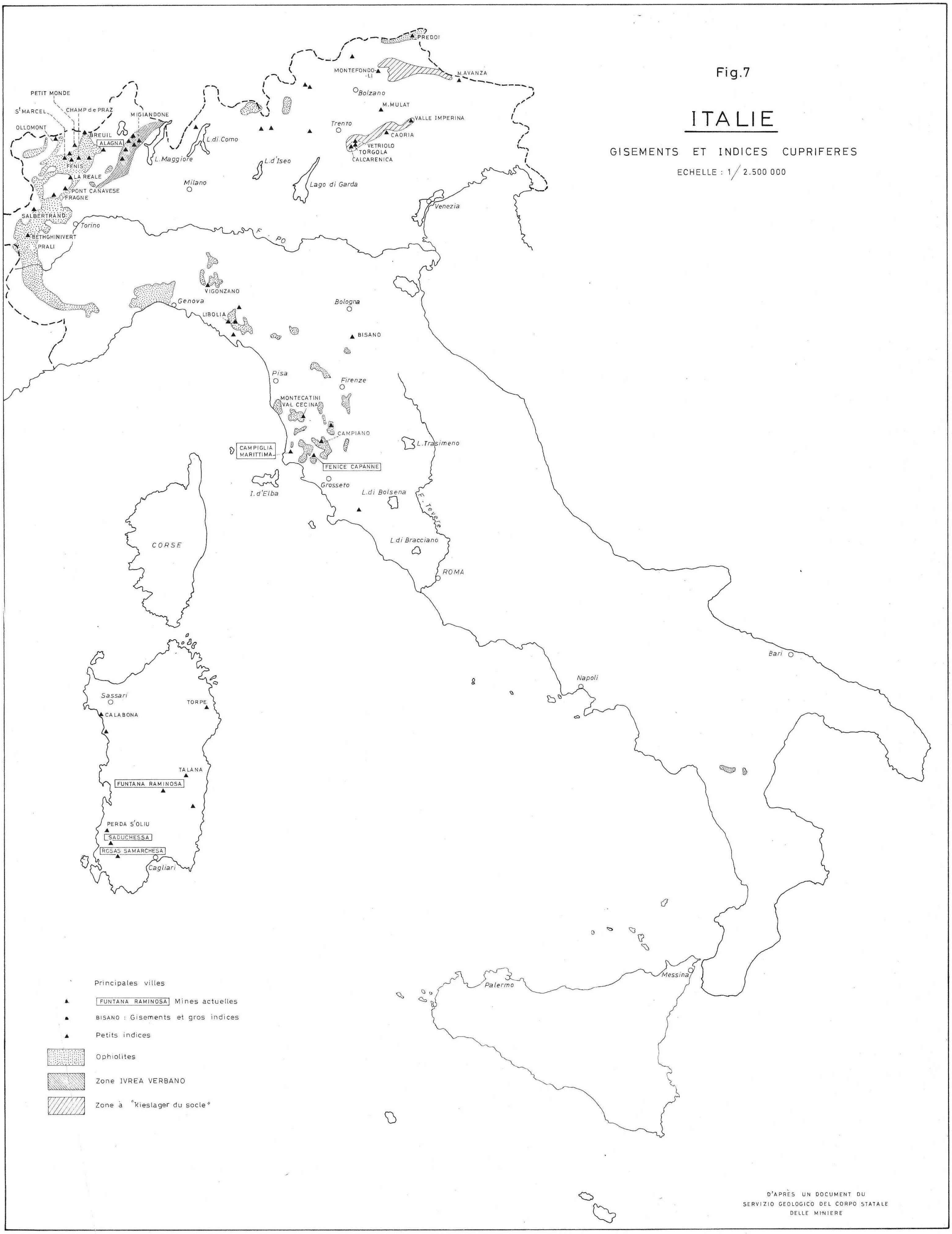


Fig.7

ITALIE

GISEMENTS ET INDICES CUPRIFERES

ECHELLE : 1/2.500 000

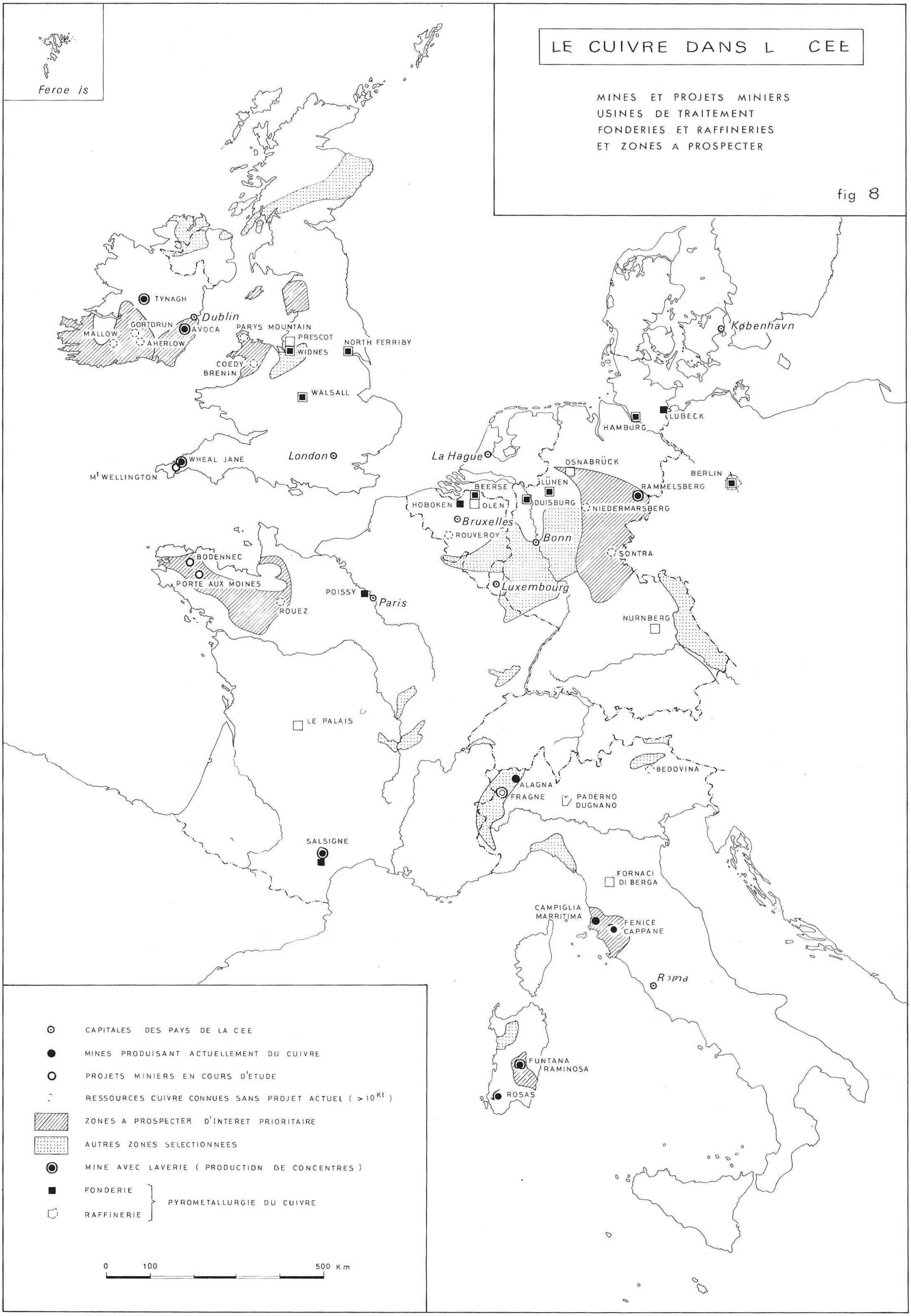

LE CUIVRE DANS L CEE
MINES ET PROJETS MINIERS
USINES DE TRAITEMENT
FONDERIES ET RAFFINERIES
ET ZONES A PROSPECTER
Feroe is
TYNAGH
Dublin
AVOCA
GORTDRUN
MALLOW
AHERLOW
PARYS MOUNTAIN
PRESCOT
WIDNES
NORTH FERRIBY
COEDY BRENIN
WALSALL
WHEAL JANE
Mt WELLINGTON
London
La Hague
HOBOKEN
BEERSE
OLEN
Bruxelles
ROUVEROY
DUISBURG
LÜNEN
OSNABRÜCK
HAMBURG
LUBECK
København
BERLIN
RAMMELSBERG
NIEDERMARSBERG
Bonn
SONTRA
Luxembourg
BODENNEC
PORTE AUX MOINES
ROUEZ
POISSY
Paris
NURNBERG
LE PALAIS
BEDOVINA
ALAGNA
FRAGNE
PADERNO DUGNANO
SALSIGNE
FORNACI DI BERGA
CAMPIGLIA MARRITIMA
FENICE CAPPANE
Roma
FUNTANA RAMINOSA
ROSAS
CAPITALES DES PAYS DE LA CEE
MINES PRODUISANT ACTUELLEMENT DU CUIVRE
PROJETS MINIERS EN COURS D'ETUDE
RESSOURCES CUIVRE CONNUES SANS PROJET ACTUEL (> 10 Kt)
ZONES A PROSPECTER D'INTERET PRIORITAIRE
AUTRES ZONES SELECTIONNEES
MINE AVEC LAVERIE (PRODUCTION DE CONCENTRES)
FONDERIE
RAFFINERIE
PYROMETALLURGIE DU CUIVRE
0
100
500 Km

fig 8

COMMISSION DES COMMUNAUTÉS EUROPÉENNES

200, rue de la Loi - BRUXELLES

COPPER DOSSIER

VOLUME 2

RECOMMENDATIONS FOR R. AND D. ACTIONS

BUREAU DE RECHERCHES GÉOLOGIQUES ET MINIÈRES
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(France)

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00198 Roma
(Italie)

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CONTENTS (Vol. 2)

RECOMMENDATIONS FOR R. AND D. ACTIONS

RECOMMENDATIONS FOR R. AND D. ACTIONS

1 GEOLOGY AND PROSPECTION

The conclusions (1.2.9.) of the first chapter concerning geological characteristics stress the need to increase prospection for copper in the EEC and the fact that there is a reasonable hope of new discoveries. A numerical estimate for the latter cannot be given but as an approximation it can be stated that in the next 10 years it may be possible to find 2 to 4 mines capable of producing 50 kt of metal/year which would be equivalent to 5 to 10 % of EEC consumption. Prospection, possibly together with improved scrap recovery, is the field in which the greatest improvements in the EEC copper balance might be made. Vigorous encouragement of this prospection would however be necessary.

1.1. Principal obstacles to Copper Prospection

These obstacles are of various types and some of them are common to all the base metals :

1.1.1 Psychological Obstacles

- Western Europe has the reputation of being poor in copper and this limits prospection activity by private companies. It is of course true that the Alpine chain and the Kupferschiefer deposits are richer in copper to the East than in the West but we have already seen that this reputation is only partly justified ;

- Many mining geologists believe that the EEC contains only small deposits. It is true that there are no large copper deposits ; but certain base metal mines, for example Navan and Rammelsberg, are large enough to have international importance.

It is therefore important that the mining profession become thoroughly convinced that there are reasonable possibilities of finding deposits of all sizes within the EEC.

1.1.2 Legal Obstacles

Mining legislation differs a great deal from an EEC country to another.

All of these legislations are, by some way, defectives in their formulations or in the manners in which they are applied. From this legal point of view, mining prospection can be encouraged by legislation which completely garantee the rights of the prospector, on areas large enough to be adapted to modern methods but with reasonable time limit. A full information must then be available on old concession holders and their rights and legislation has to avoid that some zones will not be prospected for long periods of time. From this point of view of mineral rights, it would be probably very helpful for exploration and mining EEC companies if the different legislations in the EEC states become more homogeneous. There shoud also be a guarantee that it will be possible to mine the deposit and that it will not be frozen for environmental reasons. As concerns this latter aspect, it is possible to take measures to conserve the environment and then to reconstitute it after the period of exploitation which today is rarely longer than 30 years. At the European scale it is perhaps easier to trace a framework for legislation which insure that environment will be preserved but, on the other hand, that mining companies will be able to really participate to the supply of copper, from mines situated in EEC.

1.1.3 Financial obstacles

It is an understatement to say that currently copper prices are very low. With these market conditions, private companies are not encouraged to invest in copper prospection. But we saw (1.2.9.2.) that exploration expenditures have to be not too strongly dependent from the market. Therefore, at the EEC scale like at the world scale, private companies cannot insure the necessary investment for raw materials supply and the states themselves spend more and more money in prospection.

1.1.4 Technical Obstacles

- The geology of Western Europe is very well known but geological mapping sometimes lags far behind because it does not take into account advances in the field. New geological maps for prospection purposes and therefore on a scale of not less than 1/50,000 are necessary. In addition existing up-to-date and high quality data are often scattered in many different publications while regional synthesis are often lacking and shoud be encouraged.

- As to prospection methods, surface techniques are often very efficient and not very expensive. The problem is principally to adapt them to each context and several years are often necessary to develop an effective prospection strategy. It is, however, necessary to improve deep prospection methods (geophysical methods and drilling) for in the EEC the best possibilities for discovering large tonnages lie in the prospection for concealed and deep-seated deposits. The latter points of a technical kind are further considered under 1.3.

1.2 Types of Deposits and Prospection Zones

1.2.1 Types of deposits

The introduction covering types of copper deposits and reserves throughout the world shows that 4 main types account for 93 % of past production and 96 % of reserves. In order of increasing size these are as follows :

- Ni-Cu deposits linked to ultramafic rock ;
- volcanosedimentary deposits ;
- Stratiform sedimentary deposits ;
- Porphyry copper and associated mineralizations.

Aside from Alpine ophiolites within the EEC there is only a limited quantity of ultrabasic rock. The Ni-Cu ores associated with the latter consist of occurences, the best of which in the Harz Mts. and the Ivrea-Verbano zone are not worked at the present time. Ophiolites in the Western Alps have been studied recently, have little sulphur and seem to be less encouraging than that further East in the Alpine Arc. In the overseas territories of the EEC the only known deposits of this type are in the Sukkertoppen region of Greenland and these are comparable to those in the Canadian Grenville province. Thus theoreticcaly, there is no hope of discovering large metal tonnages.

The possibilities of discovering economically feasible Ni-Cu deposits within the EEC are thus relatively slight and in any case only small quantities of copper would be produced as a byproduct of nickel.

There is no known porphyry copper in the West part of the Alpine chain and the disseminated Cu-Mo occurences in others geological environments of EEC i.E. Argyllshire (Scotland), Coed y Brenin (Wales), La Helle (Belgium), Chateau Lambert (Vosges) and Calabona (Sardinia) have grades too low to be mineable at the present time. The overseas territories of the EEC member countries are much more promising for this type of deposit. The existence of porphyry molydenum at Malmbjerg is encouraging for prospecting of porphyry copper linked to Eocene intrusions in Central Greenland and there are mineralizations of this type in the Solomon Islands whose economic importance has not yet been proven although the context is very favorable.

Prospection for porphyry copper shoud therefore be directed towards these two zones of the overseas territories. But it could be of some interest to study the possibilities of extraction by leaching in situ of copper contained in the known disseminated mineralizations of Western Europe.

There are many volcanosedimentary deposits in Europe and some of them are actually mined. There are also stratiform deposits and many occurences in sedimentary environments. Prospection within the EEC should therefore be directed towards these two kinds of deposits and the range of intermediate types. The general objectives should be as follows :

- Intense prospection of the volcanosedimentary series in the Brioverian, Caledonian, Variscan, and to a lesser extent, Alpine geosynclines ;
- Detailed study of Devonian and Permo-Triassic cover series.

More favorable zones can be selected in these two types of formations either because the copper potential has already been demonstrated or because the general geological conditions seem to be encouraging. It should be recalled that several years ago before the discovery of the volcanosedimentary deposits Brittany was considered to be particularly poor in copper.

The list of these types of deposits given above does not include the Sn-Cu deposits of Cornwall and the Cu Pb Zn deposits of Southern Tuscany. The former were the biggest producers of copper in the past and within the EEC area but there seem to be few copper reserves in this region which has been very intensely prospected. The case of Southern Tuscany is different and will be discussed hereunder.

1.2.2 Zones to be Prospected (Map. 8)

These are subdivided below into 3 groups :

- Volcanosedimentary series in Europe ;
- Sedimentary cover series in Europe ;
- Various zones in the overseas territories of the EEC.

Within each group, the districts are classified in order of decreasing interest according to data for past production, proved reserves, the extent to which the geological context is favorable and available information as to the intensity of prospection.

- Volcanosedimentary Series

- Brittany. This area has never produced copper but the proved reserves are from 200 to 250 kt in recently discovered polymetallic deposits (Zn, Pb, Ag, Cu). Known other occurences and geophysical and geochemical anomalies will be studied and drilled while prospection is currently very active.

- The Geanticlinal Axis of the Southern Caledonides (SE Ireland, North Wales). This district has already produced more than 200 kt and there are about 1 Mt in low grade pyrite-Cu deposits (Avoca, Parys Mountain, Coed y Brenin). The Cambro-Ordovician series of this district has priority as a target but it has already been actively prospected ;

- The Harz Mts. is also a favorable district with its 250 kt of past production and 17.5 kt of reserves at the Rammelsberg polymetallic deposit. The surface is however limited while intense and long continued prospection led principally to discoveries of Pb-Zn mines.

- The SE of Sardinia has produced 10 to 20 kt and there are proved reserves of the same size at Funtana Raminosa. Many other geochimical anomalies and occurences are known especially in the Upper Silurian which is a good guide but at the present time there is little active prospection ;

- The Rhine Schist Massif has produced around 150 kt of copper but this came from veins which are now almost entirely exhausted. Ramsbeck and Meggen have no copper but they show that sulphide bodies associated with volcanism exist. There was intense prospection in the past but this has certainly slowed down and it is no facilitated by the fact that many small concessions still exist ;

- Frankenwald-Bayernwald. The Ordovician in this district contains the old Kupferberg, Bayerland and Bodenmais mines which produced 20 to 30 kt and have reserves of the same order. Recent prospection is carried on but the potentialities of this zone are still not precisely known. The Northern Metamorphic Caledonides (Republic of Ireland, Ulster, Scotland, Shetland Is.) show negligible production and the reserves are very small. There are, however, occurences and the geological context is favorable while there has been little prospection in this zone except recently in the NE. The Dabradian basic volcanics and marble beds should be prospected in detail ;

- The Alpine ophiolites (Franco-Italian zone with "schistes lustrés" or "calcescisti con pietre verdi") have produced around 20 kt and the proved reserves are not more than 30 kt. There has been little prospection outside of the Fragne and Alagna areas and systematic (geological and geochemical stream sediment) studies could be made ;

- Appenine ophiolites (Italy) produced about 70 kt of Cu (Montecatini, Libolia) but there is actually none proved reserves. This zone is better understood than the Alps and the geological and deposit formation data can serve as prospection guides. The possibilities of discovering large deposits are also slight but one can hope to find orebodies similar to those exhausted (10-20kt Cu).

- The devono-dinantian of NE Massif Central and SE Vosges

Recent exploration in these volcano-sedimentary series have not led to the discovery of new ore deposits.However by their geological characteristics and the presence of many little old copper mines (Sain-Bel, Chessy, Chizeuil, Charrier etc...) These formations seems in France the most promising after Brittanny. Prospection for copper is still active and merit to be encouraged.

The other volcanosedimentary districts in France seem relatively not so promising for copper. The Pyrenees are principally a lead-zinc province and the special example of Salsigne in the South of the Massif Central remains an isolated one.

- The Basement of Central and Eastern Italian Alps contain some little polymetallic deposits, whose total past production is around 20-30 kt Cu. None reserves is known there but this area and the orebodies are not well known and scientific studies are necessary to orient the mining research.

- NW Sardinia Little stratabound oxidized copper mineralizations are known there which seem to be linked to a tertiary tuffaccous volcanic series. Their metallogenic characteristics and their eventual link with prophyry type mineralizations are not well known. But a recent geochemical prospection has shown copper anomalies which are still to be studied.

This brief review shows that for prospection of copper bearing massive sulphides (sensu lato), Brittany and the south caledonides geanticlinal axis are the districts with priority while the SE of Sardinia and the Rhine Schist Massif are of secondary interest and systematic prospection of the Alpine ophiolites could also be recommended.

The Sedimentary Cover Series in Europe

- The German kupferschiefer of Zechstein is definitely the priority target amongst the stratiform sedimentary deposits. Production has been small (around 40 kt) but there are 400 kt of proved reserves near Sontra and the kupferschiefer potential in West Germany is supposed to amount to a minimum of 12 Mt. It is necessary to make a general study of mining, metallogeny and geology of the deposits for the kupferschiefer to the East of the Rhine Schist Massif and this will make it possible to select the most promising sectors ;

- SW Ireland has Cu deposits in the Upper Devonian and Lower Carboniferous. The total production was about 70 kt while there are proved reserves of more than 100 kt. The entire province should be studied and prospected and there is a good possibility that new economically feasible discoveries will be made. Exploration work is actually carried on.

Two other districts have less favorable characteristics but prospection is nevertheless required since the previous studies were carried out some time ago and it is possible that new guides will be found :

- The Triassic in the center of the U.K. The known metal content is low but the Triassic evaporite boundary seems to be a good guide ;

- The Permian of the Nahe Basin seems to require a general geological study for the definition of prospection zones.

The other cover series with copper occurences are as follows :

- The Devonian (Old Red Sandstone) in NE Scotland ;
- The Devonian in the South of the Ardennes massif (Belgium, Luxembourg, W Germany) ;
- The Triassic of the Sarre and Lorraine ;
- The Permian basins of the South of the Massif Central (Brive, Lodève, etc.) ;
- The Permo-Triassic cover of the SE of France (Maures, Estérel, Mercantour).

Up to the present time prospection in these districts has not been very productive since the many known deposits are either very small or even occurences. It should however be noted that the geology and especially the paleogeography of these cover zones is not always very much known. In addition, prospection has often been limited to zones around known deposits and to shallow depths.

The Southern Tuscany

The polymetallic (Cu Pb Zn) ore deposits of southern Tuscany, some of which (Fenice Cappane, Campiglia) are currently mined, have produced 20-30 kt of copper and actually known reserves amount to 70 kt. The main guides for prospection seem to be their geometric association with acid intrusions and with faults limiting uplifted zones. More regional metallotects probably exist and, taking into account the very complex geological environment of Southern Tuscany, it appears that detailed geological, gitological and metallogical studies are necessary to orient prospection. New data will be available from Campiano, another ore deposit where a recent drill hole has shown the presence of high grade mixed sulphide mineralization.

Prospection zones in the Overseas Territories

- Greenland because of its surface and the variety of its geological contexts is clearly the principal objective for prospection especially with :

- The Kelitidian and Nassugtoquidian belts in the central West part.

- The porphyry copper linked to Eocene intrusions in the center of Greenland especially on the East coast ;

- The stratiform copper deposits of the Upper Precambrian and Permo-Triassic in the Caledonian chain in the Eastern central Greenland. The latter type seems to be the most favorable and justifies intense field and geochemical prospection. Exploration costs in Greenland are much higher than in Europe but they still remain low comparing with necessary investment for mining in any country. The latter does not seem prohibitive in Greenland because of the nearness of the sea.

- The Solomon Islands show a geological environment which is very favorable for porphyry copper and massive sulphide bodies linked to volcanism and both types have been found. The nearbye example of Bougainville shows that a limited surface is not necessarily an unfavorable factor and there is a good chance of finding mineable copper in these British islands ;

- In New Caledonia The NE part (Diahot valley) has a volcanosedimentary series with small copper deposits. This constitutes a prospection target and the possibility of finding volcanogenic mineralizations and porphyry copper type deposits should not be excluded in other areas of New Caledonia.

- Lastly, a study on the Lesser Antilles should be made jointly by the United Kingdom, France and the Netherlands so as to determine the possibilities of finding copper deposits in spite of the small surface of the islands.

A sustained prospection campaign over the next ten years for these selected zones of EEC countries would probably lead to the discovery of several copper mines provided that suitable and modern prospection methods are used.

1.3 Prospection Methods

1.3.1 The Importance of Continuity

We have already stressed the fact that one of the principal factors contributing to the success of prospection in a given district is perseverance. Without this continuity there is little progress in the knowledge on the geological and mineral deposit conditions in the district and in most cases the methodology of prospection cannot be well adapted to the special regional conditions.

All earth science specialists have become more conscious of the importance of the supply of raw materials since the world energy crisis. When there is continuous prospection in a given sector, the scientific institutions, universities and government geological surveys work together towards an improved knowledge of the region. Their studies profit from the new data supplied by mining prospection while their research can give the prospectors new ideas. Time is necessary for these exchanges of views and regional studies based on existing but often scattered data.

In addition, the efficiency of the prospection methods used is largely determined by regional characteristics such as the nature of the ores, the petrography of the country rock, the extent, thickness and character of the overburden, the extent to which the bedrock and deposits are altered, etc... Even in favorable cases, several years of prospection are necessary in order to develop effective prospection methods. The investment may therefore seem to be large but experience has shown that advanced techniques are quite profitable since they make it possible to detect deposits which had previouly been unknown.

Lastly, even luck which is not a negligible factor in prospection tends to favor the tenacious and attentive prospector rather than the hasty amateur.

1.3.2 Mining Prospection Methods and Strategy

Mining prospection therefore requires a great deal of time. Several phases can be distinguished in a prospection operation for a new district (cf. "The Phases of Mining Prospection", J.P. Dumas, Bull. BRGM, Sect. 11 N°3, 1976). The zone studied becomes smaller in each stage but the necessary time and investment increase.

Two phases are distinguished in strategic operations :

- First region is chosen in view of a theroretical estimate of its possibilities (economic, legal and geological contexts). A first selection of sectors is made on the basis of a complete documentary search, a brief geological field study and in some cases, photogeological studies. It is especially at this stage that good regional studies on geological and gitological[1] conditions are of great importance.

(1) Gitology : Geology of mineral deposits (P. Routhier).

- In these selected districts the so-called strategic methods are used for prospecting for points of interest i.e. geological field study and especially geochemical stream sediment and airbone geophysical campaigns.

The latter methods are used with increasing regularity but the systematic coverage of favorable sectors in the EEC is far from complete. In addition, use of the computer in the last few decades has led to great improvments in these methods. For example in geochemistry, the cost of collecting and analysing a sample has shown a much greater relative increase than that of analysis and interpretation. Multielement analyses (20 to 30 elements) are made in order to derive the maximum possible amount of data from the samples and these have been made possible by improvements in analytic techniques and additional possibilities for interpretation through computer processing. It has also been possible to use airbone electromagnetic methods even in heavily populated areas such as those of the EEC where there are many parasitic anomalies (railway, various kinds of distribution networks). For example, the Rouez deposit in Brittany was in fact discovered by this method.

In addition, in the geochemistry field, methodological studies now being carried on show that it is possible to improve anomaly definition by limiting analysis to selected phases (altered minerals and iron hydroxide phase).

With this method it is possible to use a less dense stream sediment sampling and thus to reduce the cost of strategic prospection. In relation to multielement analyses there is however an information lost and the method should only be used in prospection directed towards specific targets (for example base metal deposits).

Geochemical and geophysical strategic methods are in general well developed and recent improvements have made it possible to do a better selection amongst anomalies.

Tactical operations in zones selected as described above make it possible to evaluate the deposits discovered through detailed geological surveys dense grid soil geochemistry, geophysical ground research and, finally, drilling. These tactical methods are generally well developed. At this stage it would probably be of interest to develop percussion drilling methods which are much less expensive than diamond drilling and for an equal expenditure make it possible to test more anomalies or to improve the determination of target geometry.

Operations on very little areas are later carried out for the reconnaissance of the orebody and are based essentially on drilling or in some case on exploration mining. The costs are high and the geological techniques involved lay special emphasis on sampling quality. The deposit conditions are then studied in order to orient development work. Improved knowledge of the various types of deposits can thus be very useful but the greatest progress will be made through the improvement of core drilling techniques leading to a reduction in the drilling cost per meter. The recent example of the development by the IFP (Institut Français du Pétrole) of a drilling bit which considerably increases oil drilling speeds and the lifespan of the tool shows that there are possibilities for considerable improvements in this field.

Mining prospection shoud also follow the lead of oil prospection and collect more data from cores study and to increase use of logging.

This mining prospection pattern is applicable principally to sectors which are not well known or which do not have overburden problems. In the countries of the EEC it would be very profitable to exploit existing work especially in favorable districts. Data could be derived from civil engineering works and oil drilling which, in some cases, could be continued for mine prospection pruposes. In addition mines about to be closed could be used for drilling.

Surface geochemical studies encounter problems due to large allochthonous cover consisting basically of glacial formations and peat in the EEC countries. These obstacles can be handled at the tactical stage through depth sampling under the cover with the auger drill. Improvements in these shallow drilling techniques and costs would be very useful. Various methods have been tried at the strategic stage i.e. sampling and analyses of vegetation, geochemical studies of lakes whose muds are concentrating agents and analysis of concentrates selected from glacial formations. These methods are not operational and at best only relatively large areas can be selected (10-50 km2).

1.3.3 Problems Specific to Cu Deposits in the EEC

We have seen above that two principal types of deposits must be searched in the EEC. Their characteristics and contexts are quite different and therefore different prospection methods are used.

Volcanosedimentary Deposits

There has recently been a great deal of progress in the understanding of deposits associated with volcanism. Many deposits described in recent years are not of the classical massive sulphide body type of the Canadian Archean and the Kurokos Miocene in Japan which are located in volcanic piles and over vents.

There are in fact massive sulphide deposits located in a sedimentary environment associated with volcanism which is at a distance and/or discrete. These deposits are therefore proximal or distal in relation to the volcanic centers and/or in relation to the volcanic vents and up to the present time this concept had not been well defined. The quantitative extent of volcanism is thus not a general guide and the entire volcanosedimentary series should be prospected.

As to the nature of the metal content it is observed statistically that proximal deposits in basic series are richer in copper but those associated with acid volcanism are more numerous larger in size and polymetallic. In addition, deposits which are distal in relation to volcanism and vents are principally lead-zinc bearing. However the chemical character of volcanism favorable to mineral deposit formation has not been well established although basic manifestations are always found to be present.

Sulphide bodies stricto sensu are proximal deposits in volcanic piles and are located above the vents. They can be considered as foreign bodies in their environments. Around the deposit there is therefore generally no large scale geochemical halo in the rock. On the other hand, owing to their high sulphide-pyrite and pyrrhotite-contents and their definite boundaries, the orebodies can be clearly detected by geophysical methods (electromagnetic methods, self-potential and even gravimetry). Hydrothermalism is also indicated by different types o alterations and chemical sedimentary layers (chert, carbonate, chloritic rock) extending to various degrees beyond the sulphides zone and which can thus be used as prospection guides.

Deposits which are distal in relation to volcanism can be related to a volcanic vent and thus have characteristics which are close to the preceding ones. More often, however, they also distal in relation to the vent and are trapped in favorable sedimentary structures. Permanent geological study is necessary in order to detect and understand these structures especially as synsedimentary tectonics played an important role in the preparation of the trap (faults, synsedimentary faults, basin paleotopography, etc.). This essentially sedimentary environment explains why these deposits are surrounded by large geochemical halos of base metals and sometimes manganese. Geochemical study is therefore more useful for this type of deposit which is often less massive, more spread out, less rich in pyrite and therefore less accessible to geophysical methods.

In conclusion, one of the best means for effective prospection of volcanosedimentary series is the appropriate use, at all prospection phases, of geochemical and geophysical methods. A good drilling target consists of the superimposition of a geophysical anomaly (S.P. or INPUT) over a geochemical anomaly. The geological data which in the case of a sulphide body is used principally for zone choice and during drilling (Phases I and IV) should be applied throughout the prospection of a distal deposit in a more sedimentary environment.

Stratiform Sedimentary Deposits

These deposits are essentially determined by sedimentary traps. Precise paleogeographical reconstructions are therefore used beginning with the phase of zone selection. These reconstructions should be made on the basis of existing data i.e. classical geological studies and maps, geophysical maps (aeromagnetic, gravimetric studies, etc.), various types of drilling of a mining type as well as hydrogeological and oil types. It is sometimes difficult for the prospector to assemble some of this data.

In addition, the cover series have often been prospected only along a very narrow band on the basin boundaries and around the known surface occurences. This band which is accessible to prospection must be enlarged. In order to do so, people in charge of mining search sometimes ought to do geological drill-holes not directed towards mineralized targets but towards sedimentary traps. Now Deep-seated stratiform deposits will no doubt be detected only by this means. The discovery of large copper deposits of this type at Lublin (Poland) and Olympic Dam (Australia) shows that the big amount of necessary investment is in relation with the size of the deposits. Therefore studies on the paleogegraphy and the geology of sedimentary deposits shoud be encouraged.

It should be stressed that for copper the sedimentary deposits have much higher average contents than the other types of deposits. Considerable efforts have been made at the world scale to exploit low grade deposits. However an equal effort has not been made to develop more powerful techniques capable of detecting rich ore at greater depths. It is not certain that the choice made was the most economical if we take into account the present difficulties involved in the exploitation of certain large low grade porphyry copper deposits which require very large investments. Prospection for concealed and relatively deep-seated deposits especially sedimentary deposits should therefore be further developed.

In general, geochemical methods have certain advantages over geophysical techniques in that the cost is lower and it is a direct method reporting directly on the presence of ore while the cause of a geophysical anomaly often is not exactly known before drilling. This advantage seems to be greater in the case of sedimentary deposits because the frequent presence of a regional anomaly in the host-rock horizon and zonation can serve as guides towards an economically feasible deposit. However geophysical techniques have much greater penetrating powers and are the determining means for the discovery of deep deposits. Methodological studies applied to selected basin boundaries including several types of techniques especially seismic, gravimetric, magnetic and induced polarization means are necessary before the systematic prospection of favorable sectors is begun.

Up to the present time attempts by the geochemists to prospect at greater depths have been relatively unsuccessful. These efforts have been in two directions, either analysis of more mobile elements (water and gases) capable of indicating the presence of ore in depth or rock analysis connected with geochemical halos around the deposits(especially primary halos linked to hydrothermal alteration or to the original ore deposit zonation). Hydrogeochemistry can sometimes be effective but there are difficult, unsolved equilibrium and modeling problems involved in its systematic use. Gas geochemistry (soil sample degasification) has not yet given the expected results. On the other hand, rock geochemistry has already shown its effectiveness. It will probably be further developed but at the tactical stage and for a relatively small surface. The geochemical study of halos around ore deposits should therefore be further developed.

In summary we recommend the following for the development of copper prospection in the territories of the EEC :

- <u>Favorable conditions for exploration should be created</u> by demonstrating to the mining companies that there are sufficient chances of success and by modifying mining legislation in the direction of improved guarantees for mining rights and the promotion of the rapid development of the claims and concessions.

- <u>Improved use of actual geological data and current prospection methods including in particular the following</u> :

+ There should be adequate coordination between prospection and scientific research as concerns geological surveys, regional studies and the types of deposits studied ;

+ In the geochemical field, regional multielement prospection with computer processing should be developed ;

+ In the field of geophysical techniques, strategic airborne methods should be used more systematically;

+ In the drilling field, there should be more frequent use of percussion drilling and in diamond drilling, constant use of wireline core barrel. In addition data from cores and holes (logging) should be more effectively collected.

- Methodological studies on currently used techniques in order to improve :

+ In geochimistry, selection of phases analysed (stream sediment and soil) and the knowledge of deposit halos;

+ In geophysics, Selection of anomalies (electrical methods) especially in volcanosedimentary series (graphite problem) and definition of traps for deep-seated sedimentary deposits (magnetic, gravimetric and seismic methods) ;

+ Improve the efficiency of shallow drilling and core drilling equipment so as to reduce the costs of geochemical prospection under overburden and of deposit evaluation phase.

These recommandations are applicable to the following types of deposits and selected zones :

- Volcanosedimentary mineral deposits in Brittany, the Southern Caledonides (SE Ireland and N Wales), the SE of Sardinia, the Rhine schist massif, the Alpine zone of "schistes lustrés", the Appenines, the Post-Archean belts of Greenland and the New Caledonia ;

- The sedimentary deposits of the German Zechstein (kupferschiefer), the SW of Ireland, the Triassic of Cheshire, the Nahe basin.

- The Polymetallic deposits of Southern Tuscany

- The copper prophyries of the Solomon Islands.

In the field of primary raw materials, research and development work on techniques and methods will generally be a useful contribution to prospection. As concerns copper, the present ore production in the EEC is so low (0.4 % of consumption) that it seems necessary to directly encourage prospection by means of mining inventories, regional prospections etc. if the middle term objective is to be a significant increase in ore production (for example 5-10 % of EEC consumption in the nineties).

2 MINING

There are no problems specific to the copper production sector as concerns mining technology and its application in the member countries of the EEC. The problems encountered in the copper field are the same as those for all metal deposits and especially lead-zinc mines and identical approaches are used.

2.1 Characteristics of European mines

Although EEC copper mine production is now very low, it is possible to point out some characteristics which might prove to be common to present as well as future mines.

The most important of these characteristics are the depth and narrowness of the deposits. The method most generally used will thus inevitably be underground mining although there may be an earlier open pit phase.

Possible studies cover the following fields :

++ working methods : optimization of existing methods and study and development of new ones or adaptation of those used in other countries or for other substances ;

++ deposit configurations i.e. for a given method, which deposit characteristics are necessary for profitable development (size of reserves, depth, minimum vein size, acceptable grade, etc.) ;

++ techniques for simulation of exploitation ;

++ environmental problems, restoration of site, mine subsidence problems especially in urban areas ;

++ soil stability problems and consequences for safety conditions.

2.2 Working methods

2.2.1 Detailed study of documentation on mining methods

Since existing and future mines within the EEC will face similar problems it is important that studies on specific cases be made on the basis of all available data concerning underground mining methods for deposits of the vein type or with thin beds.

This body of data will be very useful since optimization of mining operations requires the methods and solutions already used at other mines be adopted. It is therefore necessary to periodically update studies on mining methods and solutions recommended for various problems.

This highly technical study should be coordinated with economic evaluations of the methods.

The study requires visits to the principal mining countries where there are similar mines. Since it is necessary to update the data, the initial work which will probably take more than a year should be then continued by further work on a permanent basis.

This work would cover much the same area as for other mineral substances.

2.2.2 Optimization of mining equipment and increased mechanization

Because of deposit characteristics, it is necessary to use low capacity equipment. What are the present or future limits on the size of this equipment ? In view of the costs in the EEC especially for labor it can be supposed that increased mechanization will continue to be the main objective of the mining companies. What kind of solutions can be proposed in this field and what kind of progress can be expected ?

The study can be made either for the specific case of a given mine or at the level of a more general study of future possibilities. It will also be necessary to take the economic factor into account.

The method used for this study requires that many contacts be made with mine operators as well as with manufacturers of mining equipment.

2.3 Necessary dimensions and configurations for mines

Supposing that a deposit is of the vein type, one of the technical and economic problems concerns the dilution of the ore. The first question is the acceptable ore grade at the concentrator which depends upon whether or not the ore can be preconcentrated. The acceptable ore grade is thus defined by the ore dressing.

Once the ore has been defined for several typical cases and on the supposition that existing methods and equipment are used, study will make it possible to determine the cost per tonne extracted and then processed on the basis of various hypothetical conditions. Acceptable deposit configurations would thus be defined. For example, minimum vein thickness would be determined in terms of the parameters specific to each type of deposit.

2.4 Simulation of mining operations

In recent years computerized data processing has been widely developed for the preliminary study of mines especially for resource and reserve evaluation by geostatistical methods. With the basic principle of simulating the deposit, the method has several other possible applications. The following two lines of study seem to be of interest :

2.4.1 Optimal open-pit design

While most deposits can be best mined by underground methods, open-pit operations are possible in some cases for all or part of the mine. The object of the study would be to develop a data processing programme for pit design as a function of various cut-off grades and technical constraints.

2.4.2 Simulation of mine exploitation

On the basis of a set of drillings on a given grid, it is possible to simulate them on a closely spaced grid so as to compute mine block estimates. By simulation it is possible to determine grade variations at the ore dressing plant input for various mining methods.

The optimal number of work sites can then be computed.

It should be noted that a simulated deposit has the same average characteristics (average grade-tonnage) and the same dispersion characteristics (grade fluctuation whithin the orebody) as the real deposit.

This study will include a theoretical phase for the development of programmes for application to existing of future exploitations in the EEC.

2.5 Environmental problems

There may be environmental problems in connection with the operations of some mines. In the future it will be increasingly necessary to take these constraints into account and the cost could become a large part of the investment or operating costs.

In the case of open-pit mines, concern as to the preservation of the regional environment could interfere with the opening of this type of exploitation in certain locations. It may be possible to make studies intended to find solutions for the resulting nuisances. A case by case approach seems to be the most suitable one in this field. When exploitation of the pits has been completed, there is the problem of restoring or redeveloping the site and the surrounding area. General recommendations are possible but, here again, a case by case approach is also preferable.

In the case of underground mines the method used is directly linked to environmental constraints. In most cases caving cannot be used in mines in urban zones because of the resulting subsidence. There are very few exploitations to which these constraints will not apply in the EEC context with its dense urban and industrial tissue.

Lastly, in general and for all mining methods, pollution or contamination may occur especially in the water sector.

2.6 Soil stability and safety problems

There are soil stability problems both in the case of open-pit and underground mines. It is the responsibility of the mine operators to study optimum stability conditions in association with rock mechanics specialists.

The legitimate and increasing concern with mine safety requires a permanent supply of information on the techniques used and their improvements. This is the necessary preliminary to the solution of safety problems.

The distribution of technical information and the execution of certain model projects could be encouraged at the EEC level. Work in this field could also be extended to the other mineral substances.

2.7 Other proposals

Other research fields could also be selected depending upon the specific interest in each case for present or future exploitations in the Community :

++ maximum recovery of pillars in underground mines ;

++ optimal utilization of tailings to be used either as fill or to be stored for the later recovery of by-products when the latter are not of immediate economic interest ;

++ closing conditions for mines : further study of the deposit, possibility of deep drilling, in certain cases maintenance of the mine, etc.

2.8 Conclusions

There are few copper producing mines in the EEC at the present time and these are not being studied on a large scale from the mining point of view. Possibilities associated with discoveries or known occurences do however exist and this has to encourage analyses of their exploitation under optimal technical conditions. The specific characteristics of most of the potential EEC deposits (depth, low tonnage, etc.) make it necessary to carry out further studies on optimal development conditions. Without this research the European deposits because of their special difficulties either might not reach the exploitation stage or could not guarantee sufficient profitability to the mining companies.

3 ORE PROCESSING

There appear to be three main research and development targets for the improvement of EEC copper supply conditions :

++ improvement of the efficiency of existing concentration plants in Europe ;

++ improvement of the ore processing techniques for application to potential ore deposit discoveries in Europe ;

++ acquisition of knowledge on the mineral processing of ores presenting special difficulties so as to facilitate Community participation in the exploitation of deposits with these types of ores.

3.1 Improvement of the efficiency of existing concentration plants in Europe

The efficiency of existing concentration plants in Europe can be increased essentially through improved beneficiation of the copper of complex ores in which it may be associated with iron, lead, zinc, arsenic and antimony sulphides and which may also show fine or interlaced crystallisation.

These characteristics lead to low extraction efficiency at the mineral processing level while the metalworking plants impose penalties for undesirable elements. Thus if we take into account losses at the ore dressing level, smelting and refining costs and the various penalties (with transport costs excluded), the payment received by the processor for copper concentrate produced from a complex ore and sold to a metal-working plant amounts to less than 50 % of the value of the copper content. It can even be only 10-20 % of the latter or even less (for example at Rammelsberg, Tynagh and Fenice Capanne). This proportion is generally 60-70 % in the case of monometallic ores.

The possible means for improving the efficiency of existing plants include the following :

++ development of homogenisation processes for supplying mineral dressing plants for relatively short periods (about 24 hours) which can be set up with comparatively small investments without degradation of the physico-chemical characteristics of the ore. (Equipment exists for this purpose but it is adapted to the larger tonnages handled in iron industry) ;

++ development of continuous analysis processes adapted to plants of small capacity ;

++ development of new and more powerful reagents and new selective reagents for improved preconcentration by flotation (collection of coarse grains containing little sulphide) and more highly purified concentration products. In order to be effective, this research requires close collaboration between metalworking plants, mineral processing plants and research centres so that technical and economic guidelines can be defined and the work carried out in accordance with the latter. In particular, it is of importance to increase the available knowledge concerning constraints due to undesirable constituants at the metalworking stage and on the factors entering into the calculation of smelting and refining costs.

It is, however, doubtful whether this kind of research on flotation techniques will lead to substantial improvements and at the present time few other techniques are applicable. Hydrometallurgical recovery methods for mixed polymetallic concentrates therefore seem to be of interest.

Even if this research does not lead to industrial operations for deposits now being worked with small reserves (except perhaps for Rammelsberg where new ore dressing techniques could increase the workable ore tonnage), it might improve the operating conditions for potential deposits.

In addition to the obvious impact on copper recovery, the success of this kind of operation could also make it possible to set up hydrometallurgical plants near copper deposits. (At Fragne the transport of the concentrate produced accounts for about a quarter of the sales price.)

This is true because from an economic point of view, the minimum capacity for a hydrometallurgical plant is much lower than for a pyrometallurgical plant (5,000 to 30,000 t/y of copper against 100,000 t and more).

In any case, copper recovered from mines located within the EEC accounts for less thant 0.5 % of Community consumption and an improvement in the efficiency of ore processing plants supplied by existing mines would not significantly increase the Community's self-supply rate.

It would however constitute useful technical progress from the point of view of increasing efficiency for the processing of ore from newly discovered and developed deposits.

3.2 Development of improved techniques for application to newly discovered mines in the EEC

In the light of the present state of geological knowledge concerning the territory of the EEC, there appear to be possibilities essentially for the discovery of two types of deposits i.e. volcanosedimentary and stratiform. There does not seem to be any possibility of finding porphyry copper deposits with grades high enough to justify development with conventional methods or economically feasible oxide ore deposits.

In the volcanosedimentary group (Brittany province type deposits such as Bodennec, Port aux Moines, Rouez, etc.),there are possibilities of finding polymetallic sulphide ores (Pb, Zn, Cu, Ag) often rich in pyrite. In the stratiform deposit group, the ore may show crystallisation of varying degrees of fineness and the "Kupferschiefer" type deposits constitute an extreme case (copper grade 2-3 %, carbonate content about 40 %, high carbon content, ore deposit thickness 40-60 cm and fine mesh liberation size). In the stratiform group there are also possibilities of finding more coarsely crystallised ores but with complex associations (sulfide-arsenides and antimonides, presence of gold), for example in the SW of Ireland.

The research subjects concerning the improvement of techniques applicable to potential deposits are thus largely the same as those for the improvement of existing plants :

- improved selectivity and recovery for the physical processing of finely crystallised polymetallic ores ;
- development of hydrometallurgical or mixed techniques for the processing of complex polymetallic concentrates or ores and, amongst the cases of special interest, can be mentioned kupferschiefer type deposits, ores containing sulfo salts and pyritic orebodies.

We also believe that it would be useful to prepare for the working of copper deposits by in situ leaching through further research in this field.

In addition to the fact that it is applicalbe to low grade deposits often found in the neighborhood of mines which have already been developed, this process has the advantage of minimising the environmental disturbances caused by open pit mining. This is an important factor in Europe with its high population densities. There is, however, certainly a danger that groundwater will be contaminated and research should be carried out to determine under what conditions pollution can be controled.

3.3 Conclusion

The development of the set of techniques described above which are applicable to ores usually considered hard to process, would allow the Community to participate in mining and metallurgical operations in other countries in exchange for a contribution at the technical level and thus increase Community control over its copper supply.

4 METALLURGY

It is our opinion that the following will be the principal constraints affecting copper metallurgy in Europe :

- concentrates and raw materials will probably be of lower grade and more impure and complex. It seems logical to suppose that the mining countries will continue their efforts towards further integration of the mining and metallurgical processing sectors and will keep the richer and more easily treated concentrates for extracting by proven conventional methods.

- fossil fuel energy such as oil, gas and coal will be more expensive and increasingly rare.

- labor will be increasingly expensive and it will be more and more difficult to find manpower for the heavier work.

- plants with old equipment and low unit capacity will be in an unfavorable position.

- anti-pollution measures will be applied more and more strictly and from the anti-pollution point of view hydro-or pyrometallurgical method can be considered altogether satisfactory. Pollution problems will continue to exist.

On the basis of these factors the following research and development guidelines can be proposed.

4.1 Research on flexible metallurgical treatment flow-sheets so as to

4.2 Research on energy in the following directions :

4.2.1 Continued development of autonomouns thermal processes with maximum recovery and the use of oxygen.

4.2.2 Research on processes with low energy consumption.

4.2.3 Studies on the possibilities of coupling chemical and metallurgical processes with exchange of energy within the complexes formed.

4.2.4 Adaptation of nuclear energy in order to use heat developed in the reactors either directly or by transduction of this form of energy to metallurgical plants by means of an intermediate fuel.

4.3 Basic metallurgical reserach on processes in order to achieve the following :

4.3.1 Reduction of operational sequences.

4.3.2 Increases in the specific capacity of reactors.

4.3.3 Increases in the absolute capacity of production lines (here production line means the series of basic machines with plant capacity being equal to the sum of parallel production lines).

4.3.4 Study of additional operations intended to recycle the intermediate products of the processes so as to reduce and, if possible, eliminate recycling.

4.4 Improved analytic control and computerization of metallurgical plant operations

4.4.1 Standardization of laboratory analysis methods.

4.4.2 Computerization of analyses for smelting plants.

4.4.3 Development of on-line computerized metallurgical plants.

4.5 Pollution

4.5.1 Research intended to improve anti-pollution measures concerning specific substances for existing processes (this work to be carried out by the plants).

4.5.2 In the case of new processes, the entire metallurgical cycle including effluents should be taken into account.

4.5.3 Development of new techniques for the treatment of residues from ore processing plants.

4.6 Technological research and development

4.6.1 Development of computerized techniques for repetitive operations so as to reduce the number of personnel.

4.6.2 Research on furnace construction materials so as to increase their lifetime when used in intensive processes thus facilitating application of the latter methods.

4.7 Research on more efficient use of raw materials and recycling (see chapter on recycling).

5 RECYCLING

As the study carried out in the Dossier has pointed out, large losses occur in the recycling in respect of potential obsolete copper products, in spite of the outstanding properties of copper, its chief use in massive form (dispersive uses are extremely low), and a rather high price of the metal.

The present structure of the recycling industry, the technologies applied, the basic assumption that the recycling operation must be a profitable one, and objective difficulties in reclaiming some types of obsolete products are responsible for the present low recycling rate of old scrap.

An additional limitation is represented by low prices copper in a period of crisis as is the present economic situation.

On the other hand, the potential of obsolete products in the EEC countries, where about a third of the Western world's total copper consumption is concentrated, is important. In fact, the copper contained in products entering the EEC market, which should become obsolete by 1985/1990, is estimated to total over 2 million tons.

The extension of recycling to marginal scrap to allow more materials to be recycled, with the purpose to limit as much as possible the dependance on foreign raw materials supply, could prove, within limits, to be of interest to the European Community.

Therefore, it should be advisable to undertake a joint and concerted examination of the problem by government bodies who are responsible for the environment protection policies, preservation of resources, and raw materials supply policies ; associations of economic operators in the field of recovery and recycling ; experts in the recycling industry.

The problem of extending recycling should be tackled comprehensively by removing impediments of any nature and by creating some forms of incentive and promotion ; and by the realization of research and development programmes to improve collection and processing methods to ameliorate economies in recycling.

Some suggestions have been included in chapter 5.3.5.

A study has been undertaken to determine the potential of recovery of used materials, on the basis of metal content of finished products, reviewing the correct technology of recovery (for aluminium, copper, lead, zinc and tin), by ITE, SGM and CERIMET.

In particular, a survey has been carried out by SGM of the state of the art in copper recycling, concentrating on electrical engineering, mechanical engineering, and building materials.

Most of the recommendations for R and D actions herein considered refer to problem areas which need more detailed investigation as developed in a joint document by ITE in co-operation with the co-pilots SGM and CERIMET.

It is however necessary to state in advance that a study should be conducted with the purpose to define exactly the scrap resources in the EEC, in what conditions they occur, and their potential in recycling ; to carry out preliminary studies to find out if sophisticated technologies are applicable in the collection and reclamation of obsolete copper products from building materials.

The main aspects for R and D actions of increase recycling are :

. To make the scrap more homogeneous new techniques should be developed, although some new processes, like cable chopping or car shredding, are already applied on a large scale. Important progess is still to be made to generalize such methods to mixed scrap.

This outline reflects the general principle of a greater convenience when scrap is directly used rather than processed in a copper refinery.

By contrast, a less selective sorting could offer the possibility to extend recycling to more highly diversified scrap and mixed scrap, considering the prospects of application of technologies in scrap sorting which are to be finalized through systematic studies, and the prospects of pyrometallurgy and hydrometallurgy.

. A type of scrap to which more research work should be applied is low-grade "irony" scrap, such as electric motor scrap or copper-steel residues from car shredders.

. Cryogenic grinding is a new way of processing some types of scrap, but its economics is still uncertain, being too dependent on the price of the cooling agent. An estimation of the conditions under which such operations are profitable could help assessing the future of this method ; scrap and copper prices, choice and price of the cooling agent, size of the cryogenic plant, etc. should be taken into account.

. For secondary copper smelters and alloyers, a reduction of the treatment costs depends primarily on furnace operations. Methods like oxygen enrichment of the air, air preheating, natural gas poling, are to be studied further to be applied on a large scale.

. The improvement in the recovery of by-products from copper scrap and of the proportion of copper recycled from the converter to the blast-furnace as a slag should also be increased.

. Electrolytic refining should also be the subject for research on the way of treating anodes produced from very impure blister copper or complex scrap. In some cases, this might allow to avoid either a preliminary refining, or the production of second-quality copper that is more difficult to utilize.

6 ECONOMIC AND ECONOMETRIC RESEARCH TOPICS RELATED TO EEC COPPER SUPPLY

Adequate knowledge of the economic factors determining the copper market must be available if the EEC is to increase its autonomy in the field of copper supply.

Two main guidelines are therefore recommended for economic or econometric studies :

6.1 Further study of the economic factors determining mine exploitation throughout the world.

The economic viability of copper mine operations is not only a function of market variations and in particular of metal price fluctuations. It also depends upon the specific characteristics of the mine in comparison with those of competitive exploitations.

It is therefore recommended that a file be set up indicating the economic operating conditions for several hundred copper mines throughout the world and that it be continuously updated. The factors to be taken into account should concern not only the geological and technical data but also economic and financial elements such as investments, rules of depreciation and the various operating costs, etc...

6.2 Further study of the world market as well as econometric simulation.

In attempting to improve their supply conditions, the European industries are faced with the following two contradictory conditions :

- as manufacturing industries, they prefer the primary metal price to be relatively low or at least competetive with that paid by their competitors benefitting from upstream integration ;

- as extractive industries, their interest is to have copper prices high enough to allow the exploitation of marginal copper deposits in Europe itself or to promote investments in mines located overseas.

The European Economic Community should undertake simulation studies leading to improved evaluations of the technical and economic consequence of various policy choices concerning purchasing, mine investments, scrap recovery, the creation of stocks, etc. in terms of general economic tendancies and the fluctuations specific to the copper market.