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F I N A L R E P O R T

ANALYSIS OF THE GLOBAL RESOURCES OF THE METALS OF THE PLATINUM GROUP, CHROMIUM, TIN, FLUORINE, PHOSPHORUS AND LEAD.

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FINAL REPORT ON THE ANALYSIS OF THE GLOBAL RESOURCES OF THE METALS OF THE PLATINUM GROUP, CHROMIUM, TIN, FLUORINE, PHOSPHORUS AND LEAD.

Summary and Conclusions

The aim of this study was to infer on the basis of the ≠'Mimic' model (1,2), irrespective of social or political considerations, the physically available potential reserves and long term price trends for the metals of the platinum group, chromium, tin, fluorine, phosphorus and lead. Furthermore, the concentrations ranges for these elements in the natural environment as inferred by the Mimic model, were to be determined in order to provide a basis for the distinction between normal concentrations and man-made pollution.

For this purpose a critical examination has been made of the different parameters entering into the 'Mimic' calculations and their significance for the availability and future development of such resources into reserves.

The classification and terminology used by the Commission for the estimation of mineral resources (3,4) is discussed and compared with the recently published classification of the US Geological Survey and US Bureau of Mines (5). The classifications are illustrated in figures 1 and 2.

The external parameters required by the 'Mimic' Model for the estimation of mineral resources have been selected from available literature and from additional, unpublished data submitted by the Commission ; complementary information has been obtained through consultation with expert colleagues.

The research work performed under this contract has been executed by IRC - Alkmaar and computer runs with an improved Fortran IV vèrsion of the 'Mimic'model (6) were made on the IRC - terminal in the Honeywell-Bull-General Electric Mark III time-sharing system.

≠ Mimic = Mining Industry Model for the Inventorization and Cost evaluation of mineral resources.

Some of the more important results from these studies can be summarised as follows :

1. The average concentrations of the elements in the Earth's crust (known in geochemical literature as "clarkes"), in combination with the specific mineralizabilities as calculated from the demonstrated reserves and ore deposits of the studied elements indicate "target" prices which for all 6 elements fall well within the empirical range of 0.77 - 1.30 times their historical (until 1971) long-term average prices, as illustrated by figure 3 and table 1.
The relation between clarke,specific mineralizability and average long term price had first been established for the elements copper, zinc, lead and gold and used to estimate the most likely long term price for uranium in constant currency value (1,2). The applicability of this relation has subsequently been extended for the elements manganese, molybdenum, antimony and mercury (7,8) and includes also the elements of the present study.

   This observation strongly supports our opinion that the average price ratios between different raw materials constitute a valid reference set of values for our technological civilization which is based on the use of raw materials. Allowing for a gradual, but by no means critical depletion of the mineral resources investigated, as well as for changes in demand pattern these largely naturally defined long term price ratios can be expected to change only very gradually under free market conditions.
However,the price ratios ofelements with higher specific mineralizabilities, where geo-political factors also play a more pronounced role, and those with lower values, will show a systematic tendency to increase with continuing demand.

   The energetical aspect (energy for mining, concentration, transportation, processing, refining and environmental protection and reclaining) has not been treated in this study. It could influence the price ratio and in general the economical and ecological consequences of ever increasing exploitation of these mineral resources.
A special investigation would be needed to assess the possible impact of this multiparametrical factor.

TABLE 1

Element	Clarke	Spec. Min.	Target Price (in US $ of 1970)	Long run price	Ratio
PGM	2.8 E-8	0.197995	1771	1700	1.04
Cr.	8. E-5	0.286692	0.006	0.07	0.89
Sn.	2.2 E-6	0.276770	3.01	2.75	1.09
F	6.	0.180313	0.13	0.12	1.08
P	1. E-3	0.197487	0.05	0.05	0.99
Pb.	1.6 E-5	0.286245	0.32	0.27	1.20

2. Departing from directly comparable estimates on the quality and size of the demonstrated reserves of these elements, the inferred marginal resources (prices up to 1.33 and 1.67 times the target prices), which are shown in figures 4-9, can be compared with recent estimates by the USGS (5) as shown in table 2.

TABLE 2

Element	Reserves (tonnes)		Resources	Ratio Mimic/USGS	
	Demonstrated	Inferred	Marginal	Reserves	Resourc
PGM	1.5 E+4	3.2 E+5	2. -10.E+6	8.5	50 - 250
Cr.	4.5 E+8	7.1 E+8	1.4 E+10	1.3	7
Sn.	2. E+7	8. E+7	3. - 8.E+8	3.	6 - 15
F	7.4 E+7	4.7 E+9	0.4-1.1E+11	-	250 - 650
P	6.2 E+9	1.2 E+10	1. -2.5E+11	2.	7 - 17
Pb.	2.7 E+8	5. E+8	2. -5. E+9	1.9	1 - 3

Estimates in each category of resources do include the foregoing categories and demonstrated reserves do include production to date.

3. The Mimic estimations are based on the assumption that element concentrations are log-binomially distributed in the natural environment. Normal concentration ranges (median ± 2 standard deviations) which for a log-normal distribution would have a fixed probability, with the log binomial distribution depend also on the size of the sampled area in respect to the total area. However, for subdivisions of the environment above the 10th order (sample area less than 1/1000 of the total environment) this probability would be in the order of 96 %. In tables 4-9 the concentrations ranges for different subdivisions of the environment are given. In table 3 these ranges are given for a 40th order subdivision of the earth's dry land surface to a depth of 2.5 km, which corresponds with partial environments of about 1 million tonnes of rock.

TABLE 3

Element	Median	log S.D.	Normal range Median ± 2 S.D.	upper limit in g/t
PGM	1.26 E-8	1.267	9.99E-10 - I.59E-7	0.16
Cr.	1.45 E-5	1.862	3.49E- 7-- 6.00E-4	600
Sn.	4.49 E-7	1.794	1.24E- 8 - 1.63E-5	16
F	3.11 E-4	1.151	3.11E- 5 - 3.10E-3	3100
P	4.53 E-4	1.263	3.62E- 5 - 5.66E-3	5700
Pb.	2.91 E-6	1.859	7.07E--8 - 1.20E-4	120

Under the assumption that the clarkes are correctly estimated, values exceeding the indicated upper limits would on the average in only about 2 % of the cases be caused by the natural distribution pattern.
Actually the clarkes are probably somewhat underestimated and therefore such higher values could have somewhat higher natural frequency of occurrence.

Explanation on figures 4-9

Figure 4 - Text of the caption

Inferred resources of platinum group metals (PGM). Plotted on the horizontal scale are the inferred resources of platinum group metals occurring in deposits containing between 0.1 and 2.8 E+10 (28 billion) tonnes PGM. (the earth's dry land surface to a depth of 2.5 km).

On the vertical scale PGM concentrations are given in g/tonne (parts per million).

The diagonal line indicates PGM deposits of the highest possible concentration for a given PGM content : from this diagonal, lines of equal metal content for lower grade deposits of possible economic interest are plotted and form the Iris or rainbow diagram.

Superimposed on this diagram are the average production costs expressed as a fraction of the theoretical target price as determined from the clarke and specific mineralizability. This target price is considered as the statistical price unit S.P.U. for the elements in question.

Figures 5 to 9

Explanation as for figure 4.

I. Classification and terminology of reserves and resources

A comparison was made between the classification used by the Commission (3,4) as illustrated in figure 1 and the classification along the lines proposed by McKelvey which recently has been agreed upon between the US Geological Survey and the US Bureau of Mines and which is illustrated in figure 2 (5).

Both classifications are modifications of the Blondel-Lasky classification of 1956 (9) and both go in the same direction of highlighting the importance of the degree of certainty (quantitative aspect) and feasibility of recovery variables (qualitative aspect). Both classifications have been developed with the aim that resources categories and definitions should be applicable to all naturally occuring concentrations of metals, non-metals and fossil fuels. They both limit the definition of "Reserves" to that portion of the estimated resources which, at the time of the estimation, can be economically extracted with current technology (the Survey's definition explicity adds the aspect of legality).

Some minor differences in definitions and terminology between the two classifications should be noted.

In the USGS definition "Resources" are limited to concentrations of naturally occurring solid, liquid, or gaseous materials in or on the earth's crust in such a form that economic extraction of a commodity is currently or potentially feasible. Such resources correspond with the "Reserves" + "Potential Reserves" of figure 1, whereas the Commission's "Resources" are defined to represent the total of all concentrations of mineral materials in the geological environment, irrespective of the feasibility of economic extraction at any time. Thus a class of "latent" resources is added in the Commission's definition of resources.

In the Survey's classification, the rather ambigeous terms of "probable" and "possible" and consequently of "proved" reserves are omitted. These terms which are used by the mining industry for economic evaluations in specific deposits and districts, commonly have been used loosely and interchangeably with the terms "measured", "indicated", or "inferred". The terms "proved" and "measured" are often essentially synonymous. The terms "probable" and "possible" however, are not necessarily synonymous with "indicated" and inferred and especially when these terms are used to describe partially sampled deposits they would be described in both classifications by the term "indicated".

On the other hand, the terms "demonstrated" and "reasonably assured" ; the latter being commonly used in uranium resources estimates (10), appear to be largely synonymous.

The subdivision of yet unmeasured mineral deposits in the USGS classification into "Identified inferred" and "Undiscovered hypothetical" and "Undiscovered speculative" resources refers to geographic distribution and type of such resources and does not correspond with the Commission's "inferred" and "not estimated" resources. Actually both the Identified "inferred" and the Undiscovered "hypothetical" and "speculative" resources fall under the Commission's definition of inferred resources.

The Commission's classification has been developed with the partial purpose to accomodate the MIMIC model. Therefore the class of "Not estimated" (because unknown) resources was explicitly introduced to reflect the fact that the MIMIC estimations, although covering the whole range of possible size-grade specifications for "inferred" resources in the geological environment, are necessarily minimum estimates.

For the analysis of the global resources of the forementioned elements the Commission's classification and terminology will be used in this study.

II. The distribution of elements in the geological environment

Different models have been proposed to describe the distribution of elements in the geological environment. Although the log-normal model of element distribution (11,12) is most frequently used in geochemical exploration and ore-reserve evaluation, our own observations made us choose the closely related log-binomial model as proposed by De Wijs (13,14) for the MIMIC model.
Apart from giving more consistent results when applied over very large probability ranges ($\gamma \pm > 3\sigma$) the model has the added advantage that observations on the distribution of mineral resources and ore deposits in metallogenetic provinces and districts can be easily understood.

In this model the geological environment (R) is considered as an inhomogenous mixture of a given element with a fixed average concentration (X) and a matrix material consisting of all other elements ($X \ll 1$).

Ore deposits represent, in this environment, extreme concentrations which are related with the average concentration through a series of intermediate concentrations. The weighted frequencies of the logarithms of these concentrations tend to fit a binomial probability curve which is determined by a median concentration (γ) and a standard deviation (σ).

Both the median concentration and the standard deviation are determined by the average size of the measured concentrations with respect to the size of the environment as a whole, and by a dispersion coefficient which is typical for each element in a given environment. This dispersion coefficient reflects the tendency of an element to occur in concentrated form and therefore has been named the "specific mineralizability" (Q) of the element (15).

Knowing the average concentration and the specific mineralizability of an element for a given geological environment, the probability of occurrence (expressed as tonnes of metal) for its "inferred resources" occurring in mineral deposits of all possible size-grade specifications can be calculated from (2) :

$$M = R.X. = \sum_{k=0}^{\alpha} \frac{\binom{\alpha}{k}}{2^{\alpha}} R.X. \; (1 + Q)^{\alpha-k} \quad (1 - Q)^{k} \qquad [1]$$

in which :

M = total resources of an element in R.
R = size of the environment in tonnes of rock.
X = The average concentration of the element in R.
Q = The specific mineralizability.
α = A rational number indicating the order of subdivision of the environment.
k = An integer 0,1,2,....,N $\left[\alpha - 1 < N \leq \alpha \right]$

Two boundaries limit the validity of the model :

- the minimum size of the concentrations for which the model is valid is determined by the size of the individual minerals making us the environment. Although this limitation does not pose a problem for resources evaluations, it always should be considered when planning a sampling program.
- no concentration greater than 1 could possibly occur and for all practical purposes the highest concentration which should be considered in resources estimates is the concentration of an element in its most common ore mineral.

Especially with elements of high natural abundance ($>$ 0.1%) or those with high specific mineralizabilities ($>$ 0.25) a large part or all of the reserves may occur in deposits for which the average grade approaches this theoretical maximum concentration. In such deposits the element concentrations are not log-binomially distributed any longer (overflow conditions).

Using the current average cost of finding, developing and mining mineral deposits of given size-grade specifications, the inferred resources can be subdivided into inferred "marginal", "submarginal", and "latent resources".

It has been observed by John R. Menke (quoted by Schurr and Marschak (16)) that " the cost of mining and refining per lb of ore of widely differing minerals is remarkably alike ... In other words, then, the cost of minerals won by <u>similar mining methods</u> is determined in good part by the number of lbs. of ore mined per lb of pure mineral."

A very similar observation was made with regard to exploration costs (17) which depend on the chance to discover deposits of given size-grade specifications in the geological environment. This chance can be determined with the log-binomial model and is proportional to the number and size of the available targets in the environment. Therefore, for about equally mature mining industries with similar cost structures the cost differences per unit weight between different minerals appear to be largely determined by the natural parameters "average abundance" and "specific mineralizability" of the elements in the accessible part of the earth's crust. Thanks to economies of scale, mineral deposits within a rather large range of size-grade specifications can be exploited at about equal costs and this explains why these average cost differences also are reflected in the average long-term price differences between elements with similar mining cost structures but rather different degrees of industrial maturity.

It was found that the historical average price differences between gold, copper, lead and zinc until 1971 correspond within 30 % with the target price differences as calculated by the MIMIC model from the estimated values for the average abundance and specific mineralizability of these elements :

$$T = \mathrm{Exp}\,(\,8.96637 - 25.5688 \times Q\,) \,/\, (\,X \times 10^{6}\,)$$

T = <u>Target price</u> in US $ (1971 value) per kg element contained.

The observations confirmed the calculated average long-term price as predicted by MIMIC for the much younger uranium industry and subsequent studies on the price, quality and size of the reserves of antinomy, manganese, mercury and molybdenum also confirm this relationship. The 30 % differences which are supposed to be largely due to small differences in mining, milling, refining and marketing costs between these different metals should be seen in the light of unit price differences covering 4.3 orders of magnitude (US $ 0.055 - 1126.000), 6 orders of magnitude for abundance values (10^{-9} - 10^{-3}) and Q values between 0.19 and .39.

The indicated relationship between these three parameters, apart from its very important economic and monetary implications, also can serve as a valuable tool to check on the estimation of the specific mineralizabilities of the different elements for which we want to make the resources estimates.

Very few published estimates for the specific mineralizability of different elements have been made yet (12,13,14,18). These estimates were all made from measurements in the ore deposits of the different metals. Representing relatively small parts of the total environment and obviously biased in favor of economic mineralisation, such estimates not necessarily are representative for the environment as a whole. Nevertheless, the values given by De Wijs for copper, zinc, lead and gold in hydrothermal fissure veins are comparable with values more recently found from geochemical surveys for these elements (2) and with the estimations made by the MIMIC model on the basis of the quality and size of the known ore reserves and former production of these metals.

Another important indication for the approximate value of the specific mineralizability can be found from the geographical distribution of the ore reserves of an element,
A relatively equal dispersion of reserves and deposits over the globe would be indicative for lower values.
If the majority of the reserves occurs in one, or possibly just a very few metallogenetic provinces, districts or deposits, higher values for the specific mineralizability are indicated (gold, mercury, antimony). Here, geopolitical factors are playing an increasingly important role on future price trends.

III. Determination of the parameters required by the MIMIC model

Resources and reserves of the different elements can be inferred by the MIMIC model from only 3 external parameters :

- the average concentration of an element in the geological environment,
- the specific mineralizability of the element for that environment,
- the size of the geological environment.

For the purpose of this study the reserves and resources we want to study are limited to the upper part of the dry land surface of the earth's crust and therefore we take this part as the geological environment for these resources.

1. The average concentration of the elements in the earth's upper crust.

Literature on the subject is rather extensive and for most elements reasonable agreement exists between different authors. A notable exception is tin, for which estimates by different contemporary authors vary within one order of magnitude.
Average concentrations normally are determined from average concentrations found in the major rock types of the earth's crust. Extreme concentrations normally are discarded and the arithmatic average concentration then is taken. This practice in many cases will result in a underestimation of the average concentration.
However, no better estimates are available.
For resources estimation the systematic underestimation of the average concentration is largely compensated by a consequent overestimation of the specific mineralizability.
However, for the determination of normal concentration ranges (median $\pm$ 2 st. deviations) of these elements in the natural environment the concentrations may have been underestimated.

2. The specific mineralizability of the elements for the upper part of the earth's crust.

As mentioned before, literature on the subject is still very limited and for most elements no published estimates exist at all.

The specific mineralizability can be estimated in several ways :

- from the dispersion of the element in its geological environment (7). Very few routine geochemical surveys for which data and results have been published were conducted with the purpose of determining the dispersion of an element in its environment as a whole. Most geochemical surveys are conducted in order to locate so-called anomalies (dispersion trains and primary or secondary halos) which supposedly are caused by hidden or exposed mineral deposits of possible economic size and grade. Although such surveys have proved very effective in locating many important mineral deposits, insufficient consideration normally can be given to the weight and mutual independency of the individual samples to result into dependable estimations of the specific mineralizability. Furthermore, such locally found specific mineralizabilities are not necessarily representative for the earth's upper crust as a whole and therefore a weighted average from many of such specially conducted surveys will be required for significant extrapolations.
- from the quality and size of individual ore deposits. Considering the enrichment in the highest grade ore deposit in a series of equally sized deposits as the maximum possible enrichment for that particular size, an apparent value for the specific mineralizability can be calculated. Repeating this procedure for different sizes, a number of values is found. As all deposits occur in the same environment, the highest of the thus found values should be the best approximation of the true specific mineralizability. This estimation on the basis of the probability of occurrence of the rarest type of ore deposit depends on only one observation which is a rather poor basis for statistical interpretations.
- from the quality and size of the ore reserves plus former production of a mineral commodity.✗ The demand for mineral commodities, at any given time in history is closely related to the state of technological progress at that time. Under free competition the price of a commodity is determined by the contribution of the marginal producer to the supply of a commodity.

✗ which is considered as the measured probability of occurrence of deposits of given average grade and size.

The grade and size of the ore deposits making up the reserves of a commodity therefore will show an at least semi-quantitative relation with the natural availability of mineral deposits of similar size-grade specifications. The quality of a mineral reserve is determined by the quality of its ore deposits, which in turn is largely determined as a function of grade, size and cost of extraction. Both grade and size of the ore deposits of a mineral commodity tend to be log-normally distributed (19). This can be explained from the fact that for a given average size ore deposit, deposits of higher than average grades become increasingly scarce, whereas deposits of lower average grade can be exploited only in special circumstances. A similar reasoning holds for the distribution of economic sizes for a given average grade. For the determination of the average quality of the ore deposits making up the reserves and determining the economics of a mineral industry it was found that from the relatively extended field of possible economic size-grade specifications both the average grade and the average size of the ore deposits making up the reserve can best be defined independently from each other as the grade and size for which 50 % of the reserves occur in higher grade, respectively greater size deposits ; the other 50 % in resp. lower grade and smaller size deposits.
For the estimation of the specific mineralizability with the MIMIC model the quality and size of the "demonstrated" reserves of a commodity normally is used.

- from the established relationship between the average concentration in the crust, the long term average price and the specific mineralizability, the latter can be estimated as a function of the first two parameters. This, obviously, could be a rather dangerous procedure as it could lead to a viceous circle in which the parameter values that are required for the estimation of the specific mineralizability are determined from the value of the specific mineralizability as found from this abundance-price-specific mineralizability relation. However, this procedure can be used and justified for cases where published data on these parameters indicate a range of possible values.
This has been the case, for instance, for the estimation of the average grade of mercury in its ore deposits (8).

In general it will be necessary for the estimation of the resources of elements for which no estimates have been published yet to use as much of the available methods and data as possible.
Only a combination of the different methods will show possible weaknesses in the estimations and indicate the direction for further research.
For the resources estimate, however, the most conservative results normally should be accepted as the estimates are minimum estimates in any event.

3. <u>The size of the geological environment of ore deposits</u>

For the purpose of this study only the upper part of the earth's continental crust will be considered as the environment for ore deposits.

The weight of this environment is taken as 10^{18} tonnes of rock, roughly corresponding with the dry land surface of the crust to a depth of 2.5 km. The currently accessible depth for mining is somewhat over 3 km but the great majority of the reserves occur between 0 - 1000 m.

4. <u>Cost calculations</u>

Average unit production costs are determined from the size/grade specifications of the mineral deposits making up the demonstrated and inferred resources.

Unit production costs in constant 1971 US $ are calculated by discounting cash flow for the moment when a fully equipped mineral deposit enters into production. Costs are divided into exploration, capital investment and exploitation costs.

In all our calculations a discount rate of 10.4 % is used which corresponds with an 8 % after tax return on equity for a risk capital ratio of 0.3 and 50 % marginal tax on profits.

Both the depletion and amortization time of investments in a mineral deposit are set at 16 years.

For technical and/or economic reasons the following corrections and restrictions are made for the resources and cost calculations.

i. highest element concentrations are limited by the element concentration in the most common ore mineral.

ii. costs are corrected for specific gravity of the ore minerals.

iii. for deposits with annual production capability exceeding 10 % of world requirements, costs are calculated as for deposits producing 10 % of these requirements. Excess ore after amortization of investments, is valued at discounted unit exploration cost.

iv. daily ore handling capacity is limited to 250.000 short tonnes. Excess ore is valued as under iii.

v. if daily ore handling capacity exceeds 30.000 short tons, unit capital investment costs are calculated as for a ore handling capacity of 30.000 daily short tons.

In order to compensate as much as possible for the small cost differences between different elements due to differences in mining, milling, refining and marketing costs, all costs are expressed in respect to the target price of each element, which is considered as the "statistical price unit" (S.P.U.) of that element.

Thus all deposits with costs lower than 1.00 S.P.U., per definition can be considered as reserves, whereas deposits with higher costs are to be considered as potential reserves. In our calculations deposits with indicated average costs up to 1.67 S.P.U. will be considered as marginal resources.

IV. The development of reserves from resources

Production of a commodity depletes its demonstrated reserves. This is counteracted by mineral exploration, which :

a) develops inferred reserves into demonstrated reserves, and

b) attempts to discover new reserves from the "not estimated, exploitable resources" (Only deposits with equal or better size-grade specifications than the current average ore deposit can occur in this class.

Their discovery will result in a reevaluation of the specific mineralizability), and

c) technical improvements and economies of scale on the extraction, transport and marketing of the product, which brings potential reserves into the reserves, and

d) rise of price of the product, which also brings potential reserves into the reserves.

During the relatively long mining history of many mineral commodities it has been observed that the first three factors normally can guarantee the development of adequate reserves and production capacities and that the price has a long-term tendency to stabilise or even decrease with respect to a constant currency value. The unbalanced price developments since the monetary crisis of August 1971 and the subsequent de facto elimination of gold as a monetary standard although largely predictable in its trend, have not been taken into account and all historical price developments refer to the 1971 US $ value with the gold price at US $ 35.00 oz Au (US $ 1125.28 kg Au).

V. Estimation of the MIMIC model parameters and "inferred" resources for the metals of the platinum group, chronium, tin, fluorine, phosphorus and lead

1. The metals of the platinum group.

Average concentration of the Pt-group metals in the crust

element	(1)	(2)	(3)	(4)	(5)	
Ru (44)	-	p		0.0001	-	
Rh (45)	0.00x	0.001		0.001	-	
Pd (46)	0.0x	0.01	0.013	0.02	-	
Os (76)	-	p		0.0001	-	
Ir (77)	0.00x	0.001		0.001	-	
Pt (78)	0.0x	0.005		0.005	0.028	
Pt-group	0.0x	0.017	0.013	0.0272	0.028	g/tonne

(1) Niggli (after Goldschmidt), 1948 (20)
(2) Rankama & Sahama (after Goldschmidt), 1950 (21)
(3) Vinogradov, 1962 (22)
(4) Green, 1959 (23)
(5) PP 820 (after Lee and Yao), 1973 (5)

The crustal abundance of the platinum group metals is not very well known yet but the order of magnitude (0.0x g/tonne, in which x = 1 - 9) seems to be rather well established. From the quoted estimates which are not always independent of each other, an average crustal abundance of 0.021 ± 0.005 g/tonne seems to be indicated. The fact that some of the estimates do not include all Pt-group metals and that the more recent estimates are markedly higher than this calculated average make it likely that the true average will be higher. For this reason the value of 0.028 g/tonne, which is given in PP 820 (1973) for the abundance of platinum alone is suggested for this study as the ave. abundance of the platinum group metals.

The apparent ratio Pd : Pt : Ir : Rh : Os : Ru = 13 : 5 : 1 : 1 : 0.5 : 0.5 probably is only a very rough approximation of the true ratio.

Production and reserves

Production in 1971 amounted to 126 tonnes of platinum group metals (1970 = 132 tonnes) as compared to an average production of 33 tonnes per year for the period 1954 - 1958, and indicating an annual growth rate of approximately 9.3 %(24,25). Production came from the following countries :

Canada	(platinum and other Pt group metals)	11.4 %
S.Africa	(platinum group metals + Os from gold)	30.7 %
U.S.S.R.	(platinum group metals)	56.4 %
others	(Colombia,U.S.A.,Japan,Philipines,..)	1.5 %

In Canada most of the platinum is recovered as a by-product of copper and nickel production from the Sudbury district, Ontario. PGM content of the ores is less than 1 g/tonne and platinum and palladium occur in about equal amounts.

In S.Africa platinum is produced from the Merensky reef in the stratiform Bushveld Complex and osmiridium is recovered as a by-product from gold production in the Witwatersrand.
The average grade in the Merensky Reef has been estimated by Newman at 6.2.g/tonne of PGM and is confirmed by others (26).

Platinum constitutes approximately 63 %, palladium 24 % and the other PGM 13 % of the mineral content of the ores. For the Bushveld Complex as a whole the platinoid + gold metal content of the ores has been estimated as :

Pt	-	60 %
Pd	-	27 %
Ru	-	5 %
Rh	-	2.7 %
Ir	-	0.7 %
Os	-	0.6 %
Au	-	4.0 %

In Russia (U.S.S.R.) platinum is produced from placer deposits in the Ural Mountains in which platinum exceeds all other PGM, and from platinum deposits in the Noril'sk District (northern Siberia) where the deposits occur in a concentric mafic complex (5,26). In the latter deposits palladium is from 2 - 10 times as abundant as platinum. PGM grades in the ores vary from 0.06 - 10.6 g/tonne in the Rudnaya Gora mine from which about 60 % of the production is obtained and average grades have been estimated in the order of 4 - 6 g/tonne.

Demonstrated reserves per Januari 1971 were estimated at 13,187 tonnes of platinum group metals (25) which figure also has been used in PP 820 for the identified resources of the platinum group metals. Of the demonstrated reserves 47 % or 6,220 tonnes occur in the Merensky Reef in South Africa which therefore is by far the most important known future source of the platinum group metals. Its inferred reserves and resources exceed the demonstrated reserves.

Another 47 % of the demonstrated reserves occur in the U.S.S.R. but little details as to the distribution over ore deposits is available. Large amounts are supposed to be found in the Noril'sk District with comparable, but probably slightly lower grades than the Merensky Reef. Other important reserves are supposed to occur on the Kola Peninsula.
4 % of the demonstrated reserves occur in the Sudbury District, Ontario in copper, nickel ores containing less than 1 ppm PGM. The other 2 % of the reserves are found in the United States and in Colombia.

Total production to date has been estimated at 1980 tonnes PGM and forms only a minor fraction of the demonstrated reserves.

From the foregoing reserve statistics it can be seen that the quality of the reserves is largely determined from the grade-size specifications for the Merensky Reef deposit, making up approximately 47 % of the known reserves. The true average grade of the reserves will be slightly lower than 6.2 ppm PGM, whereas the average size will be slightly below 6.220 tonnes PGM. Instead of entering into uncertain interpolations it is proposed to use the reserve estimate for the Merensky Reef to determine the specific mineralizability for the platinum group metals.

Long term average price

From the reported value of imported, unrefined platinum group metals in the U.S.A. during the period 1954 - 1971, an average long term price of US $ 1696 ± 16.6 % per kg PGM was found (24,25).

Input data for the resources estimation by "MIMIC"

From the foregoing description the following parameters for the estimation of platinum group metals resources have been used :

1. Average concentration in the crust = 0.028 E-6 = $0.028.10^{-6}$ or 28 parts per billion.
2. Concentration in largest ore deposit = 6.2 E-6
3. Size of the environment (tonnes rock) = 1. E+18
4. Size of largest ore deposit (tonnesPGM) = 6.22 E+3
5. Size of reserves largest ore dept (") = 6.22 E+3
6. Average long term price (US $ per kg) = 1.70 E+3
7. Name of estimated commodity = PGM

From these data, a composite specific mineralizability of 0.197995 could be determined for the platinum group metals. The corresponding target price of US $ 1771.22 per kg PGM contained is 1.04 times the average long-term price for these metals (24,25).

From this specific mineralizability the reserves and resources estimate of figure 4 was made.

Inferred reserves are calculated as 320,000 tonnes PGM and include approximately 15,000 tonnes PGM in demonstrated reserves and former production. This estimate is about 7.5 times the recent USGS estimate of 43,200 tonnes PGM in all idetified, hypothetical and speculative resources.

A profit optimum, which is defined as the maximum value for the product of inferred resources times indicated profit per unit, is found for inferred reserves of 149,200 tonnes PGM occurring in deposits of average 240 tonnes PGM and with an average grade of 6.86 ppm PGM. This value was found for annual requirements of 150 tonnes PGM and the optimum therefore is determined by the restriction that no single deposit should produce more than 10 % of the annual requirements. Without this restriction inferred reserves would be substantially larger and a profit optimum would be found for inferred reserves of 470,000 tonnes PGM in average deposits of 1190 tonnes PGM and an average grade of 4.12 ppm PGM.

Marginal resources at costs up to resp. 1.33 and 1.67 SPU are estimated between 2. - 10. million tonnes PGM.

Some reservations as to the validity of these estimates should be made. The PGM distribution in the ultrabasic rocks in which most PGM ore deposits are found may not be typical for the essentially granitic upper layer of the earth's crust and discontinuities, invalidating the interpolated resources estimates therefore could occur (15). The only way to validate such assumptions would be by direct measurement of PGM distribution in the earth's crust.

Assuming the validity of the foregoing assumptions, table 3 illustrates the normal expectable concentration ranges of the platinum group metals in the natural environment for different sample sizes. From the table it would be indicated that average PGM concentrations greater than 0.2 g/tonne have only a small chance ($\pm$ 2 %) to be the result of a natural distribution, whereas concentrations over 1 g/tonne PGM would be definitely suspect. In view of the fact that concentrations in the range 0.2 - 1.0 g/tonne in the Stillwater Complex, Montana (0.5 g/tonne) are classed under identified potential reserves (5) seems to support that the indicated range may be to high for the earth's crust as a whole.

Table 3

The total environment corresponds to the earth's dry land surface of 1.5 E+8 square km to a depth of 2.5 km and an average specific gravity of 2.7.

Clarke PGM = 2.8 E-8   Specific mineralizability = 0.197995

Environment tonnes	km^2	median	log SD	Normal range Median + 2SD	remarks
1.03+18	1.5 E+8	2. 8E-8			
1,0E+17	1.5 E+7	2.62E-8	0.3657	1.345E-8-5.102E-8	min.max
1.0E+16	1.5 E+6	2.45E-8	0.5172	8.709E-9-6.90E-8	
1.0E+15	1.5 E+5	2.29E-8	0.6334	6.45E-9 -8.14E-8	
1.04+14	1.5 E+4	2.15E-8	0.7314	4.98E-9 -9.27E-8	
1.0E+13	1.5 E+3	2.01E-8	0.8177	3.92E-9 -1.03E-7	
1.0E+12	1.5 E+2	1.88E-8	0.8957	3.12E-9 -1.13E-7	
1.0E+11	1.5 E+1	1.76E-8	0.9675	2.54E-9 -1.22E-7	
1.0E+10	-	1.65E-8	1.034	2.09E-9 -1.30E-7	
1.0E+ 9	-	1.54E-8	1.097	1.72E-9 -1.38E-7	
1.0E+ 8	-	1.44E-8	1.156	1.42E-9 -1.46E-7	
1.0E+ 7	-	1.35E-8	1.213	1.19E-9 -1.53E-7	
1.0E+ 6	-	1.26E-8	1.267	9.99E-10-1.59E-7	

2. Chromium

Average concentration of chromium in the earth's crust

Estimates on the average abundance of chromium in the earth's crust show rather large variations between different authors and average around 130 g/tonne ± 70 %. The most recent estimates by Vinogradov (22) and Lee and Yao (5) average at 80 ppm ± 5 % and are probably more representative for the true average than the higher estimates. Chromium is closely associated with magnesium and nickel in ultramafic rocks in which its concentration ranges from 1100 - 3400 ppm and drops to some 200 ppm in gabbro and basalt. In granites the chromium content is only about 5 ppm which explains the widely different estimations of its average crustal abundance.

Production and reserves

Production of chromite in 1971 amounted to approximately 6.3 million tonnes (6.05 million tonnes in 1970) against an average of 3.9 million tonnes during the period 1954 - 1958, indicating an average growth rate of 3,3 %. The average chromium content of the concentrates is about 30 %. Production came from 18 countries, which in order of importance were :

U.S.S.R.	28.55 %
South Africa, Republic of	26.12 %
Turkey	9.59 %
Albania	8.51 %
Philipines	6.86 %
Rhodesia, Southern	5.77 %
India	4.15 %
Iran	3.17 %
Malagasy Republic	2.23 %
Finland	1.77 %
Others	3.27 %

The tremendous resources of chromite of South Africa (Bushveld Complex) and the Great Dyke in Rhodesia make up 96.21 % of the demonstrated reserves of 1.69 billion (10^9) metric tonnes of chromite containing approximately 30 % Cr.

Two thirds of this amount are found in S.Africa in high-iron ores (Cr_2O_3 = 40 - 46 % ; Cr/Fe = 1.5 - 2.0) the other third in Rhodesia in high Chromium ore (Cr_2O_3 more than 46 % ; Cr/Fe = more than 2). The U.S.S.R. has approximately 1.26 % of the estimated reserves and the remainder is devided over more than 12 other countries of which Finland with 0.60 % apparently has the largest demonstrated reserves.

The Bushveld Complex in South Africa is a rudely basinshaped layered mass extending over some 70.000 km^2. The chromite occurs in stratiform bands of remarkably uniform thickness, parallel to the pseudo-stratification of the enclosing basic igneous rocks and can be traced for many kilometers. Over 29 chromite layers or groups of layers occur in various segments of the chromite bearing zone, which is exposed over a distance of over 100 km along the eastern margin alone (5,27). The individual layers vary in thickness from less than 1 inch to over 30 inches (1 - 75 cm) and the Steelport Main Seam, averaging about 1 m in true thickness has been followed over a distance of some 65 km.
It reserves have been estimated at 0.5 billion tonnes of chromite with an estimated average grade of $\pm$ 43 - 45 % Cr_2O_3 (30 % Cr). However, a unknown dilution factor should be taken into account for the determination of the true ore grade.

The Great Dyke in Rhodesia is a layered structure more than 500 km long and between 5 - 10 km wide. In cross section the Dyke is a synclinal with dips of 20 - 40° near the margins. Chromite layers, varying in thickness from 2 - 18 " (5 - 46 cm) occur over the entire length of the Dyke, and individual layers have been traced contineously the entire width though they vary in number from 6 - 12 in different segments. Nearly all the chromite is of the high-chromium variety (5). Only layers 6 " (15 cm) or more thick are mined below the outcrop and some of the thicker layers are mined more than 1000 feet (300 m) down the slope. Assuming a minimum mining width of 60 cm dilution up to 25 % of the chromium content in the chromite could occur.

From the foregoing description it can be seen that the Steelport Main Seam certainly is the largest individual ore deposit, containing an estimated 1.5×10^8 tonnes of Chromium in ore (allowing for 50 % dilution) of an average of 15 % Cr.

Production to date is estimated as less than some 150 million tonnes of chromite or 45 million tonnes of chromium which is only a relatively small fraction of the demonstrated reserves.

In view of the fact that the estimated reserves of the Steelport Main Seam, which contain about 1/3 of the total known reserves are certainly more representative than any extrapolated average grade and size for the ore deposits making up the reserves, it is proposed to use this reserve estimate of the largest deposit for the estimation of the specific mineralizability.

Long-term average price

Assuming a rough proportionality between the Chromium content of chromite concentrates and the price of the kg of chromium contained a long term average price of 0.0725 \$ per kg of chromium was found for the period 1956 - 1971 (24,25).

Input data for the resources estimates by "MIMIC"

From the foregoing description the following parameters for the estimation of the chromium resources have been used :

1. average concentration in the crust = 8.00E-5
2. concentration in largest ore deposit = 1.5E-1
3. size of the environment = 1.E18
4. size of largest deposit (tonnes Cr) = 1.5E8
5. size of reserves largest deposit (tonnesCr) =1.5E8
6. name of estimated commodity = Chromium
7. average long term price (US \$ per kg cont. Cr) = 7.25E-2

Resources estimate

From the input data a specific mineralizability of 0.286692 is determined for chromium. The corresponding target price of \$ 0.064 per kg contained chromium is 0.89 times the average long-term price (24,25).
With this specific mineralizability the reserves and resources estimate of figure 5 was made.

Inferred reserves are calculated at 710 million tonnes of chromium and include 450 million tonnes of demonstrated reserves plus former production. This estimate is only 1.3 times the USGS estimate for inferred reserves. Marginal reserves at 1.5 S.P.U. are estimated at 14 billion tonnes of chromium which is about 7 times the USGS estimate on all identified, hypothetical and speculative resources (5).

A profit optimum is found for 318 million tonnes chromium reserves in average deposits containing 4.3 million tonnes chromium with an average grade of 38 % chromium. This optimum falls well within already demonstrated reserves of 450 million tonnes chromium, indicating that with our current knowledge of chromium ore deposits few additional reserves can be inferred. Gradual depletion of the still very large demonstrated reserves (225 years at the current rate of production of some 2 million annual tonnes) therefore will tend to increase exploitation costs with decreasing ore grades.

Just as for platinum, the chromium distribution in the ultrabasic rocks in which most ore deposits are found may not be typical for the granitic upper layer of the earth's crust. However, thanks to the much higher specific mineralisability of chromium, the reserves would be concentrated mostly in these ultrabasic rocks anyhow and due to this fact eventual distribution discontinuities would hardly affect the resources estimates.

Assuming the validity of the calculated specific mineralizability for the whole concentration range, table 4 illustrates the range of normal expectable concentrations (median + 2 standard deviations) for different sample sizes. From the table it would appear that concentrations greater than 600 g/tonne would have approximately 2 % chance to be the result of natural distribution, whereas for concentrations greater than 4000 g/tonne this chance would be about 0.001. However, 1100 - 3400 g/tonne is the normal range for ultramafic rocks.

Table 4

The total environment corresponds to the earth's dry land surface of 1.5 E+8 square km to a depth of 2.5 km and an average specific gravity of 2.7.

Clarke CR = 8.0 E-5 Specific mineralizability = 0.286692

Environment tonnes	km^2	median	log SD	Normal range Median ± 2 SD	remarks
1.0E+18	1.5 E+8	8.0 E-5			
1.0E+17	1.5 E+7	6.9 E-5	0.5376	2.604E-5-1.848E-4	min.max.
1.0E+16	1.5 E+6	6.0 E-5	0.7603	1.315E-5-2.752E-4	
1.0E+15	1.5 E+5	5.2 E-5	0.9311	8.104E-6-3.359E-4	
1.0E+14	1.5 E+4	4.5 E-5	1.0752	5.269E-6-3.886E-4	
1.0E+13	1.5 E+3	3.9 E-5	1.2021	3.545E-6-4.344E-4	
1.0E+12	1.5 E+2	3.4 E-5	1.3168	2.444E-6-4.739E-4	
1.0E+11	1.5 E+1	2.9 E-5	1.4223	1.716E-6-5.075E-4	
1.0E+10	1.5 E+	2.5 E-5	1.5205	1.223E-6-5.356E-4	
1.0E+ 9	-	2.2 E-5	1.6128	8.819E-7-5.586E-4	
1.0E+ 8	-	1.9 E-5	1.7000	6.424E-7-5.768E-4	
1.0E+ 7	-	1.6 E-5	1.7830	4.719E-7-5.905E-4	
1.0E+ 6	-	1.4 E-5	1.8623	3.492E-7-6.001E-4	

3. Tin

Average concentration of tin in the earth's crust

Estimates of the average concentration of tin in the earth's crust show differences of one order of magnitude between different authors. Rankama and Sahama, 1949 estimate the average abundance (after Goldschmidt, 1933, 1937) at 40 g/tonne, which estimate is more or less confirmed by Green, 1959 as 32 g/tonne. Niggli, 1948, with x.0 g/tonne ; Vinogradov, 1962, with 2.5 ; Lee and Yao, 1970 with 1.7 and Sainsbury and Reed, 1973 with 2 - 3 g/tonne average around 2.2 g/tonne which here is accepted as the most likely value.

Production and reserves

Production of tin amounted to 233,000 tonnes in 1971 (233,000 in 1970) against 191,000 tonnes per year during the period 1954 - 1958. The average annual production during this period was 202,000 tonnes ± 6.6 %. Average annual growth during this period has been about 1.3 %. Production came from 36 countries of which the most important are :

Malaysia	32.35 %
U.S.S.R. (estimated smelter production)	12.20 %
Bolivia	11.96 %
Thailand	9.30 %
China (Peoples Republic of) (estimate)	8.71 %
Indonesia	8.46 %
Australia	4.08 %
Nigeria	3.05 %
Zaire	2.79 %
Brazil	1.12 %
South Africa	0.87 %

From this production, historically about 75 % has been produced from alluvial and eluvial placer deposits and the majority of the demonstrated reserves are in deposits of this type.
Total production plus demonstrated reserves in 1960 were estimated by Moussu, 1962 (28), at 17.000.000 tonnes tin.
Presently, demonstrated reserves are estimated at 4.200.000 tonnes Sn (25) and 3.700.000 tonnes (5).
Assuming that reserves during the period 1960 - 1971 were discovered at the rate of production, the total production to date can be estimated around 16 million tonnes and the demonstrated reserves at 4 million tonnes Sn, in order of importance divided as follows :(5)

Malaysia	16.44 %
Indonesia	13.70 %
China (guestimate)	13.70 %
Bolivia	13.29 %
Brazil	8.22 %
Burma	6.85 %
Thailand	5.95 %
U.S.S.R.	5.48 %
Nigeria	3.78 %
Other countries (less than 100.000 tonnes)	12.59 %

As the majority of reserves occur as placer deposits, reported ore grades are not directly comparable with ore grades in other mining industries. The mineralization includes a large amount of sterile overburden, which in normal open-pit mining would be recognized as such. Thus the fact that for placer operations in Malaysia, Indonesia and Nigeria ore grades of respectively 216, 270 and 300 g/m^3 or resp. 137, 171 and 190 g/tonne were reported is not indicative for comparable ore grades in hardrock mining, where much higher grades are reported 0.3 - 12 % Sn. Actually, even at sea the tin is found in a relative thin layer between 1 - 100 cm in which the metal concentrations may vary between 10 - 30 %. In hard-rock mining, allowing for dilution size of the deposits and mineable width this would probably correspond to average grades of 0.3 - 1 % or more, in line with the mineable grades found for pegmatite deposits, pipes and veins, of which some of the most famous may be mentioned here :

<u>pegmatite deposits</u>

The heavily altered Manono pegmatite in Zaire has been estimated to contain 1 million tonnes of tin over its total length of 14 km, width of 200 m and depth of 50 m, which would indicate an average grade of some 0.3 % Sn.

pipes

The vulcan pipe of North Queensland in Australia, which has been exploited between 1891 - 1930 (Moussu, 1962) has yielded 8200 Tonnes Sn in ore averaging 4.5 % Sn.

vein deposits

Several large deposits of this type have been found. One mine in the Llallagua District of Bolivia has produced 500.000 tonnes of Sn from ores averaging between 2 - 3 % Sn (Bateman, 1949). In Cornwall, England the Dalcoath lode has produced some 80.000 tonnes Sn from ores averaging between 1 - 2 % Sn (Moussu, 1962).

The average size of the placer deposits is somewhat difficult to estimate. The majority of deposits contain between 1000 - 10.000 tonnes Sn but a number of very important deposits contain far in excess of this amount. Thus, the average size of the deposits as defined for the MIMIC estimates will be somewhere between 10.000 - 20.000 tonnes Sn.

Long term average price

Prices of tin are controlled within certain limits by a International Tin Council, which determines floor and ceiling prices below which the manager of the tin buffer stock should buy from the open market and above which he has to sell from this buffer stock. The average price over the period 1954 - 1971 has been US $ 2.75 ± 25 % per kg Sn.

Input data for the resources estimation by MIMIC

From the foregoing description the following parameters for the estimation of the tin resources are proposed :

1. average concentration in the crust	(X1)	= 2.2E-6
2. concentration in ave. ore deposit	(X3)	= 1.0E-2
3. size of the environment	(R1)	= 1. E18
4. size of ave. ore deposit (tonnes Sn)	(M3)	= 1.4E4
5. reserves + former production (ton.Sn)	(M2)	= 1.8E7
6. average long term price (US $ per kg)	(A$)	= 2.75
7. name of estimated commodity	(c5)	= Tin

Resources estimate

From the foregoing data a specific mineralizability of 0.276770 is determined for tin. The corresponding target price of $ 3.00 per kg tin is 1.09 times the average long term price (24,25).

With the indicated specific mineralizability the reserves and resources estimate of figure 6 could be made.

Inferred reserves are calculated at 80 million tonnes of tin and include 20 million tonnes of demonstrated reserves plus former production. This estimate is about 3 times as high as the comparable USGS estimate of 1973 (5). Marginal reserves at prices up to resp. 1.33 and 1.67 S.P.U. are estimated at 300 and 800 million tonnes and are 6 - 15 times the USGS estimates for all identified, hypothetical and speculative resources (5).

A profit optimum was found for 30 million tonnes of tin in average ore deposits of 120.000 tonnes tin and average grade of 0.48 % tin.

Table 5 illustrates the normal concentration ranges for tin in the geological environment as a function of sample size (median ± 2 standard deviations). It can be seen that average concentrations due to natural distribution patterns and higher than some 16 g/tonne would occur in approximately 2 % of the investigated cases, whereas concentrations over 100 g/tonne have a chance of about 0.001. Many granites do have average tin concentrations within this range 16 - 100 g/tonne.

Table 5

The total environment corresponds to the earth's **dry** land surface of 1.5 E+8 square km to a depth of 2.5 km and an average specific gravity of 2.7.

Clarke SN = 2.2 E-6   Specific Mineralizability = 0.276770

Environment tonnes	km^2	median	log SD	Normal range Median $\pm$ 2 SD	remarks
1.0E+18	1.5 E+8	2.2 E-6			
1.0E+17	1.5 E+7	1.9 E-6	0.5180	7.498E-7-4.954E-6	**min.max.**
1.0E+16	1.5 E+6	1.6 E-6	0.7325	3.901E-7-7.306E-6	
1.0E+15	1.5 E+5	1.4 E-6	0.8971	2.459E-7-8.896E-6	
1.0E+14	1.5 E+4	1.2 E-6	1.0359	1.632E-7-1.029E-5	
1.0E+13	1.5 E+3	1.1 E-6	1.1582	1.119E-7-1.151E-5	
1.0E+12	1.5 E+2	9.9 E-7	1.2687	7.861E-8-1.257E-5	
1.0E+11	1.5 E+1	8.7 E-7	1.3704	5.620E-8-1.350E-5	
1.0E+10		7.6 E-7	1.4650	4.074E-8-1.429E-5	
1.0E+ 9	-	6.6 E-7	1.5539	2.988E-8-1.495E-5	
1.0E+ 8	-	5.8 E-7	1.6379	2.212E-8-1.550E-5	
1.0E+ 7	-	5.1 E-7	1.7179	1.652E-8-1.593E-5	
1.0E+ 6	-	4.4 E-7	1.7942	1.242E-8-1.626E-5	

4. Fluorine

Average concentration of fluorine in the earth's crust

The natural abundance of fluorine in the earth's crust is not very well established yet. Estimations vary from 310 ppm (Niggli, 1948), 600 ppm (Rankama and Sahama, 1950), 470 ppm (Lee and Yao), 660 ppm (Green, 1959, Vinogradov, 1962), 650 ppm (Worl et al., 1973, lit 29) to 800 ppm (Goldschmidt, 1937). In this study an average of 600 ppm F will be accepted as the weighted average of these estimations.

Production and reserves

Production of fluorspar in 1971 amounted to 4.64 million tonnes (1970 = 4.17 million tonnes) as compared to a negligeable production in 1900. The indicated average growth rate over this period is in the order of 12 %. Production during the period 1954 - 1958 averaged 1.62 million tonnes indicating an average growth of 7.3 % over the last 15 years.

Production in 1971 came from 27 countries, in order of importance as follows ;

Mexico	25.45 %
Thailand	9.21 %
U.S.S.R.	9.00 %
Spain	8.61 %
France	6.46 %
Italy	6.26 %
China, People's Republic of	5.48 %
U.S.A.	5.32 %
United Kingdom	5.28 %
South Africa, Republic of	5.14 %
Others (less than 100.000 tonnes or 2 % each)	13.79 %

Production until present can be estimated at approximately 74 million tonnes fluorspar containing an estimated 32.4 million tonnes of fluorine.

Demonstrated reserves have been estimated at 172 million tonnes of crude fluorspar ore containing some 24.33 % fluorine or 42 million tonnes (5).

Fluorspar reserves are found a chiefly in shallow, low temperature hydrothermal vein, pipe, and manto stratiform deposits ; however, an increasingly important source of fluorspar will be as a co-product of mining and processing certain ores of iron, lead, zinc, bismuth, tungsten, molybdenum, tin, beryllium, rare earth's and in particular from marine phosphates.

Environmental considerations in the future, in fact, may demand the recovery of fluorine during ore processing so that it is not released into surfacial material, the atmosphere, or surface waters.

Some large deposits are the El Refugio ore body in the Rio Verde District, Mexico, containing 3.5 million tonnes ore averaging 85 % fluorspar ;

La Consentida - San Louis Potosi in Mexico contained 3.25 million tonnes grading 70 % fluorspar ;

Muscadroxio-Genna, Sardinia, Italy is probably the largest single ore body found to date with 6 million tonnes ore containing 65 % fluorspar.

In general, the ore deposits are rather small. The estimated average size deposit contains some 3.5×10^5 tonnes of fluorine in ore with an average grade of 24.45 % fluorine.

<u>Long term average price</u>

The average price for domestic fluorspar concentrates in the U.S.A. during the period 1954 - 1970 has been US \$ 50.86 $\pm$ 6.5 % and has been remarkably stable for a product with such a high growth rate. As import duties in the U.S.A. for fluorspar are rather high (\$ 2.10/longton for acid grade (+ 97 % CaF_2) and \$ 8.40/long ton for metallurgical grade (- 97 % CaF_2)) and the U.S.A. imports between 40 - 30 % of the world fluorspar production, world prices during this period may have been up to 20 % lower. Assuming rough proportionality between price and fluorine content of the concentrate, the price of fluorine would be in the order of \$ 0.116 per kg F contained (US price) to some \$ 0.093 per kg F (world price).

Input data for the resources estimation by MIMIC

1. average concentration in the crust	6. E-4
2. concentration in the ave.ore deposit	2.45E-1
3. size of the environment	1. E18
4. size of ave.ore deposit (tonnes F)	3.5E5
5. former production + 0.5 x reserves (t.F)	5.34E7
6. average long term price	0.12
7. name of estimated commodity	Fluorine

Resources estimate

From the input data a specific mineralizability of 0.180313 is determined for fluorine. The corresponding target price of $ 0.13 per kg contained fluorine is 1.08 times the already artificially high US long term average price (24,25).

With this specific mineralizability the reserves and resources estimate of figure 7 was made for annual requirements of 2.5 million tonnes fluorine contained.

Inferred reserves are calculated at 4.7 billion tonnes of fluorine and include 74 million tonnes in demonstrated reserves plus former production. The latter amount roughly corresponds with the total of all identified resources plus former production as estimated by the USGS (5). A profit optimum is indicated for reserves containing 2.6 billion tonnes fluorine in deposits of average size of 4 million tonnes fluorine and with average concentration of 8.73 % F. These figures indicate that the optimum is determined by the restriction that no single deposit should produce more than 10 % of the annual requirements. With increasing requirements and larger deposits this would indicate that reserves could be increased just by economies of scale.

Marginal resources at costs up to 1.33 to 1.67 S.P.U. are estimated at respectively 40 and 110 billion tonnes of F. This is between 250 and 650 times the USGS estimate on all identified, hypothetical and speculative resources (5).

Table 6 illustrates the normal expectable concentration ranges for fluorine in the natural environment as a function of sample size (median $\pm$ 2 standard deviations). It can be seen that average concentrations greater than 0.3 % due to natural distribution patterns could be expected in about 2 % of the studied environments, whereas concentrations of 1 % or more would have a chance of about 0.001. Except in some alkalic rocks and carbonatites fluorine concentrations in the range of 0.3 - 1.00 % do not occur in normal rock types. An exception is marine phosphate rock with some 3 % fluorine, but this already can be considered as a potential ore for fluorine as a by-product from the manufacture of superphosphates.

Table 6

The total environment corresponds to the earth's dry land surface of 1.5 E+8 square km to a depth of 2.5 km and an average specific gravity of 2.7.

Clarke F = 6.0 E-4   Specific Mineralizability = 0.180313

Environment tonnes	km^2	median	log SD	Normal range Median ± 2 SD	remarks
1.0E+18	1.5 E+8	6.0 E-4			
1.0E+17	1.5 E+7	5.6 E-4	0.3323	3.100E-4-1.041E-3	min.max.
1.0E+16	1.5 E+6	5.3 E-4	0.4699	2.100E-4-1.376E-3	
1.0E+15	1.5 E+5	5.0 E-4	0.5755	1.610E-4-1.609E-3	
1.0E+14	1.5 E+4	4.8 E-4	0.6645	1.275E-4-1.820E-3	
1.0E+13	1.5 E+3	4.5 E-4	0.7430	1.032E-4-2.015E-3	
1.0E+12	1.5 E+2	4.3 E-4	0.8139	8.475E-5-2.198E-3	
1.0E+11	1.5 E+1	4.0 E-4	0.8791	7.041E-5-2.371E-3	
1.0E+10		3.8 E-4	0.9398	5.903E-5-2.533E-3	
1.0E+ 9		3.6 E-4	0.9968	4.986E-5-2.688E-3	
1.0E+ 8		3.4 E-4	1.0507	4.237E-5-2.834E-3	
1.0E+ 7		3.2 E-4	1.1020	3.620E-5-2.972E-3	
1.0E+ 6		3.1 E-4	1.1510	3.106E-5-3.103E-3	

5. Phosphorus

Average concentration of phosphorus in the earth's crust

The natural abundance of phosphorus has been estimated by different authors as 1032 ppm $\pm$ 11 %. Estimations vary between a low 800 ppm by Niggli, 1948, to a high of 1200 ppm by Goldschmidt, 1937 ; Rankama and Sahama, 1949 and Lee and Yao, 1970.

Production and reserves

Production in 1971 amounted to 87.5 million tonnes of phosphate rock (1970 = 85.4 million tonnes) containing in the order of 13.6 % phosphorus. During the period 1954 - 1958 production was 31.7 million tonnes of phosphate rock, indicating an average growth over this period of 6.8 %.

Production came from 32 countries which in order of importance are :

U.S.A.	40.31 %
U.S.S.R. (concentrate 39 % P_2O_5)	13.31 %
(concentrate 19 - 25 % P_2O_5)	11.43 %
Morocco	13.72 %
Tunesia	3.61 %
Nauru Island	2.38 %
South Africa, Republic of	1.98 %
Togo	1.96 %
Senegal	1.77 %
China, People's Republic of	1.35 %
Vietnam, North	1.31 %
Christmas Island	1.24 %
Others (less than 1 million tonnes each)	5.63 %

Based on U.S.A. production statistics total world production to date amounts to more than 1.5 billion tonnes of phosphate rock containing approximately 200 million tonnes of phosphor.

World reserves are estimated at 6 billion tonnes phosphorus, divided as follows :

United States	23.33 %
Africa : sedimentary	58.33 %
igneous	.25 %
Latin America : sedimentary	1.33 %
igneous	.22 %
guano	.08 %
Near East	2.17 %
Asia : sedimentary	3.33 %
igneous	3.33 %
Australia	3.33 %
Others	4.30 %

About 75 % of the world's production of phosphate rock is from deposits of marine phosphorite. Individual beds may be several feet thick and contain over 13 % P, and the beds may extend over hundreds of square miles (5). The Permian Phosphoria Formation of the Western United States is a good example which contains about 56 % of the demonstrated reserves in the U.S.A. or .78 billion tonnes phosphorus. The phosphate zone is 150 feet thick and the main bed is 5 feet thick and contains nearly 14 % P (26,27).

Extrapolation of the average thickness and phosphorus content over the entire area underlain by the Phosphoria indicates a total potential resource of about 16 billion tonnes phosphorus, or more than 2.5 x the world's demonstrated phosphorus reserves. Of this 16 billion tonnes P, about 2 billion tonnes occur in phosphate rock containing some 14 % P, 6 billion tonnes P in rock averaging between 4.4 and 10.5 %,5 billion tonnes P in phosphate rock ranging between 10.5 and 13 % and the remainder in rock averaging less than 4.4 %. For the deposit as a whole an average grade of some 8.34 % P is indicated.

The land pebble phosphate of Polk and Hillsborough Countries in Florida is a gravel composed of 10 - 50 % of phosphate pebbles in a matrix of sand, clay and soft phosphate, along with fossil teeth and bones of land and marine animals. Production statistics from Florida indicate an average grade of some 6 % P; which easily can be upgraded to 13 %. Demonstrated reserves are in the order of 200 million tonnes of P and another 100 million tonnes of P has been inferred.

Other known deposits are of similar order of magnitude and it is indicated that resources of individual deposits should be measured in hundreds millions of tonnes phosphorus. No exact data on the extensive African deposits were found however and it is proposed to use the demonstrated reserves of the Phosphoria Formation as the largest single deposit found.

Long term average price

The average price for marketable phosphate rock during the period 1954 - 1970 has been $ 6.90 ± 6 % per tonne, containing an average of some 13.64 % P. As the United States is a net exporter of phosphates and no duties on imports exist, this price probably gives a fair representation of world prices. This would indicate a unit price of about $ 0.051 per kg of P contained.

Input data for the resources estimate by MIMIC

1. average concentration in the crust	1.E-3
2. concentration in largest ore deposit	1.4E-1
3. size of the environment	1.E18
4. size of largest deposit (tonnes P)	7.8E8
5. size of demonstrated reserve (tonnes P)	7.8E8
6. average long term price	0.051
7. name of estimated commodity	Phosphorus

Resources estimate

From the foregoing data a specific mineralizability of 0.197487 is determined for phosphorus. The corresponding target price of $ 0.050 per kg P contained is 0.99 times the long term average price (24,25).

With the indicated specific mineralizability the reserves and resources estimate of figure 8 could be made.

Inferred reserves are calculated at 12 billion tonnes of phosphorus or twice the recent USGS estimate (5). They include 6.2 billion tonnes in demonstrated reserves and former production. It can be seen that normal ore grades would require almost pure fluor apatite deposits, with theoretical phosphor percentage close to 18 %.

Actual grades of the ores are in the order of 13 % with a cut-off grade of some 10.5 %P (24 % P_2O_5). Such deposits would fall in the marginal resources at prices up to 1.33 S.P.U., or in deposits of very large size in which exploration costs are neglected. This actually is the case in most mining districts where cheaper than average mining methods can be employed.

Marginal resources at costs up to 1.33 and 1.67 S.P.U. are estimated at respectively 100 and 250 billion tonnes of P contained which is from 7 - 17 times as the recent USGS estimate on all identified and hypothetical resources together.

Table 7 illustrates the normal expectable concentration ranges for phosphorus in the natural environment as a function of sample size (median $\pm$ standard deviations). It can be seen that average concentrations greater than 0.6 % could be expected in about 2 % of the investigated areas whereas concentrations over 2 % P would have a natural chance of occurrence of 0.001 approximately.

Table 7

The total environment corresponds to the earth's dry land surface of 1.5E+8 square km to a depth of 2.5 km and an average specific gravity of 2.7.

Clarke P = 1.0 E-3 Specific Mineralizability = 0.197487

Environment tonnes	km^2	median	log SD	Normal range Median ± 2 SD	Remarks
1.0E+18	1.5 E+8	1.0 E-3			
1.0E+17	1.5 E+7	9.3 E-4	0.3647	4.815E-4-1.820E-3	Min.Max.
1.0E+16	1.5 E+6	8.7 E-4	0.5158	3.123E-4-2.458E-3	
1.0E+15	1.5 E+5	8.2 E-4	0.6317	2.318E-4-2.902E-3	
1.0E+14	1.5 E+4	7.6 E-4	0.7295	1.785E-4-3.302E-3	
1.0E+13	1.5 E+3	7.1 E-4	0.8156	1.406E-4-3.672E-3	
1.0E+12	1.5 E+2	6.7 E-4	0.8934	1.127E-4-4.016E-3	
1.0E+11	1.5 E+1	6.2 E-4	0.9650	9.140E-5-4.338E-3	
1.0E+10		5.8 E-4	1.0316	7.488E-5-4.640E-3	
1.0E+ 9		5.5 E-4	1.0942	6.185E-5-4.922E-3	
1.0E+ 8		5.1 E-4	1.1534	5.143E-5-5.186E-3	
1.0E+ 7		4.8 E-4	1.2097	4.301E-5-5.433E-3	
1.0E+ 6		4.5 E-4	1.2635	3.616E-5-5.663E-3	

6. Lead

Average concentration of lead in the earth's crust

The average concentration of lead in the earth's crust has been determined at 16 ppm by Rankama and Sahama, 1949, Green, 1959, and Vinogradov, 1962. A lower estimate of 13 ppm was made by Lee and Yao, 1970. Morris, Heyl and Hall, 1973 (5), 15 ppm lead as the approximate average concentration of lead in the earth's crust.

Production and reserves

In 1971 a total production of 3.4 million tonnes of lead was reached (1970 = 3.4 million tonnes) against an average annual production of 2.3 million tonnes of lead during the period 1954 - 1958, indicating an average annual growth of 2.64 %.

Production came from 51 countries which in order of importance were :

United States	15.42 %
U.S.S.R.	13.22 %
Australia	11.71 %
Canada	11.38 %
Peru	5.22 %
Mexico	4.61 %
Yugoslavia	3.68 %
China, People's Republic of	3.17 %
Bulgaria	2.93 %
Korea, North	2.35 %
Sweden	2.28 %
Marocco	2.18 %
Japan	2.07 %
Spain	2.04 %
South-West Africa	1.96 %
Poland	1.92 %
Others (less than 60.000 tonnes each)	13.86 %

A very large part of the lead reserves occur in combined lead-zinc deposits in which silver often is an important by-product. Mineral concentration often is given as lead equivalent and no exact ore-grade estimations for lead can be made from available literature. The Southeast Missouri District in the U.S.A., with some 27 million tonnes lead in estimated reserves is by far the largest predominantly lead district. The deposits contain between 3 - 8 % lead, .5 - 1.o % zinc, 0.1 % copper and minor amounts nickel, cobalt, cadmium, silver, germanium, and indium. The ore deposits, which belong to the type of stratiform, epigenetic deposits, represent the most common type of lead ore deposits and are replicated in many areas of the world with comparable grades. Although lead content of 20 % or more are not rare, the lead content of the majority of ore deposits appears to be situated between 5 and 6 % Pb. Due to co-products, in actual mining somewhat higher average lead equivalents may be found (6 - 8 % lead equivalent from combined Ph-Zn-Ag etc.).
Concerning the size of the average lead deposit it is rather difficult to make estimates for lead alone. In most cases combined lead-zinc deposits are described in literature. Bauchau, 1971 (30), has made a study on the size of lead-zinc deposits. He makes a distinction between ore deposits (corps) and ore fields (champs). According to his statistics on 212 deposits and 391 fields the average field is equivalent to approximately 3.43 deposits and his statistics can be combined to express this equivalency between deposits and fields. It was found that 50 % of the reserves occur in deposits or field containing 500.000 tonnes or more combined lead and zinc, whereas the median deposit contains 32.000 tonnes combined metal only.

World production of lead from prehistoric time until 1800 has been estimated at less than 5 million tonnes ; from 1801 until 1900 about 23 million tonnes and from 1901 to 1971 at some 117 million tonnes or a total of some 145 million tonnes.

Ore reserves which in 1963 were estimated at about 50 million tonnes (Callaway, 1962, Brinck, 1967) ; at about 84.4 million tonnes in 1965 (Bauchau, 1971) ; in 1971 were estimated at 128 million tonnes (5). It should be noted that the older estimates preceded the discovery and development of enormous lead deposits in Southeast Missouri, north-central Australia, Roumania and elsewhere during the past 10 years.

The demonstrated reserves of 128 million tonnes are divided as follows :

United States (21.28 % Southeast Missouri)	27.80 %
U.S.S.R.	12.77 %
Australia	13.13 %
Canada	11.25 %
Common Market	5.87 %
Mexico	3.19 %
Others (Asia, South America and Africa)	25.99 %

Long term average price

The average price of lead during the period 1954 - 1970 has been around 25 cents kg lead. This is the weighted average price from the New York and London prices.

The long term average price over 75 years (Mabile, 1968) has been about $ 0.27 kg Pb (1958 $ value) and $ .33 in 1971 $ value (Brinck, 1971). For the purpose of this study the long term average price of .27 $ Kg lead will be used.

Input data for the resources estimate by MIMIC

1. average concentration in the crust	1.6 E-5
2. concentration in ave.ore deposit	6. E-2
3. size of the environment	1. E18
4. size of the average ore deposit	5. E5
5. size of production + 0.5 x reserves	2.1 E8
6. average long term price	0.27
7. name of estimated commodity	Lead

Resources estimate

From the foregoing data a specific mineralizability of 0.286245 is determined for lead. This is a somewhat higher than the value of 0.2793 which had been determined on the basis of the demonstrated reserves in 1963, indicating that important deposits of higher than average grade and size have been discovered since. The target price of $ 0.32 per kg lead is 0.98 times the average long-term price as used in the original estimate and 1.2 times the long term price as used for the current estimates (24,25).

With the indicated specific mineralizability the reserves and resources estimate of figure 9 could be made.

Inferred reserves are calculated at 500 million tonnes lead and include demonstrated reserves plus former production of 270 million tonnes which is essentially the USGS estimate of identified reserves including some inferred reserves. The profit maximum is calculated for 184 million tonnes of lead in average deposits containing 960.000 tonnes of lead at an average grade of 5.17 % Pb.

Inferred reserves at prices up to 1.33 and 1.67 S.P.U. respectively are estimated at 2 - 5 billion tonnes, which is only from 1 - 3 times the USGS estimate on the total of identified, conditional (paramarginal), hypothetical and speculative resources. The main contribution here is made by the conditional resources of lead as co-product of manganese nodule exploration from the ocean floor and therefore is not strictly comparable with the resources indicated for the upper 2.5 km of the dry land surface of the earth.

Table 8 illustrates the normal expectable concentration ranges of lead in the natural environment as a function of sample size (median $\pm$ 2 standard deviations)

It can be seen that concentrations greater than 120 g/t could be expected in about 2 % of the investigated areas whereas concentrations of over 770 g/tonne would have a natural chance of occurence of only 0.001. In many cases, however, the chemical compound in which the lead is found could decide at much lower concentrations whether or not the concentration should be considered as a form of pollution.

Table 8

The total environment corresponds to the earth's dry land surface of 1.5 E+8 square km to a depth of 2.5 km and an average specific gravity of 2.7.

Clarke PB = 1.6 E-5   Specific mineralizability = 0.286245

Environment tonnes	km^2	median	log SD	normal range Median ± 2 SD	remarks
1.0E+18	1.5 E+8	1.6 E-5			
1.0E+17	1.5 E+7	1.3 E-5	0.5367	5.219E-6-3.692E-5	min.max.
1.0E+16	1.5 E+6	1.2 E-5	0.7590	2.639E-6-5.496E-5	
1.0E+15	1.5 E+5	1.0 E-5	0.9296	1.628E-6-6.708E-5	
1.0E+14	1.5 E+4	9.0 E-6	1.0734	1.059E-6-7.759E-5	
1.0E+13	1.5 E+3	7.8 E-6	1.2001	7.135E-7-8.673E-5	
1.0E+12	1.5 E+2	6.8 E-6	1.3147	4.923E-7-9.463E-5	
1.0E+11	1.5 E+1	5.9 E-6	1.4200	3.460E-7-1.014E-4	
1.0E+10		5.1 E-6	1.5180	2.467E-7-1.070E-4	
1.0E+ 9		4.4 E-6	1.6101	1.781E-7-1.116E-4	
1.0E+ 8		3.8 E-6	1.6972	1.298E-7-1.152E-4	
1.0E+ 7		3.3 E-6	1.7801	9.542E-8-1.180E-4	
1.0E+ 6		2.9 E-6	1.8592	7.067E-8-1.200E-4	

References

1. Brinck J.W. : "Mimic", Eurospectra X, nr 2, pp 46-56, June 1971.

2. " : "Prediction of mineral resources and long-term price trends in the non-ferrous metal mining industry", Vol. IV pp 3-15, Proc. 24th Session of the International Geological Congress, Montreal 1972. (Eur. Doc. 116/XVII/71-E)

3. CGE : "Second Target Program for Nuclear Energy in the Community" Annex IV The fuel cycle (July 1972).

4. Brinck J.W. : "An illustrative example of normal growth from a modified world model", Eurospectra XIII, 2, pp 47-51, June 1974.

5. USGS : "United States Mineral Resources"; Geological Survey Professional Paper 820, Washington 1973, Library of Congress catalog-card N° 73-600060.

6. IRC : "Program IRC"004, IRC - P.O.Box 471 - Alkmaar - Netherlands. 1974.

7. Brinck J.W. : "The geochemical Distribution of Uranium as a primary Criterion for the formation of ore deposits" Proc. IAEA symposium on the "Formation of Uranium Ore Deposits", Athens, May 1974.

8. " & Van Wambeke L. : "World resources of Mercury" Proc. First International Mercury Congress, Barcelona, May 1974.

9. Blondel F. & Lasky S.G. : "Mineral reserves and Mineral resources" Economic Geology 51 (7), 1956.

10. NEA - IAEA : Uranium, Resources, Production and Demand, Paris, 1973.

11. Ahrens L.H. : "The log-normal distribution"-Geochemica & Cosmochimica Acta - Vol. 5, nr 2, 1954.

12. Matheron G. : "Traité de Géostatistique Appliquée" Tomes I,II,BRGM - Mémoires N° 14 & 24 ; Paris 1962, 1963.

13. De Wijs H.J. : "Die statistische Auswertung der Probenahme von Erzlagerstätten; Erzmetall, September 53.

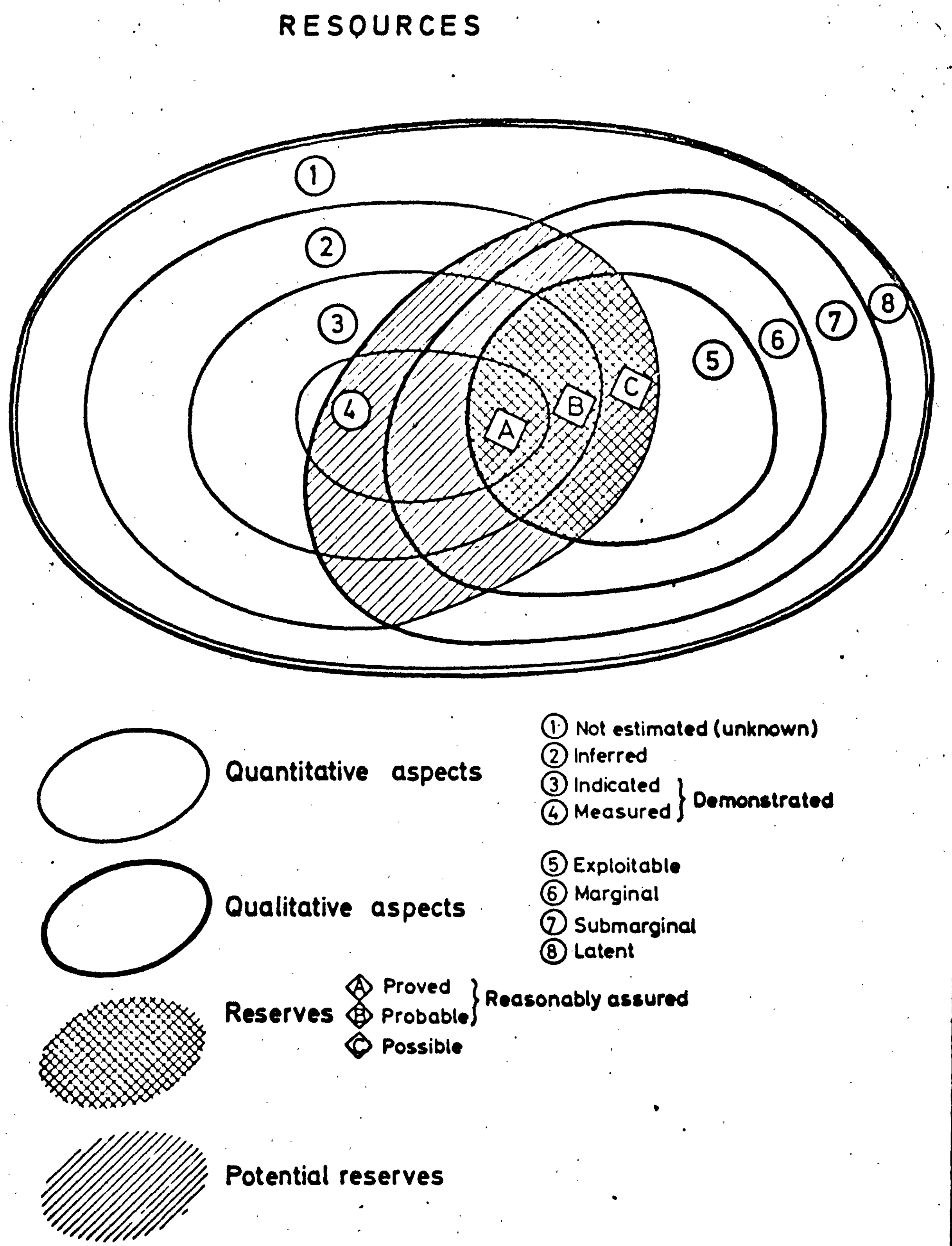
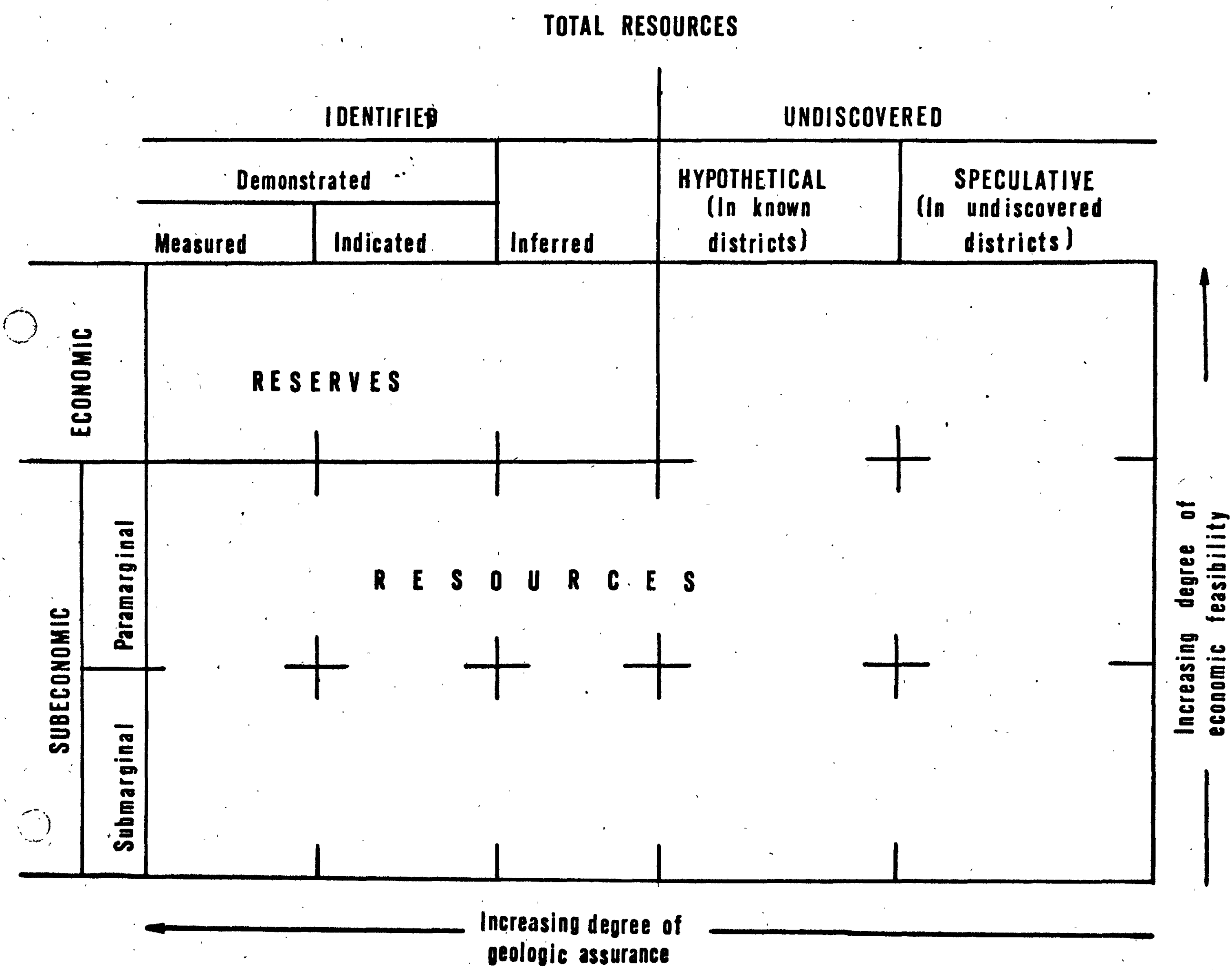

RESOURCES
1
2
3
4
5
6
7
8
A
B
C
Quantitative aspects
Qualitative aspects
Reserves
Potential reserves
1 Not estimated (unknown)
2 Inferred
3 Indicated
4 Measured
Demonstrated
5 Exploitable
6 Marginal
7 Submarginal
8 Latent
A Proved
B Probable
C Possible
Reasonably assured

Fig. 1

Figure 2.—Classification of Mineral Resources



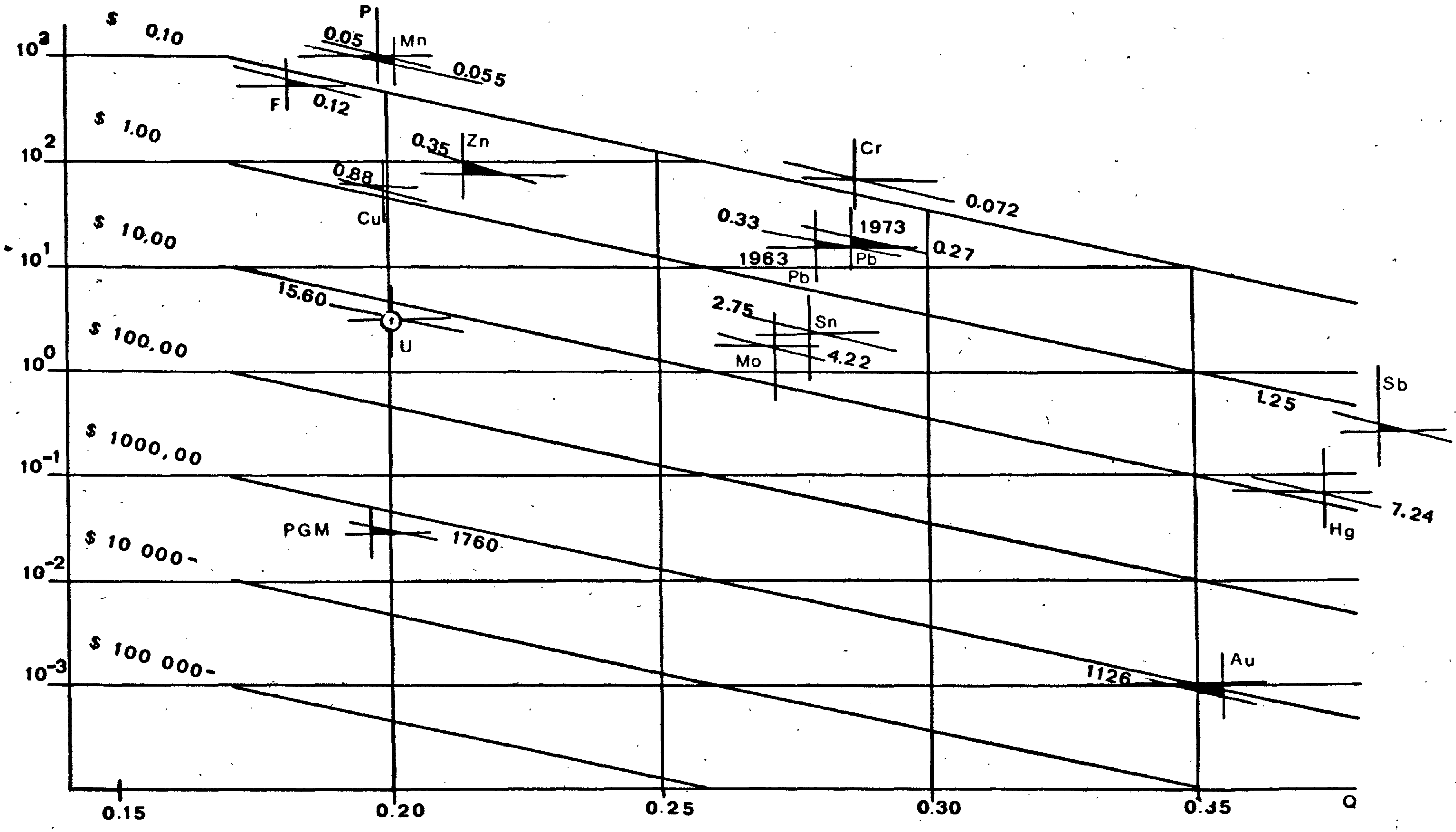
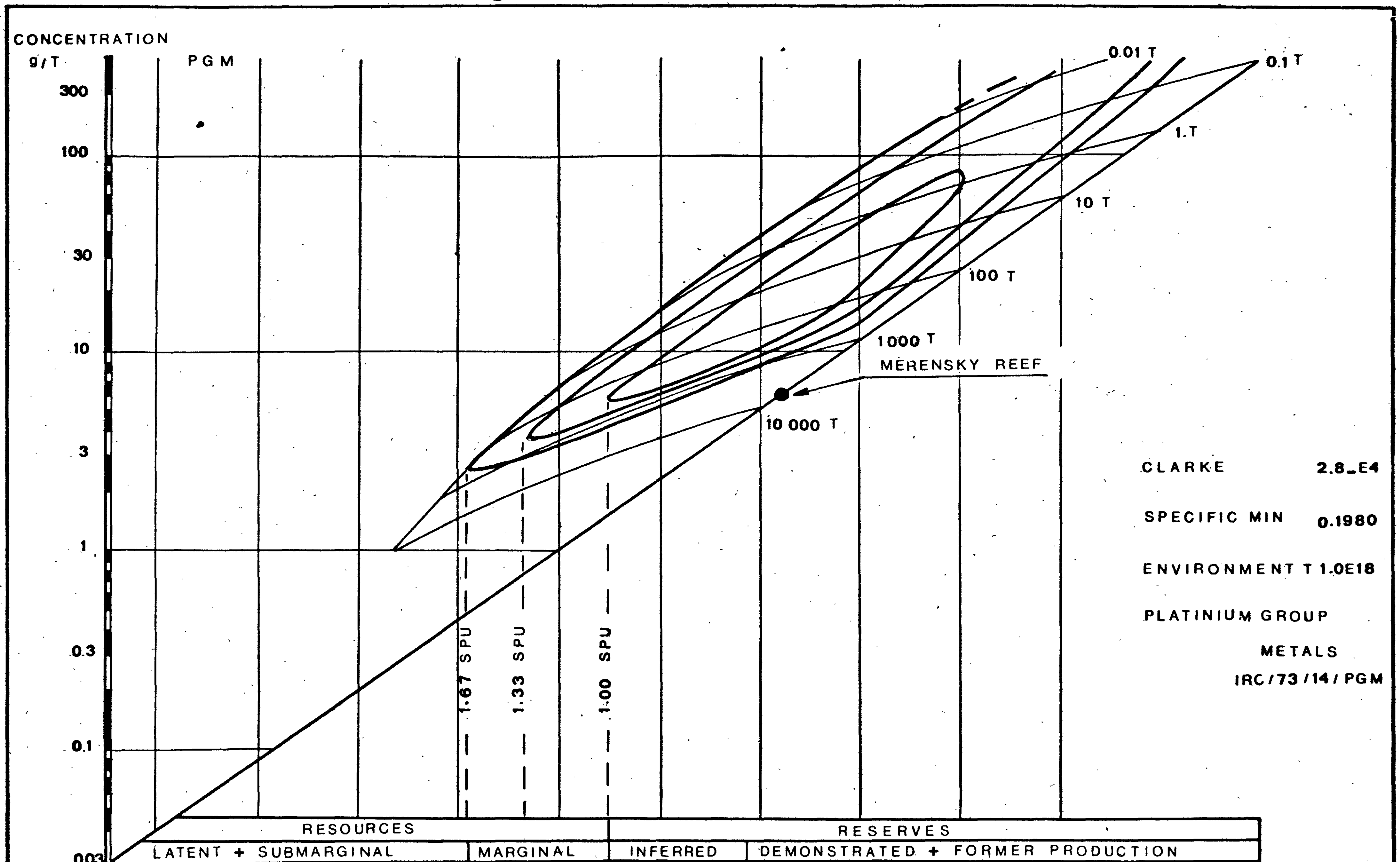

$ 0.10
$ 1.00
$ 10.00
$ 100.00
$ 1000.00
$ 10 000-
$ 100 000-
10^3
10^2
10^1
10^0
10^-1
10^-2
10^-3
0.15
0.20
0.25
0.30
0.35
Q
P
0.05
Mn
0.055
F
0.12
0.35
Zn
0.88
Cu
Cr
0.072
0.33
1973
1963
Pb
Pb
0.27
15.60
U
2.75
Sn
Mo
4.22
Sb
1.25
Hg
7.24
PGM
1760
Au
1126

FIG. 3



CONCENTRATION
g/T
PGM
300
100
30
10
3
1
0.3
0.1
0.03
0.01 T
0.1 T
1.T
10 T
100 T
1000 T
MERENSKY REEF
10 000 T
1.67 SPU
1.33 SPU
1.00 SPU
RESOURCES
RESERVES
LATENT + SUBMARGINAL
MARGINAL
INFERRED
DEMONSTRATED + FORMER PRODUCTION
10
9
8
7
6
5
4
3
2
1
LOG TONNES PGM
CLARKE 2.8_E4
SPECIFIC MIN 0.1980
ENVIRONMENT T 1.0E18
PLATINIUM GROUP
METALS
IRC/73/14/PGM

FIG. 4

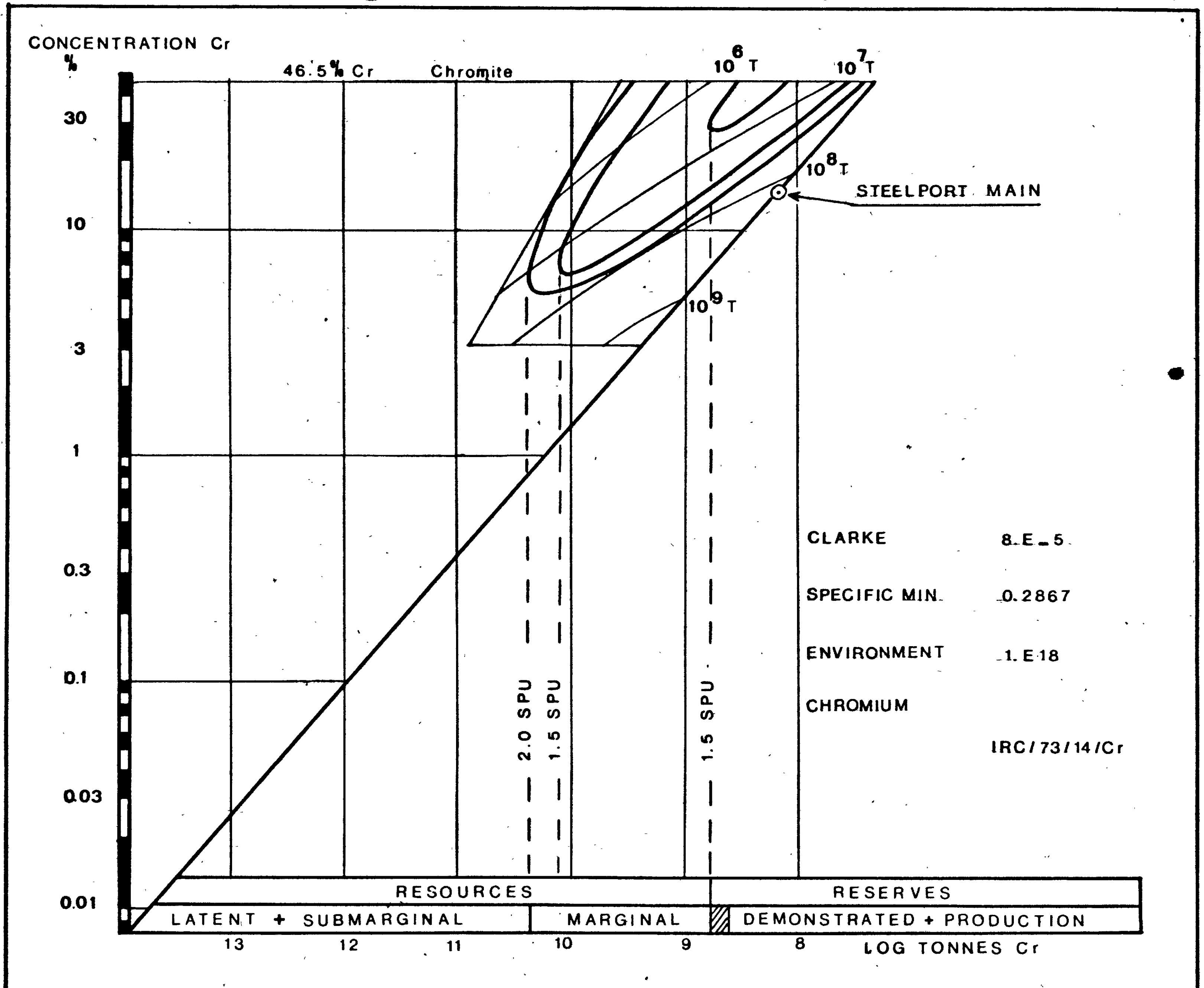

CONCENTRATION Cr
%
46.5% Cr
Chromite
10^6 T
10^7 T
10^8 T
10^9 T
STEELPORT MAIN
30
10
3
1
0.3
0.1
0.03
0.01
2.0 SPU
1.5 SPU
1.5 SPU
CLARKE 8.E-5
SPECIFIC MIN. 0.2867
ENVIRONMENT 1.E18
CHROMIUM
IRC/73/14/Cr
RESOURCES
RESERVES
LATENT + SUBMARGINAL
MARGINAL
DEMONSTRATED + PRODUCTION
13
12
11
10
9
8
LOG TONNES Cr

FIG.5

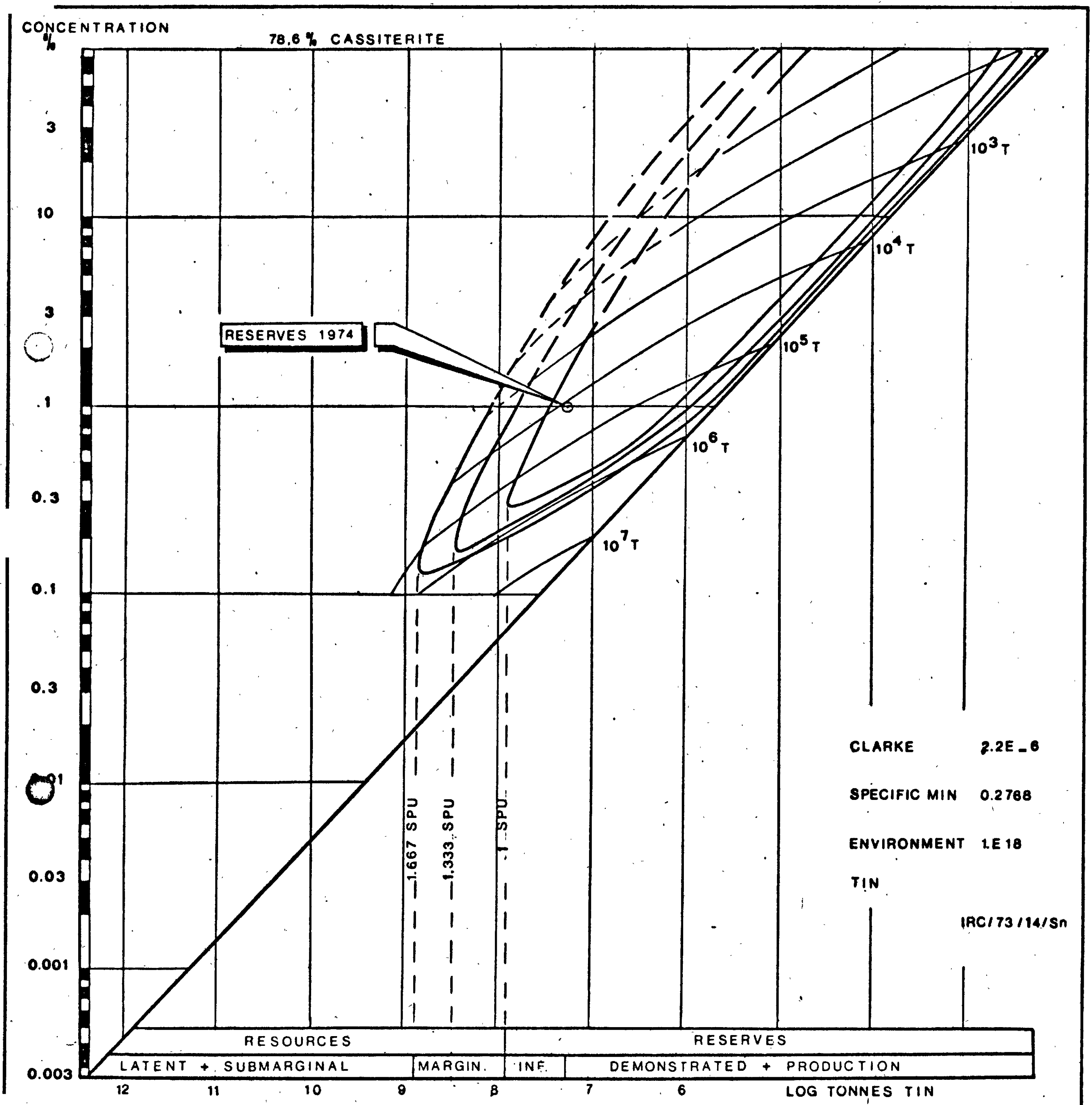

CONCENTRATION %
78.6 % CASSITERITE
RESERVES 1974
10^3 T
10^4 T
10^5 T
10^6 T
10^7 T
1.667 SPU
1.333 SPU
1 SPU
CLARKE 2.2E-6
SPECIFIC MIN 0.2768
ENVIRONMENT 1.E 18
TIN
IRC/73/14/Sn
RESOURCES
RESERVES
LATENT + SUBMARGINAL
MARGIN.
INF.
DEMONSTRATED + PRODUCTION
LOG TONNES TIN

FIG.6

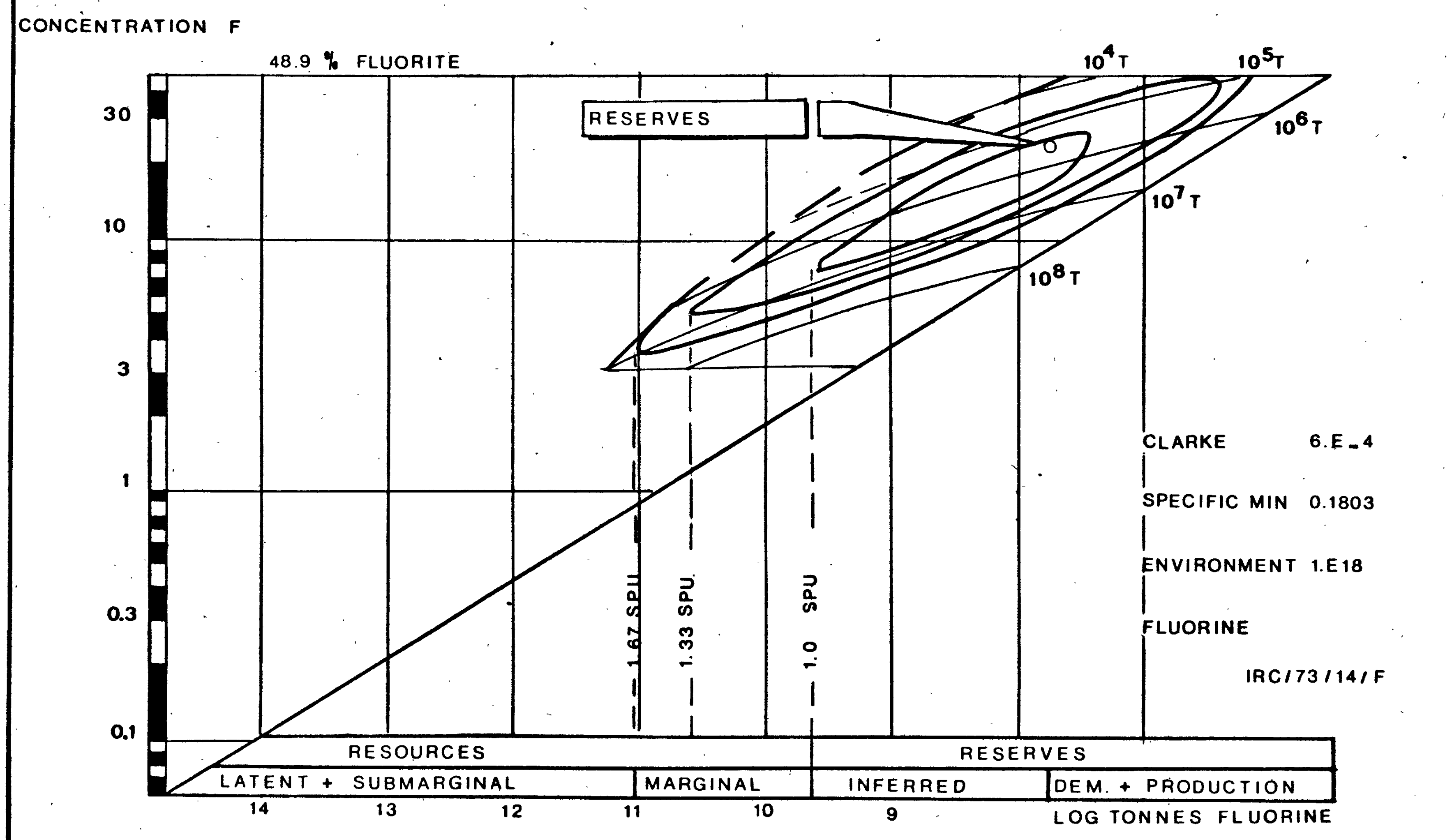
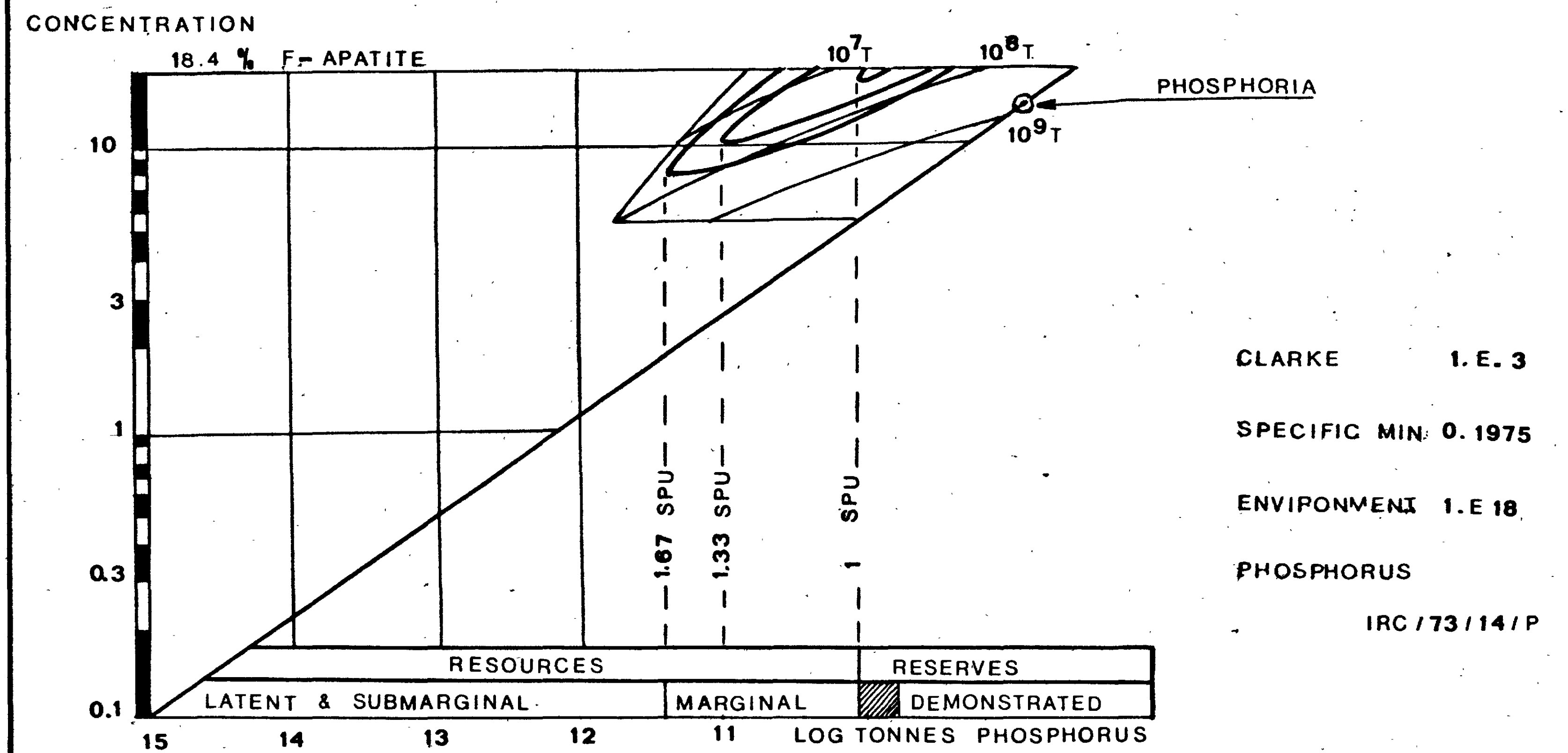
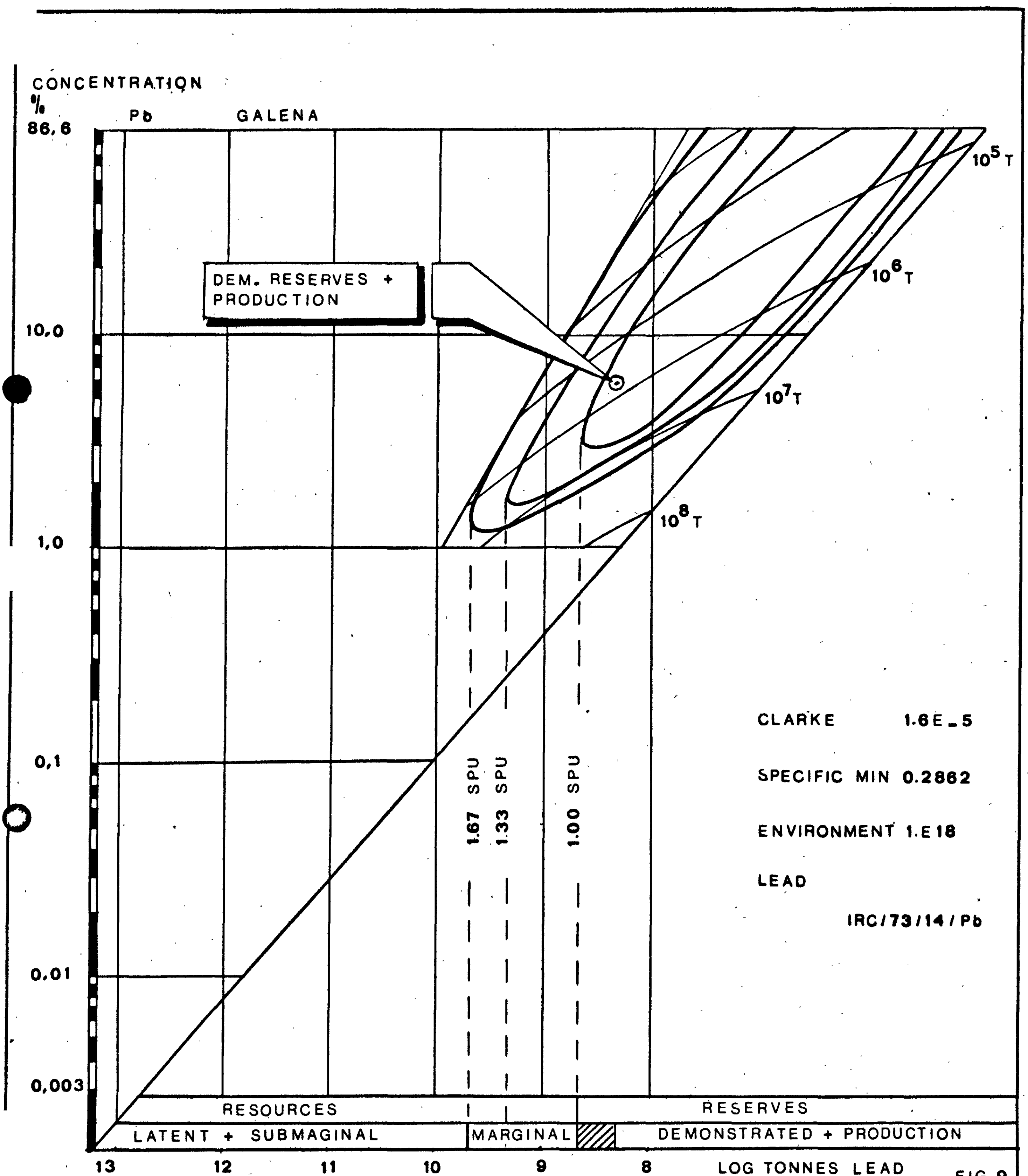

CONCENTRATION F
48.9 % FLUORITE
30
10
3
1
0.3
0.1
RESERVES
10^4 T
10^5 T
10^6 T
10^7 T
10^8 T
1.67 SPU
1.33 SPU
1.0 SPU
CLARKE 6.E-4
SPECIFIC MIN 0.1803
ENVIRONMENT 1.E18
FLUORINE
IRC/73/14/F
RESOURCES
RESERVES
LATENT + SUBMARGINAL
MARGINAL
INFERRED
DEM. + PRODUCTION
14
13
12
11
10
9
LOG TONNES FLUORINE

FIG.7



CONCENTRATION
18.4 % F- APATITE
10
3
1
0.3
0.1
15
14
13
12
11
LOG TONNES PHOSPHORUS
10^7T
10^8T
10^9T
PHOSPHORIA
1.67 SPU
1.33 SPU
1 SPU
RESOURCES
RESERVES
LATENT & SUBMARGINAL
MARGINAL
DEMONSTRATED
CLARKE 1. E. 3
SPECIFIC MIN. 0. 1975
ENVIPONMENT 1. E 18
PHOSPHORUS
IRC / 73 / 14 / P

FIG. 8



CONCENTRATION
%
Pb
GALENA
86,6
10,0
1,0
0,1
0,01
0,003
DEM. RESERVES + PRODUCTION
10⁵ T
10⁶ T
10⁷ T
10⁸ T
1.67 SPU
1.33 SPU
1.00 SPU
CLARKE 1.6E-5
SPECIFIC MIN 0.2862
ENVIRONMENT 1.E18
LEAD
IRC/73/14/Pb
RESOURCES
RESERVES
LATENT + SUBMAGINAL
MARGINAL
DEMONSTRATED + PRODUCTION
13
12
11
10
9
8
LOG TONNES LEAD

FIG.9