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# SOVIET NONFERROUS METALLURGY

NO. 17

SELECTED TRANSLATIONS



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### Introduction

This is a serial publication containing selected translations on nonferrous metallurgy in the Soviet Union. This report consists of translations on subjects listed in the table of contents below.

### Table of Contents

## 1. Efficient Ways of Utilizing the Pyrite Concentrates of Polymetallic and Copper-Pyrite Ores of the Altay Mining Region

/Following is the translation of an article by N. Satybaldin entitled "Puti Effektivnogo Ispol'zovaniya Piritnykh Kontsentratov Polimetallicheskikh i Mednokolchedannykh Rud Rudnogo Altaya" (English version above) in Vestnik Akademii Nauk Kazakhskoy SSR (Herald of the Academy of Sciences Kazakh SSR), No. 3, Moscow, March 1960, pages 3-10./

Efficient processing of the pyrite concentrates obtained from the beneficiation of the polymetallic ores of Altay and particularly the cupropyritic rib ores of the Nikolayevskoye Deposit, is of decisive importance to a rational utilization of the bulk of the sulfur and iron in these ores.

At present pyrite concentrates are obtained only in the Leninogorsk and Zyryanovsk concentrator plants. It was originally planned that the pyrite concentrates of these plants would be delivered to the Dzhambul Superphosphate Plant for the production of sulfuric acid with subsequent use of the pyrite calcine in lieu of imported iron fluxing agents in the lead plants of Altay. However, in the long run, the demand of the enterprises of Dzhambul and other consumers in Southern and Central Kazakhstan could be satisfied by the pyrite concentrates of the Tekelinyskiy (100,000 tons), Balkhash (200,000 tons), Achisayskiy (35,000 tons), and other concentrator plants. Moreover, the use of the refuse of slag distillation in lieu of iron fluxing agents, and the conversion of the Leninogorsk Plant to electric smelting as well, will result in reducing to its minimum the demand of the lead plants of Altay for iron fluxes and sulfur pyrites. Therefore, the planned path of utilization of the Altay pyrite concentrates is an interim measure, considering moreover that their output will greatly increase in the long run.

The present seven-year period in Altay will be a period of the development of the Nikolayevskoye Deposit, whose ores consist of rib-type copper pyrites containing on the average 38 percent sulfur, 40 percent iron, and a small percentage (10-15 percent) of gangue. The direct metallurgical processing of these ores considerably complicates technology and, most important, it requires the large-scale expansion of the metallurgical capacities of Altay and hence also sizable capital investments. The most realistic and efficient method of beneficiating the Nikolayevskoye ores appears to be flotation, resulting in copper and zinc concentrates and semifinished bulk products which are transmitted for final [metallurgical] processing to the existing plants in Altay. While it is not possible to obtain selective concentrates, the metallurgical processing of even bulk concentrates is much easier and cheaper than that of low-grade bulk ore.

The flotation of these ores will lead to the isolation, in the form of flotational tailings, of a large amount of pyrite concentrates, plus over 20 percent of the nonferrous metals and a still higher percentage of rare and dispersed elements.

The effectiveness of such a technological variant of the processing of Nikolayevskoye ores is basically determined by the possibilities for a rational utilization of these pyrite concentrates.

According to the data of the Giprotsvetmet /State Institute for Design and Planning of Nonferrous Metallurgy Establishments/, by the end of the seven-year period pyrite concentrates will be produced in large quantities at the Leninogorsk, Zyryanovsk, Zolotoushinskiy, and Nikolayevskiy concentrator plants, and in the concentrator plants of the Irtysh Polymetal Combine as well. Within a longer perspective, by 1970, when the Nikolayevskoye Deposit will be developed to full projected capacity, and when the extraction in the already existing mines will rise further, the amount of the Altay pyrite concentrates will increase still further.

The problem of utilizing the sulfur and iron of the Altay pyrite concentrates has to be solved, on the one hand, in close consonance with the plan of the development and geographical distribution of the sulfuric-acid and acid-consuming branches of industry and ferrous metallurgy as well, and, on the other hand, on taking into account the efficient recovery of their content of nonferrous metals: lead, zinc and copper, and rare and dispersed elements too.

There exist two principal ways of utilizing pyrite concentrates: their oxidizing roasting for the production of sulfuric acid from sulfur-bearing gases, and the retreatment of pyrite concentrates to obtain elementary sulfur. In both cases it will be necessary to retreat the pyrite cinders for the purpose of recovering nonferrous and precious metals and utilizing the ferriferous residue.

The principal consumer of both pyrite concentrates and elementary sulfur is the sulfuric-acid industry. In 1960 nearly 85 percent of the pyrite concentrates and over one-half of the elementary sulfur will be consumed for the production of sulfuric acid. At present 93 percent of the total amount of floated and sulfur pyrites consumed for the production of sulfuric acid in this country is delivered by the Urals, and hence the national economy sustains considerable losses because of the expenditures involved in hauling Ural pyrites over long distances.

Elementary sulfur is a concentrated raw material and, compared with pyrite, it can be much more efficiently hauled over large distances. However, an appraisal of the effectiveness of obtaining elementary sulfur from pyrite concentrates should not ignore the qualitative and quantitative changes in the balance sheet of the production and consumption of elementary sulfur which will occur in the course of the present seven-year period. The plan of the development of the large chemical industry during the seven-year period provides for a sizable increase in the volume of the extraction of natural sulfur, activation in the immediate future of the country's largest sulfur combine on the Razdel'skoye Natural Sulfur Deposit, and considerable expansion of the capacities of the existing sulfur combines. A large deposit of natural sulfur has been discovered in L'vovskaya Oblast. The planned volume of extraction of natural sulfur radically alters the balance sheet of the production and cost of elementary sulfur for the country as a whole. The prices of natural elementary sulfur will be cut.

In the opinion of the workers of the Giprokhim /State Institute for Design and Planning of Chemical Industry/ there exists a virtually unlimited possibility for increasing the extraction of natural elementary sulfur and supplying it to users of sulfur and sulfuric acid who are located in remote regions.

Under such conditions, the production of elementary sulfur from the Altay pyrite concentrates and its reroasting to obtain sulfuric acid could be justified only in the presence of an economical method of reciveing sulfur and a substantial reduction in the expenditures on sulfur-bearing materials.

At present the NIUIF Institute has developed and tested a most economical method of recovering sulfur from pyrite concentrates on interacting it with circulating sulfurous anhydrite in a fluidized bed of calcine.

A comparison of the effectiveness of the production of sulfuric acid from pyrite concentrate (45 percent) and from the elementary sulfur recovered from the same concentrate by the NIUIF method is illustrated by the table below.

Table

Indexes of the Production of Sulfur and Sulfuric Acid*

| Index | Production of Sulfuric Acid From Pyrite Concentrate | | Production of Sulfur from Pyrite Concentrate and Sulfuric Acid from Sulfur | |
|---|---|---|---|---|
| | Amount, tons | Sum, rubles | Amount, tons | Sum rubles |
| Production of Sulfur | | | | |
| Expenditure on raw material | - | - | 2.23 | 149.8 |
| Shop Production Cost | - | - | - | 121.0 |
| Total, Production Cost per ton of Sulfur | - | - | - | 270.0 |
| Production of Sulfuric Acid | | | | |
| Expenditures on Raw Material | 0.82 | 67.0 | 0.35 | 95.0 |
| Shop Production Cost | | 90.0 | - | 73.0 |
| Total, Production Cost per ton of Sulfuric Acid | - | 145.0 | | 168.0 |
| Unit Capital Expenditures per ton of Sulfuric Acid | - | 329.0 | - | 320.0 |
| In which, Unit Capital Expenditures on the Production of Sulfur in terms of per ton of Sulfuric Acid | - | - | - | 78.0 |

* Table compiled on the basis of design and research data of the Giprokhim and NIUIF.

The comparison of indexes cited in the above table indicates that, in the final analysis, the recovery of sulfur and its roasting into sulfuric acid increases the production cost of the acid by 16-17 percent. The unit capital expenditures in both cases are nearly identical. Thus, the recovery of sulfur in elementary form is expedient only in the event of substantial savings as a result of the replacement of the long-distance hauls of pyrite concentrates by hauls of elementary sulfur.

The utilization of the Altay pyrite concentrates in situ to produce sulfuric acid requires, in view of the absence of any large local users, hauls of considerable quantities of the acid over large distances. The establishment of the production of superphosphate in the Altay to utilize these pyrite concentrates also does not seem expedient. The amount of sulfuric acid obtained in the Ust'-Kamenogorsk Lead Zinc Combine and Leninogorsk Polymetals Combine alone will quite suffice for the organization of the production of superphosphate without having to resort to using pyrite concentrates for this purpose. This amount exceeds nearly twofold the economically expedient volume of production of superphosphates necessary for ensuring the demand of not only the entire Irtysh River Region but also a part of West Siberia.

The retreatment of pyrite concentrates into sulfuric acid in situ excludes the possibility of an efficient utilization of the iron-bearing residue of the processing of pyrite cinders for the purpose of recovering from it nonferrous metals. The absence of any base of ferrous metallurgy in the Altay Mining Region prevents the retreatment of these residues into foundry pig iron. Unrealistic also is the proposal* for retreating pyrite cinders into foundry pig iron in electric furnaces. The smelting of pig iron both in electric furnaces and in the electric blast furnace has not as yet found application in indigenous practice. Therefore, the attempt at conducting this smelting under the conditions of the Altay Mining Region would require the conduct of a large volume of research and testing work which requires not only effort and funds but also much time.

The creation of a large hydroelectric power capacity in the Altay is necessitated by the effectiveness of developing in that region primarily the electricity-consuming types of the production of the nonferrous and rare metals (lead, copper, etc.) needed by the country. Therefore it seems inexpedient to consume a large amount of

*"Rudnyy Altay," No. 3-4, 1958, pages 9-10.

electrical energy for processes that are of low effectiveness under the conditions of the Altay, such as the smelting of pig iron in electric furnaces. This is even more so because the local machine building plant and repair and machine base, whose operations are basically on a rayon-wide scale, do not need a base of their own for producing foundry pig iron.

The theses of the report of N. S. Krushchev to the 21st CPSU Congress underscore the need for a resolute struggle "against the pork-barrel concept of the complex economy as a closed economy."

The question of the utilization of pyrite concentrates should be solved on taking into account not only the comprehensive utilization of all useful components but also a broad cooperation of the Altay Mining Region with the enterprises of the chemical and metallurgical industry in other regions, particularly West and East Siberia. The demand for sulfuric acid in West Siberia will grow nearly eightfold during the seven-year period, and in East Siberia -- 4.2 times.

In our opinion, the most rational utilization of the Altay pyrites would consist in their retreatment at a site in West Siberia. In Kemerovo alone, in connection with the activation of new capacities at the local nitrogenous fertilizers combine and aniline-dye plant, the additional demand for /sulfuric/ acid will grow considerably by 1965. The production of sulfuric acid requires a great quantity of pyrite concentrates, which will be fully ensured by the Altay pyrites.

Calculations of the Giprokhim show that the additionaldemand for sulfuric acid in West Siberia in 1970 will rise to 0.9-1.0 million tons in terms of monohydrate. Of this amount only an insignificant part is covered by the acid produced in situ from sulfur-bearing waste gases. Imported Altay pyrite concentrates could be utilized for producing the remainder.

The most suitable site for organizing the retreatment of pyrite concentrates is Kemerovo, where enterprises consuming the bulk of that acid in West Siberia are located. This site is also convenient in view of the fact that the surplus part of the acid (35-40 percent) could be transported to adjacent users, located in a radius of 100-550 km from Kemerovo and, most important, because it makes possible the centralized retreatment of pyrite cinders at the Kuznetsk Metallurgical Combine. In this connection, the amount of pyrite cinders would be quite sufficient for the construction of a large hydrometallurgical shop.

A second site for the retreatment of the concentrates should be organized in the second-largest center of major users of sulfuric acid in West Siberia. In this connection, the retreatment of pyrite cinders from that site would also be conducted by the Kuznetsk Metallurgical Combine.

Another version could also be adopted: for the sake of the centralization of the retreatment of pyrite cinders, and also to avoid the hauls of the surplus acid to regions in the radius of 350-500 km, a part of the pyrite concentrates should be processed into elementary sulfur and shipped to the users of sulfuric acid in West Siberia and in East Siberia as well. The cinders remaining after the production of the sulfur could be retreated together with the cinders of the production of sulfuric acid.

The exports of pyrite concentrates from the Altay to West Siberia would, as will be shown below, result in the possibility of an efficient utilization of the iron-bearing residue of pyrite cinders, and therefore the expenditures on the transport of pyrite concentrates could be divided between sulfur and iron, according to their content in the concentrates. In this case, indisputably, the production cost in West Siberia of the sulfuric acid obtained from imported pyrite concentrates will be 16-17 percent lower than the production cost of the acid obtained from the imported elementary sulfur recovered from pyrite concentrates in the Altay.

The practical materialization of such an utilization of pyrite hinges on the pace of development of the technological processes of the retreatment of pyrite cinders. The most prepared of these is the classical chloride method, which has been the subject of numerous research and development studies in our country.

At present the Giproalyuminiy /State Institute for Design and Planning of the Aluminum Industry/ is drafting the design of a metallurgical shop for retreating the pyrite cinders of the chemical plants of the Central USSR Region, which will be built at the Novo-Tula Metallurgical Plant. The technology of that shop will be based on the chloride method. However, in contrast with the Duisburg (West Germany) Plant, which retreated a pyrite calcine containing 2.07 percent copper, 1.29 percent lead, and 2.7 percent zinc, the Novo-Tula Plant will utilize sulfuric acid-shop cinders containing 0.5 percent copper, 0.075 percent zinc, and 0.012 percent lead. The technology of the retreatment of pyrite cinders at the Novo-Tula Plant will provide for the recovery of copper and precious metals in the form of metals and of sulfur in the form of a by-product -- sodium sulfate, with simultaneous recovery of high-quality iron-bearing raw material which after sintering would proceed to blast furnaces. According to that technology pyrite cinders will together with table salt (7.8 percent by weight of cinders) be subjected to chlorination in rotary furnaces at a temperature of 350-400°C. The chlorinated cinders will be leached by water and reflux liquors in percolation tanks. In this connection copper, silver, and an insignificant part of gold will be extracted into the solution. To complete the recovery of precious metals the calcine is subjected to secondary leaching by chlorinated water. Thereupon copper, gold, and silver are precipitated from the solution by cementation. The method of the percolational leaching of calcine, adopted according to the granulometric composition of the calcine (cinders) of run-of-the-mine pyrite, is not suitable for leaching the chlorinated calcine of floated pyrites (pyrite concentrates), which at present account for the greater part of the sulfur-bearing materials used in the production of sulfuric acid. Therefore, at present other leaching methods are undergoing tests, particularly leaching in mechanical tanks. The leaching of the entire quantity of chlorinated cinders requires a large amount of equipment.

The capital-investment cost of the hydrometallurgical shop designed for the Novo-Tula Plant will amount to several million rubles. The high effectiveness of the retreatment of pyrite cinders is attested by the fact that, despite their very low content of nonferrous metals, the value of marketable output recovered from the pyrite cinders of the sulfuric-acid shops amounts to 104 rubles per ton, at total production expenditures of 81 rubles per ton (including 18.6 rubles of expenditures on raw material).* In this connection, the recovery of copper will reach 85 percent, gold -- 50 percent, and silver -- 70 percent.

The retreatment of pyrite cinders at the Novo-Tula Plant will yield a large amount of high-quality raw material -- "red ore." If the unit capital expenditures on the extraction of Tula and Krivoy Rog iron ore, on which the Novo-Tula Plant operates, amount to at least 100 rubles per ton, then the capital expenditures on the "red ore" of the hydrometallurgical shop will not be very high in comparison. The project of the Kuchukskiy Combine for the Extraction and Processing of Mirabilite, which is under construction in West Siberia, envisages that the unit capital expenditures on the production of sodium sulfate

* Giproalyuminiy Institute. "Technical Project of a Hydrometallurgical Shop for the Novo-Tula Plant," July 1957.

will amount to 490 rubles per ton. Thus, the capital expenditures on the construction of a shop for retreating pyrite cinders are nearly completely compensated by the savings in the capital expenditures on the organization of the independent fabrication of only two (sodium sulfate and "red ore") products of the retreatment of pyrite cinders, which account for 80.0 percent of the sum total of marketable output.

The unit capital expenditures on the extraction and beneficiation of the Gornaya Shoriya and Khakasiya ore at the Kuznetsk Metallurgical Combine amount to 465 rubles per ton of beneficiated ore containing 55-57 percent iron, while at the Nizhne-Angara and Angara-Ishim Deposits these expenditures amount to 207 and 272 rubles, * respectively, per ton of ore containing 49 and 48 percent iron, respectively. The construction and activation of a shop for retreating pyrite cinders at one of the metallurgical plants of West or East Siberia, a shop with the annual capacity of 700,000-800,000 tons of pyrite cinders (which will also partly retreat the refuse cinders of the sulfuric-acid shops of the given region) requires only 100-110 million rubles of capital expenditures, whereas the capital expenditures on a corresponding increase in the output capacity of iron ore enterprises in the same region would amount to 200-250 million rubles.

Calculations of the Gidroalyuminiy Institute show that even the retreatment of pyrite cinders that are very poor in their content of nonferrous metals (0.5 percent Cu, 0.075 percent Pb, 0.012 percent Zn) makes it possible to obtain iron-bearing raw material at the cost of 51 rubles per ton** on taking into account the expenditures on sintering -- 84 rubles per ton, whereas the production cost of a ton of beneficiated ore, on taking into account the expenditures on sintering, at the Kuznetsk Metallurgical Combine, amounts to 97 rubles. Here it should be kept in mind that this cost includes the cost of cinders, appraised by the "Gidroalyuminiy" at 18 rubles per ton, which, together with the cost of sulfur in pyrite concentrates, will be deducted from the sum total of the operating expenditures on the extraction and processing of, e. g.,

*"Iron Ore Base of Ferrous Metallurgy USSR," 1957, page 557.

** The expenditures on retreating cinders are divided among the various products according to the share of each product in the total marketable output, in sales prices.

Nikolayevskoye ores. The retreatment of the pyrite cinders obtained from Nikolayevskoye pyrite concentrates, containing approximately as much as one percent Cu, 0.5-0.7 percent Pb, and up to 1.5 percent Zn, can yield, while maintaining the operating expenditures on nearly the same level, a greater recovery of valuable components than the retreatment of the cinders of the sulfuric-acid shops of the Central Region. Several additional metallurgical operations, which will increase slightly the capital and operating expenditures of the entire process, will be introduced to complement the technological scheme of the Novo-Tula shop in order to perfect the recovery of lead and zinc from the cinders of the Altay pyrite concentrates.

The rare and dispersed elements recoverable during the beneficiation of ores, especially Nikolayevskoye ores, into pyrite concentrates, can be recovered from the fine dusts arising during the roasting of pyrite concentrates, and selenium and tellurium -- mainly from the slimes of sulfuric-acid shops.

In addition to the chloride method other, more perfect methods also exist. The Gintsvetmet /State Institute for Design and Planning of Nonferrous Metallurgy/ is currently working on a chloride fuming method. In that method, pyrite cinders are, upon the admixture of 1.5-2.0 percent bentonite and three to six percent calcium chloride, pelletized in pelletizing bowls. After drying at 260°C temperature the pellets are roasted in a shaft furnace at 1,250°C temperature. The shaft furnace operates on gases burned in extension burners. During the roasting 74-75 percent Cu is recovered into chloride vapors (during the retreatment of cinders with a low content of nonferrous metals), and so are 75-81 percent Zn, 72-81 percent Au, and 68-70 percent Ag.

The principal advantage of this method is the drastic reduction of the volume of equipment and of the leaching and filtering operations, which will in turn reduce substantially capital and operating expenditures. In this connection, instead of the leaching of the entire volume of chlorinated cinders, only the vapors (fumes) are subjected to hydrometallurgical processing, and they account for only five or six percent by weight of the cinders. Moreover, in the process of their shaft-furnace roasting, the pellets retain their original shape and become very strong, which is very important to their subsequent processing in blast furnaces without their sintering. After the leaching of the chloride vapors the recovery of metals from solutions presents no special difficulty and is conducted in the same way as in the chloride method.

The industrial introduction of this method of retreating pyrite cinders basically hinges on the devising of an improved design of the hermetically tight shaft furnace operating on gaseous fuel. One such design, of an enlarged-laboratory type, has been devised and successfully tested at the Gintsvetmet.

At present it is necessary to orient toward the chloride method of retreating pyrite cinders as the most realistic method for industrial introduction.

Thus, it should be noted that the comprehensive utilization of pyrite concentrates is tied to the activities of various sovnarkhozes and various branches of the national economy: chemical industry and ferrous and nonferrous metallurgy. Therefore, the practical meterialization of the herein-suggested path of utilizing the Altay pyrites requires a coordination of the efforts of various branch-of-industry, research and design institutions (Giprokhim, Gipromekh, Gintsvetmet, Giprotsvetmet, and others), and the planning organs of the concerned republics and sovnarkhozes as well.

## b. The Distribution of Dispersed Elements in Industrially Exploitable Deposits of Nonferrous Metals

/Following is the translation of an article by V. N. Leksin and L. V. Smirnyagin entitled "O Differentsirovanii Zapasov Rasseyannykh Elomentov v Promyshlennykh Mestorozhdeniyakh Tsvetnykh Metallov" (English version above) in Gornyy Zhurnal (Mining Journal), No. 3, Moscow, March 1960, pages 11-14./

The modern methods of estimating the reserves of dispersed elements (indium, gallium, thallium, selenium, tellurium, germanium, rhenium) in the deposits of nonferrous metals are based on considering their bulk content, which is determined according to a very great number of averaged samples of ore mass.

However, such an estimation of the reserves of dispersed elements does not reflect their distribution among the various /mineral/ components of ore i.e., it is not known what part of these elements gathers in the ore concentrates and what part, in ore tailings. In addition, the low accuracy of the assays of dispersed elements in bulk ore samples does not assure the necessary reliability of the estimate.

It is necessary to change radically the existing methods of assaying and calculating the bulk reserves. These methods should be based on the determination of the content and reserves of each dispersed element according to the individual mineral components of ore. The ultimate magnitude of the gross reserves of dispersed elements should in this case be determined as the sum total of the reserves calculated for each individual mineral.

The necessity of differentiating the reserves is caused primarily by the fact that, unlike the nonferrous metals, dispersed elements are geochemically associated (in the form of isomorphous or micromineral "impurities") with a large number of minerals -- sulfides, alumosilicates, etc. (Table 1).

The scattered distribution of dispersed elements inevitably results in that, first, a part of the dispersed metals associated with alumosilicates, oxides and pyrites (if the last-named are not isolated into an independent concentrate) becomes, under the existing beneficiation practice, immediately lost in ore tailings (Table 2).

## Table 1

Content of Metals in the Principal Industrial Minerals of Polymetallic Deposits

| | Principal Industrial Metals | | | | Dispersed Elements Accompanying the Metals | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Copper | Lead | Zinc | Iron | Selen-ium | Tellur-ium | Indium | Thall-ium | German-ium | Rhen-ium |
| Chalcopyrite | + | – | – | – | + | + | + | + | + | + |
| Sphalerite | – | – | + | – | + | + | + | + | + | + |
| Galenite | – | + | – | – | + | + | + | + | + | + |
| Pyrite | – | – | – | + | + | + | + | + | + | + |
| Alumosilicates | – | – | – | – | – | – | – | + | – | + |

## Table 2

**ТАБЛИЦА 2. Извлечение рассеянных металлов в селективные концентраты**

4

(в процентах от исходной руды)

| 2 [illegible] | 3 | 4 | 5 | 6 | 7 | 8 |
|---|---|---|---|---|---|---|
| 9 *Медно-цинковые руды* | | | | | | |
| 10 Концентраты: | | | | | | |
| медный | 23.8 | 12 4 | 25.6 | 12.9 | | 18,3 |
| цинковый | 1.4 | 1.0 | 7.4 | 2.1 | — | 6.9 |
| пиритный | 51.8 | 65.9 | 43.9 | 66.3 | — | 53.6 |
| Отвальные хвосты | 23,0 | 20.7 | 23.1 | 18.T | — | 21.2 |
| 15 *Полиметаллические руды* | | | | | | |
| 16 Фабрика № 1 | | | | | | |
| 17 Коллективный концентрат | 30 | 20 | 10 | 60 | | 10 |
| 18 Отвальные хвосты | 70 | 80 | 90 | 40 | 95 | 90 |
| 19 Фабрика № 2 | | | | | | |
| Концентраты: | | | | | | |
| свинцовый | 50 | 15 | — | 5 | | |
| медный | 10 | 10 | — | 2 | 7 | |
| цинковый | 10 | 10 | | 2 | 2 | — |
| Отвальные хвосты | 80 | 65 | — | 91 | 91 | — |
| Фабрика № 3 | | | | | | |
| Концентраты: | | | | | | |
| свинцовый | | — | 1 | 6 | 2 | 4 |
| цинковый | — | — | 2 | 2 | 3 | 4 |
| Отвальные хвосты | — | | 97 | 92 | 95 | 95 |
| *Общее извлечение в концентраты* | | | | | | |
| Медно-цинковых руд | 77 | 79 | 77 | 84 | — | 79 |
| Полиметаллических руд | 70 | 35 | 3 | 8 9 | 5 9 | 5 |

/Legend for Table 2 on next page/

Legend for Table 2, on page 14

1. Recovery of Dispersed Metals into Selective Concentrates (in percent of starting ore)
2. Ores and the products of Beneficiation
3. Selenium
4. Tellurium
5. Indium
6. Gallium/Thallium?/
7. Thallium /Gallium/
8. Germanium
9. Copper-Zinc Ores
10. Concentrates of:
11. Copper
12. Zinc
13. Pyrite
14. Dump Tailings
15. Polymetal ores
16. Concentrator Plant No.1
17. Bulk Concentrate
18. Dump Tailings
19. Concentrator Plant No. 2
20. Concentrates of:
21. Lead
22. Copper
23. Zinc
24. Dump tailings
25. Concentrator Plant No. 3.
26. Concentrates of:
27. Lead
28. Zinc
29. Dump Tailings
30. Total Recovery into Concentrates of:
31. Copper-Zinc Ores
32. Polymetal Ores

The low recovery /of dispersed metals/ into copper lead and zinc concentrates is attributable to the nature of the bonds between the dispersed metal and the floated or dumped minerals. For instance, in the Altay polymetal ores, gallium is associated nearly exclusively (as much as 90 percent) with alumosilicates, and selenium and tellurium -- with pyrites.

Second, in the course of the retreatment of selective concentrates, dispersed metals become differently distributed among the finished and semifinished products of principal production (copper, lead, zinc, etc.).

The effefct of the association /existing between dispersed elements and minerals/ manifests itself with particular clarity in the initial pyrometallurgical stages of technological processes. For instance, the retreatment of copper and lead concentrates into dusts and slags from which dispersed metals can be directly recovered (in particular, from dusts by the sulfating method, and from slags by fuming), involves the passing of the following amounts of dispersed metals, in percent:

| | Indium | Germanium | Thallium /Gallium?/ |
|---|---|---|---|
| Dusts of copper-smelting plants | 15.8 | 48.4 | 45 |
| Dusts of lead plants | 15.0 | 8 | 70 |
| Slags of copper-smelting plants | 81.2 | 19.5 | 39 |
| Slags of lead plants | 60 | 60 | 7 |

Thirdly, the recovery of dispersed metals from semifinished products requires various expenditures of labor, energy and reagents. The magnitude of these expenditures depends on the content of this or that dispersed metal in the semifinished product, the total amount of that product and its material composition, and the presence of other valuable components therein. Thus, the recovery of selenium and tellurium from the semifinished products of lead and zinc production that have a low content of these elements, will be more complicated and expensive than the recovery of selenium and tellurium from the concentrated semifinished products of copper production (e. g., from the slimes of copper electrolysis). The recovery of selenium from the products of copper plants may, under present conditions, reach 70 percent, and from the products of lead plants -- 25 percent, and from the products of zinc plants -- 10 percent. The possible recovery of tellurium correspondingly reaches 40, 10 and 8 percent, respectively.

Such differences in the indexes of recovery and loss according to the type of the starting concentrates are also characteristic of the other dispersed metal. The recovery of indium from lead concentrates amounts to approximately 35 percent, and from zinc concentrates --

20-30 percent, and the recovery of thallium from lead concentrates amount to 45 percent, and from zinc concentrates -- only about 20 percent.

Thus, the calculation of the reserves of dispersed metals according to their gross bulk content does indeed represent the sum total of the individual reserves, which are to a differing degree recoverable and which require differing expenditures on recovery into marketable metals. Therefore, it appears expedient to differentiate the reserves of dispersed metals according to the individual mineral components of ore serving as the basis for concentrates and tailings. In lead-zinc-copper deposits it is necessary to differentiate the reserves of selenium and tellurium among the chalcopyrites, galenites, sphalerites and pyrites; reserves of indium -- among sphalerites, galenites and chalcopyrites; thallium -- among pyrites, galenites, sphalerites, chalcopyrites, and alumosilicates; gallium -- between sphalerites and alumosilicates; cadmium -- among sphalerites, galenites, chalcopyrites, and gray copper ores. In deposits with a less complex mineralogical composition the differentiation of reserves is simpler. In particular, in the copper-pyrite deposits, reserves of selenium and tellurium can be differentiated between chalcopyrites, molybdenites, and pyrites.

The principal analytic method in the study of the content of dispersed elements should be the analysis of the fractions close to monomineral ones. These should characterize the industrial metallic minerals and the dumpable nonmetallic minerals. Experience shows that the differentiated determination of contents and reserves requires a much smaller number of analyses /assays/ (spectral and chemical) than bulk sampling, and then the accuracy of calculations is moreover higher. This may be explained by the limited sensitivity of spectral analyses, which are usually used to determine the bulk content of dispersed elements in averaged ore samples.

When the content of a dispersed metal amounts to thousandth and ten-thousandth parts of a percent, the spectral analysis makes it possible to determine that element within an accuracy of not more than $\pm$ percent, and sometimes that analysis does not work at all. On the other hand, when monomineral samples, in which the content of dispersed elements is hundreds of times greater, are subjected to analysis, the methods of analysis become simplified and the obtained results are much more accurate. As for the recalculation of reserves in terms of the entire ore mass according to individual concentrates, etc., this does not present any difficulties.

It should be noted that the gross content of dispersed metals can be calculated much more reliably according to the sum total of the analyses of mono mineral samples, because of the greater precision of such analyses. In this case, errors would hinge directly on the extent of the part of dispersed metals that is associated with the nonindustrial monomineral fractions, which are not subjected to assaying. As for the error during the determination of the <u>industrial</u> reserves of dispersed metals by the differentiation method, it will stem solely from the error of these analyses and thus it will be one or two orders of magnitude smaller than in calculations based on the analyses of bulk samples.

<u>Example:</u> A deposit of copper-zinc pyrite ores has nominal reserves of 20 million tons. The ore designed for industrial beneficiation contains 1-2 percent copper, 1-3 percent zinc, 12 percent sulfur, 50 percent silica, 15 percent alumina, 3 percent magnesium oxide, and 1 percent calcium oxide. Spectral analyses show the gross content of selenium in the ore to be 0.001 percent, and tellurium -- 0.001 percent (10 grams/ton). Eighty percent of the ore's copper is represented by primary copper minerals, 14 percent -- secondary, 5 percent -- oxidized, and 1 percent -- copper sulfate. Ninety-seven percent of the ore's zinc is represented by sphalerite. The total content of sulfides in the ore is 25-30 percent. Copper, zinc and iron are represented by pyrite, chalcopyrite, sphalerite and, in smaller amounts, covellite and gray copper ore. The nonmetallic part is represented by quartz, sericite, chlorite, and calcite. The impregnation size of the metallic minerals is, in mm: pyrite -- 0.001-2; chalcopyrite -- 0.001-1; sphalerite -- 0.001-0.05.

Analyses of monomineral samples indicate that the industrially valuable ore components contain the following amounts of selenium and tellurium, in grams per ton: pyrites -- 50 grams of selenium and 65 grams of tellurium; chalcopyrites -- 200 grams of selenium and 100 grams of tellurium; and sphalerites -- 40 grams of selenium and 20 grams of tellurium.

<u>Planned Indexes of Beneficiation</u>. An ore nominally containing an average of 1.25 percent copper and 1.5 percent zinc can be processed to yield: copper concentrate (14 percent copper and 50 percent zinc), and pyrite concentrate, containing 45 percent sulfur. The recovery should amount to: copper into copper concentrate -- 90 percent; zinc into zinc concentrate -- 90 percent; and

and sulfur into pyrite concentrate -- 50 percent.

The subsequent retreatment of selective concentrates involves the recovery of selenium and tellurium from dusts, gases, and the electrolytic slimes of the copper plants and the recovery of selenium from the slimes of the sulfuric acid shops of zinc plants. The question of recovering selenium and tellurium during the retreatment of pyrite concentrates has not been solved considering that for the time being these concentrates are stored at the concentrator plant instead of being retreated.

On the basis of the data of bulk analysis it is possible to compute only the sum total of the reserves of selenium and tellurium for the deposit as a whole (200 tons each). The determination of the actual amounts of these metals that proceed to metallurgical production and can be recovered into marketable output is in practice impossible.

With a differentiated calculation, on the other hand, on knowing, e. g., that copper, zinc and sulfur (in pyrite concentrate) in the selective concentrates are represented by chalcopyrite, sphalerite and pyrite, it is possible to determine directly the amount of dispersed metals which will pass into these concentrates. In the general case, the calculation is conducted as follows:

One ton of copper concentrate contains 140 kg of copper and 60 kg of zinc, represented by 400 kg of chalcopyrite and 89 kg of sphalerite, respectively. These minerals, according to the data of the assays of monomineral samples (see above) contain 83 grams of selenium and 41.5 grams of tellurium.

One ton of zinc concentrate contains 500 kg of zinc and 15 kg of copper, represented by 746 kg of sphalerite and 43 kg of chalcopyrite, respectively. These minerals contain 40 grams of selenium and 19.5 grams of tellurium.

One ton of pyrite concentrate contains 450 kg of sulfur represented by 980 kg of pyrite (the amount of sulfur contained in the chalcopyrite and sphalerite present in the pyrite concentrate). This amount of pyrite contains 48.8 grams of selenium and 63.4 grams of tellurium.

In the course of the exploitation of the deposit the copper concentrates which will be industrially retreated will contain a total of 110 tons of selenium and 45 tons of tellurium, the zinc concentrates -- 16 and 8 tons, respectively, and the pyrite concentrates -- 180 and 235 tons, respectively.

Thus, there exists a tangible possibility of an exceptionally simple and sufficiently accurate estimation of the amount of dispersed elements which will proceed to industrial retreatment (Table 3). On knowing the output capacity of the concentrator plant it is possible to compute with a sufficient accuracy the flow of these metals to various branches of nonferrous metallurgy during any time period. Neither is there any difficulty in computing the ultimate amounts of marketable dispersed metals which can be recovered from the ores of a given deposit.

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## Table 3

1 ТАБЛИЦА 3. Расчет промышленных запасов селена и теллура для рассмотренного примера

| 2 Промышленные минералы-носители и их запасы | 3 Селен | | 6 Теллур | |
|---|---|---|---|---|
| | 4 содержание, г/т | 5 запасы, т | 7 содержание, г/т | 8 запасы, т |
| 9 Пирит, 3800 тыс. т | 50 | 190 | 65 | 250 |
| 10 Халькопирит, 580 тыс. т | 200 | 116 | 100 | 58 |
| 11 Сфалерит, 420 тыс. т | 40 | 16,8 | 20 | 8,4 |
| 12 Итого промышленных запасов | 322,8 | | 316,4 | |
| 13 В том числе металлов, извлекаемых в товарные продукты | 205 | | 19 | |
| 14 Минимальный коэффициент использования запасов, % | 60 | | 6 | |

1. Computing the Industrial Reserves of Selenium and Tellurium for the Example Considered Above
2. Industrial minerals -- carriers of selenium and tellurium -- and their reserves
3. Selenium
4. Content -- grams/ton;
5. Reserves, tons
6. Tellurium
7. Content -- grams/ton
8. Reserves, tons
9. Pyrite, 3,800,000 tons
10. Chalcopyrite, 580,000 tons
11. Sphalerite, 420,000 tons
12. Total, Industrial Reserves
13. In which /Dispersed/ Metals Recoverable into Marketable Products
14. Minimal Index of utilization of reserves, in percent.

Thus, in our example, in which selenium is recovered from copper concentrates into marketable output to the extent of 70 percent, from zinc concentrates -- to the extent of 10 percent, and from pyrite concentrates -- to the extent of 70 percent, correspondingly, 77, 1.6 and 126 tons, respectively, of selenium (altogether 205 tons) will be recovered during the period of exploitation of the deposit. As for tellurium, its recovery from copper concentrate reaches 40 percent, and from zinc concentrate -- 8 percent (the recovery of tellurium from pyrite concentrate remains problematic), and so, correspondingly, 18 tons and 640 kg (altogether about 19 tons) of tellurium will be recovered in the course of exploitation of the deposit.

Consequently, it is possible to compute rapidly and with a sufficient accuracy the possible volume of production of dispersed metals in the individual enterprises processing the ores of the evaluated deposit, and even the tentative sum total of expenditures on such production.

The gross content of selenium and tellurium in the ore mass is as follows, according to the differentiated calculation: selenium -- 322.5 tons: 20,000,000 tons = 16 grams/ton; and tellurium -- 316,4:20,000,000 = 16 grams/ton. These are much more accurate and reliable figures than the 10 grams/ton determined from bulk assays.

The introduction of differentiated assaying will increase the accuracy of computing the industrial reserves of dispersed metals, and it will simplify the determination of the part of these reserves which can be utilized at present. The differentiation of the reserves of dispersed metals according to mineral components of ore will make it possible to justify economically the value of the tailings of concentrator plants and perhaps also of the extra-balance-sheet /substandard/ reserves of certain industrial deposits of nonferrous metals.

Lastly, the introduction of the differentiated appraisal will make it possible to simplify the estimation of the reserves of dispersed metal and cut its costs, to trace more operatively the changes in and prospects for the utilization of each such reserve, and to concretize (also in a differentiated manner) the industrial standards for multi-constituent ores.

The differentiation of the reserves of a most valuable raw material -- dispersed metals -- is a necessary and feasible task.

END