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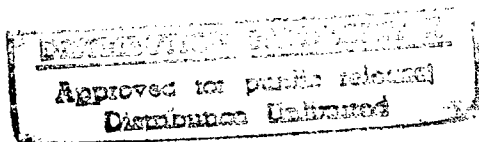
SOVIET NONFERROUS METALLURGY

No. 8

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SELECTED TRANSLATIONS

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Introduction

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

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1. A Reply to the Critics of my Article "Certain
Peculiarities of Modern Converting Practices
and Technological Efficiency at Copper-Smelt-
ing Plants of the Urals During the Seven-Year
Plan"*

/This is a translation of an article written
by L. M. Gazaryan in Tsvetnyye Metally (Non-
ferrous Metals), No. 12, December 1959, pages
28-34; CSO:3001-N/7 (4). /

Of the problems touched upon in my article, unfortunately only one, essentially, has become the subject of discussion -- the problem of the technological scheme at the SUMZ [Sredneural'sk Copper Smelting Plant] -- although the article had also considered other no less actual problems whose disregard can only result in erroneous conclusions on the SUMZ. Let us dwell on the most substantial errors of my opponents.

I. My opponents make incorrect assumptions as to the content of valuable components in the Ural copper ores and, as a result, offer an incorrect and inefficient flow diagram for their retreatment. They postulate "that the value of the valuable components of the Ural copper ores can be presented in the following form:

Value of sulfur	-- 53%
" " copper	-- 26%
" " zinc	-- 17%
" " precious metals	-- 4%."

In citing these data the authors obviously intended to employ the extent of valuable components in ore as a proof of the incorrectness of my conclusions with regard to the SUMZ, and as a demonstration of the correctness of the technological scheme they offered. The data cited by the opponents are incorrect and they substantially distort and reduce the value of the metals (Cu, Zn, Au, Ag) of the Ural copper ores, while exaggerating, without any justification, the imaginary value of the sulfur contained in these ores. This also contributes to the drawback of the technological scheme they propose: it was based by the opponents on the

*cf. "Tsvetnyye Metally," Nos. 4, 8 and 9, 1958.

calculation of that sulfur in terms of the prices of elementary sulfur. As is known, the sulfur of the Ural copper ores will, except for the Blyavinskiye ores, continue as before -- for the next 15-20 years -- to be used in the forms of pyrite concentrate, and consequently the value of the sulfur in the ore should be assumed equal to the value of the sulfur of pyrite concentrates or sulfur pyrite. Moreover, in order to avoid huge errors in such calculations, it is necessary to take the actual average prime branch ore-extraction costs, whereas in such cases the sales price, for this or that reason, diverges greatly from the mean prime cost. Considering the above, the table cited by the opponents would have the following appearance (in percent):

Value of copper	-- 64.57
" " zinc	-- 11.20
" " precious metals	-- 8.71
" " sulfur	-- 15.52

These figures demonstrate that the principal valuable component of the Ural cupriferous ores is not sulfur but nonferrous and precious metals, contrary to the assertions of the opponents. The combined value of these metals in the ore amounts to 85 percent. If, moreover, it is kept in mind that the amount of pyrite concentrates obtained as a side product of the retreatment of copper ores will exceed the demand one and one-half to two times for the next 10-15 years, then the actual value of the sulfur of the Ural copper ores will decline to only seven or eight percent. The drastic exaggeration of the value of pyrite in the Ural copper ores and underestimation of their nonferrous and precious metals has acted as a definitely negative influence in the last seven to 10 years. In the active concentrator plants (at an insignificant fluctuation in the gold content of the ore) the recovery of precious metals into the copper concentrate had decreased because of the decline in the attention devoted to increasing such recovery (Table 1).

Table 1

Recovery of Gold During the Concentration
of Copper Ores in Ural Plants by Year

Plant	1950	1955	1957
Krasnoural'sk	63.58	53.38	53.19
Kirovgrad	63.80	66.10	45.60
Karabash	62.40	53.18	53.90
Sredneural'sk	32.97	27.30	23.40

The underestimation of the value of zinc present in the cupriferous raw material has hampered the change-over of the Ural plants to selective flotation, and improvements in that flotation. The research work which should promote a widespread introduction of the selective flotation of the copper ores of the Urals began to lag considerably. As a result of that lag, the Seven-Year Plan had to provide for the retreatment of 44 percent of all copper-zinc ores containing 1.1 percent Zn by the method of direct flotation, i. e., by a method involving the irrecoverable loss of dozens of thousands of tons of zinc during the period of the Plan. We are puzzled as to the motives of the opponents in raising this question, as it bears no direct relation to the technological scheme at the SUMZ. This scheme is based not on the composition of valuable components of the Ural ores but on the composition of the copper concentrates which are smelted at that plant. The value of the valuable components in these concentrates is shown according to the breakdown in Table 2.

Table 2

Value of Valuable Components in Charge

Metal	Content in %	Recovery in %	[In %] of Recovered Metal	Price in tons/% in rub- les	Value	
					in rubles	in %
Copper	13-15	94.0	14.1	66.0	924.0	69.32
Zinc	5-8	75	6.0	33.0	198.0	14.86
Gold	-- } Silver	{ 98	--	-- }	165.0	12.38
Sulfur	44	80	35.2	1.3	45.7	3.42
					1333	100.0

Consequently, my article is not concerned with the problem of the recovery of sulfur from Ural ores, the value of whose sulfur component amounts to 53 percent according to the incorrect calculations of the opponents, but with the problem of the recovery of sulfur from copper concentrates, the value of whose sulfur component amounts to only 3.42 percent.

My opponents write that I, in my article, "deny completely the expediency of the scheme of the smelting of copper concentrates with preliminary roasting and with the utilization of the sulfur-laden gases for the production of sulfuric acid and, in particular, for the SUMZ." No such statement was contained in my article. I deny the expediency of constructing a roasting shop at the SUMZ; in this connection, the amount of sulfuric acid which could be obtained from the sulfurous gases of that plant will decrease by one-third. In the scheme I recommend, the losses of copper decrease by two to three percent, and of gold, by five to seven percent.

Calculations show that the increase in the recovery of copper and precious metals caused by the smelting of raw unroasted charge fully compensates for the inevitable decrease in the amount of the obtained sulfuric acid.

II. My opponents are mistaken in their statement that the transition to the smelting of raw charge in the United States was favored by specific conditions absent in the Urals. The motives they cite either bear no relation to the broached problem or do not correspond with reality. It is untrue that the cheapness of natural gas in the United States favored the change-over to the smelting of raw, unroasted charge. Table 3 cites a list of certain large copper-smelting plants in the United States and of the types of fuel they burn.

Table 3

Fuel Burned

Plants Smelting Calcined Charge	Type of Fuel Burned	Plants Smelting Raw Charge	Type of Fuel Burned
Garfield Anaconda	Natural Gas	Carter McGil	Petroleum Pulverulent Coal
El Paso Calumena and Arizona	" "	Morensi	Gas
	" "	White Pine	Pulverulent Coal

From the above table it is clear that, contrary to the asseverations of the opponents, the majority of the plants smelting raw charge precisely does not use natural

gas while, conversely, the majority of the plants smelting roasted charge uses natural gas. I do not intend to draw any general conclusions from this table, because to me -- as well as to the majority of readers -- it is obvious that the value of gas in capitalist conditions is levelled by the market conjuncture for fuel, regardless of the cheapness or expensiveness of its extraction. Under the conditions of a socialist economy the planned conversion of the Ural industry to cheap gas fuel adds further weight to our recommendations for introducing the smelting of cupriferous raw material without its prior roasting. Incidentally, let us note that not only in the United States but also throughout Africa the copper-smelting plants have changed over to the smelting of raw charge, and, since 1945, 50-60 percent of all the copper smelted in Canada has originated from raw charge, because that was the year in which the biggest copper producer in Canada -- the Copper Cliff Plant -- changed over to the smelting of raw charge.

There is no doubt that the considerable nonutilized production potential of the older copper-smelting plants was of help in changing over these plants to the smelting of raw charge, because in the presence of such potential the change-over to the more progressive scheme does not require additional expenditures. This circumstance is not only an accelerating but also a predetermining factor. How else could it be possible to account for the indisputable fact that in recent years all the new copper plants under construction in the United States employ the smelting of raw, unroasted charge. As in the United States so in Canada a number of the older copper-smelting plants has not as yet changed over to the smelting of raw charge. This is basically to be explained by the circumstance that the conversion of these plants to a new technological scheme requires capital investments for renovation. To conduct such renovation, the capitalist has to be convinced that the additional expenditures thus involved will yield additional profits on at least the same level as the one that would be gained if these funds had been expended on other purposes. My opponents not only provide an erroneous interpretation of the role of the reserve capacities of the copper-smelting plants in the United States but also cite a proof which bears no direct relation to the problem we broached, inasmuch as we were not considering the question of converting the SUMZ from smelting roasted charge to raw charge. As is known, for 20-25 years now the SUMZ has been smelting raw charge.

The Americans smelt rich, pure copper concentrates containing virtually no zinc. Hence the opponents conclude

that, inasmuch as the Urals lack concentrates that are rich and contain no zinc, therefore the smelting of the Ural concentrates without roasting is not possible. This argument could have been valid some 25-30 years ago when no reverberatory smelting was practiced in the Urals, but not so at present when nearly 60-65 percent of all Ural copper is obtained from reverberatory furnaces smelting raw Ural concentrates. As for the indexes of these furnaces (at Kirovgrad and Karabash), they are not inferior to the indexes of the best foreign plants. The foregoing should not be used to conclude that there is no need for the Ural plants to endeavor to obtain purer concentrates during ore dressing. The concentrator plants of the United States, which retreat ores analogous to the copper-zinc ores of the Urals, obtain purer concentrates. This was accomplished in the United States through the widespread introduction of selective flotation for such ores. Researches of the Uralmekhanobr /Ural Scientific Research Institute for Mechanical Concentration of Minerals / have fully demonstrated the feasibility of converting all Ural concentrator plants to selective flotation. The introduction of selective flotation based on an Uralmekhanobr scheme in the Krasnoural'sk, Kirovgrad and Karabash plants has not only fully justified these researches but also ensured improved indexes. It is only the workers of the SUMZ, adhering to an outmoded practice, who do everything possible to obstruct /the introduction of / selective flotation /at their plant /. There exists a decision of the State decreeing the conversion of the Sredneuralsk Copper-Smelting Plant to selective flotation, and there exists a technological scheme drafted by the Uralmekhanobr, but my opponents, represented by comrades Gorskiy, Leonov, and certain others, resist this. The position of my opponents is a strange one: first they apply every effort to impede the obtainment of pure concentrates, and then they point to the presence of impure concentrates as the principal proof against the smelting of raw charge.

III. My opponents, misinterpreting the results of the experimental smeltings of roasted charge, attempt to deny that the deep roasting envisaged in the project of the SUMZ is not tied to the plant's developmental projects (application of slag fuming) and that that project duplicates the errors made by the /Canadian / Flin-Flon Plant. To substruct their conclusions the opponents refer to the results of the performance of an experimental reverberatory furnace at the SUMZ, published in "Tsvetnyye Metally," No. 2, 1957.

That article, on page 23, cites data on the composition of the calcine obtained by fluidized-bed roasting and used for smelting in the experimental reverberatory furnace, and it specified (in Table 2) the following forms of zinc compounds present in the calcine (in percent):

Component	No. of Sample		
	1	2	3
Zinc, Total	6.87	14.40	16.37
in which:			
Sulfatic Zinc	1.44	1.00	1.67
Zinc Oxide	0.17	0.31	0.33
Silica Zinc	0.25	0.25	0.51
Sulfidic Zinc	2.70	{12.9	9.90
Ferritic Zinc	2.42		4.28

It can be seen that from 56.6 to 63.6 percent of the oxidized zinc in the calcine occurs in the form of zinc ferrite -- and yet my opponents, referring to that article, deny the presence of zinc ferrite in the calcine smelted in the experimental furnace. Thus, in both the calcine of Flin-Flon Plant and the roasted charge smelted in the experimental furnace of the SUMZ, a major part of the oxidized zinc is present in the form of zinc ferrite.

Page 28 of the same article states that the bed plate of the experimental furnace was observed to clog up periodically with unfused residues and, although the authors do not write how many times in the course of experiments the furnace had to be blown-out for this reason, on page 25 in Table 5 it can be seen that, because of the clogging of its bed plate, with unfused residue, the furnace was blown-out not less than twice in the course of three months of operation.

At the Flin-Flon Plant it is also necessary to blow out the furnaces because of the clogging of the bed plate.

Page 24 of the above-mentioned article, (in Table 3) cites the following data:

Operating Period	Content of Cu, in %		
	In Calcine	In Matte	In Slag
I	9.2	56.0	1.06
II	8.59	53.5	0.98
III	7.00	46.5	0.81
IV	8.15	44.5	0.88

i. e., to prevent the clogging of the bed plate, in the course of the experiments, the copper content of matte was systematically reduced and deep roasting was gradually abandoned. In other words, the measures taken to control the clogging of the bed plate were the same as those taken at the Flin-Flon Plant.

Table 5, page 29 of that article, indicates the composition of the "bears" /unfused, chilled residues/ in the experimental furnace. Inasmuch as the most complete data pertain to the third operating period, we will dwell on this period.

The analysis of the bed-plate "bears" of the experimental furnace is as follows, in percent:

Operating Period	Sample	Content in %			Ratio	
		Cu	Zn	Fe	ZuCu*	FeCu*
III	Composition of Matte	46.5	5.9	27.0*	0.127	0.581
	Composition of Bed Plate:					
	Upper Layer	6.2	5.7	30.2	0.919	4.871
	Middle Layer	7.1	4.0	18.1	0.564	2.950
	Lower Layer	38.4	2.7	26.0	0.07	0.677

* Calculated by the present writer

The article's authors indicate that the furnace "bears" had basically consisted of matte. There is no doubt that the "bears" of any reverberatory furnace consist of matte. It is intriguing that the composition of the bed-plate matte differs from the composition of the bath matte by containing more iron and zinc. Only one plausible explanation can exist for the enrichment of the furnace bed plate with iron and zinc. In the experimental furnace, as

in the customary practice of reverberatory smelting, this had occurred because of the gradual accumulation of ferrites on the bed plate (in the given case, mainly on account of zinc ferrite). Now this is precisely where we witness a repetition of the initial difficulties which had occurred at the Flin-Flon Plant during the deep roasting of high-zinc charge. There exists a commonly accepted method of combatting this phenomenon -- alteration of the technological regime of the reverberatory furnace and of converter reduction. Thus, the operations of the experimental reverberatory furnace referred to by my opponents have fully confirmed our misgivings.

IV. My opponents attempt, by a direct comparison of the value of metallurgical reduction calculated on the basis of the complex whole of the activities of enterprises, to question the relevancy of my conclusions regarding the costs of shop-scale reduction at the Krasnoural'sk and Sredneural'sk copper-smelting plants. The cost figures cited by my opponents never could, nor can ever serve, as a criterion for determining the economicality of the technological schemes of the metallurgical shops of these enterprises. There is a gap of 12-15 years between the respective dates of construction and activation of both enterprises, and their construction had occurred under different conditions, with different unit capital investments, different equipping of the auxiliary facilities, and different composition and volume of gross output. As a result, the plant-wide (more accurately, combine-wide) and intra-production expenditures composing the cost of metallurgical reduction are not comparable for the two enterprises. The cold, unanalyzed statistical figures used by the opponents can only confuse the reader. A comparison of the costs of shop-wide reduction according to individual expenditure items, such as is cited in my article, is a difficult and laborious project, the more so as, since 1952, the enterprises no longer conduct bookkeeping according to the different stages of metallurgical reduction. Therefore, the related calculations will always involve a number of conventions. To avoid any major errors and to minimize the conventions, I took the expenditures pertaining only to shop-wide reduction and, under these conditions, this yielded unexpected conclusions showing the inexpensive process of the smelting of raw charge to be on the same level of costs as the more expensive process of the smelting of roasted charge.

Analogous calculations, made by H. G. Stevens ("Tsvetnyye Metally," No. 8, 1959, pages 86-87) for the

copper-smelting plants of the United States show that the expenditures on the smelting of raw charge per ton of copper are always lower than the expenditures on the smelting of roasted charge and, according to the content of copper in the concentrate, range from 11 to 18 percent. In my calculations I had provided a picture of shop-scale reduction in these plants, but this does not signify that, upon an efficient conduct of the technological process, the cost of the smelting of raw charge in the Ural plants will likewise not be lower than the cost of the smelting of roasted charge.

V. My opponents deny the possibility of the simultaneous operation of old reverberatory furnaces at the SUMZ and Krasnoural'sk plants. The entire flue and draft system of the reverberatory shop at the SUMZ is designed for the parallel performance of two furnaces. According to a project, the gases of the reverberatory furnaces, on passing through utilizing boilers, should enter the flue system with a temperature of 250-300°C, but the plant has not yet installed the utilizing boiler, which has already arrived and now lies idle at the plant. Because of the failure to install the utilizing boiler, the gases sent to the flue system have a temperature of 1,250-1,300°C; under such conditions the flue system is incapable of servicing normally the two furnaces operating in parallel. This drawback is even further aggravated by the circumstance that the plant strives to raise furnace productivity by overburning the fuel. The consumption of nominal fuel amounts to 20-22 percent, whereas in the older Ural plants it amounts to 16-17 percent.

While in general the raising of furnace productivity by means of the overburning of fuel is impermissible, in the conditions of the SUMZ, where no utilizing boiler is in operation, it is simply intolerable. At such an operating regime of the furnace at the SUMZ not only is a half million rubles worth of fuel irrecoverably lost annually but also, in view of the incredible overloading of the furnace, there exist no conditions for an appropriate settling out and separation of slag from matte. In this connection, the losses of copper in slag have climbed a hundred percent compared with the initial period of the plant's operations. My opponents, instead of suggesting to the plant a way of rapid and proper reorganization of the performance of the reverberatory furnace and creation of the conditions for a full utilization of the capacity of the reverberatory shop, enumerate a number of objects the need for whose construction is not disputed by anyone, and thereby justify the

abnormal operating mode of that shop.

VI. My opponents, defending the unjustified demands as to the amounts of the gold-bearing quartz fluxes to be used in converting, resist the change-over of that process to the use of gold-bearing fluxes alone. The extraction and recovery of gold-bearing fluxes from the Ural deposits yields an excess amount of flux fines (0-8 mm) -- the so-called reverberatory size class; because of the restricted range of applications of these fines, they are accumulating. Certain lovers of administrative red tape from the Technical Division of the former Ministry of Nonferrous Metallurgy established the category of converter-class fluxes (8-20 mm), which restricts the use of fluxes of finer size for converting, whereas no such restriction is required by the converting technology. Although the converter shops need quartz flux, the plants refuse to accept gold-bearing flux fines on the ground that these are not in the converter-class category.

In the past (the years 1931-1935) the Ural plants used broadly as a quartz flux the crushed gold ore, provided by gold-dressing plants, of a still smaller size than the reverberatory-class fluxes. My opponents, instead of pointing out the harmfulness of the restriction of the use of fluxes to the converter-class size alone, point to this circumstance as an argument against the smelting of raw charge.

Professor A. A. Tseydler, repeating the errors of the other opponents, refutes the correctness of my calculations on the basis of, as he himself states, very tentative computations. Very tentative computations inevitably also entail major errors but Professor A. A. Tseydler has moreover committed still greater errors which exceed the confines of such computations. These errors are as follows:

1. Basing himself on the cost of shop-scale metallurgical reduction as accurately calculated by the present writer, Professor Tseydler took arbitrarily for the Krasnoural'sk Plant the data of the technical report of 1957 (instead of the mean annual report for 1956 used in my calculations), and for the SUMZ -- 1958 data (instead of the data of the mean annual 1957 report).

2. Having adopted the amount of charge per ton of blister copper as the basis for his calculations, Professor Tseydler commits another error by regarding the various schemes as mutually comparable. The merit of the smelting of raw charge consists precisely in that nearly no quartz flux is introduced into the reverberatory-furnace charge (at the SUMZ in 1957 quartz flux accounted for 4.8 percent

of the solid charge). Fluxes are mainly used in converter reduction and smelted without losses. At the Krasnoural'sk Plant, on the other hand, the technological scheme precludes the introduction of the needed flux through the converter into the cycle of metallurgical reduction, and therefore that plant is forced to introduce the flux into the reverberatory-furnace charge and to expend fuel on its smelting (at Krasnoural'sk in 1956 quartz flux accounted for 26.2 percent of the solid charge). Therefore, in Professor Tseydler's calculations the positive aspect of the smelting of raw charge became negative. In order to rectify this error in relation to both plants, only the amount of the smelted copper-bearing material should be taken for the calculations. After such rectification of the above-mentioned two errors, Professor Tseydler's data will coincide closely with mine.

In conclusion I wish to stress that the technological scheme for the SUMZ which my opponents defend was approved at a time when satisfactory schemes of selective flotation of Degtyarsk ores were unknown. Thus it was a compulsory solution.

At present, both the Gintsvetmet and the Uralmekhanoobr have developed a scheme for the selective flotation of Degtyarsk ores. That scheme provides for recovering 50 percent of the zinc from the ore into zinc concentrate, and for a 15-percent content of copper and nine-percent content of zinc in the copper concentrate. This new circumstance necessitates a radical revision of project solutions which, for some reason, is ignored by the opponents.

From the Editors

With the publication of the above material the Editors close the discussion of L. M. Gazaryan's article, published in issue No. 4 of this periodical.

Conclusions on the conducted discussion will be communicated by the Editors after the examination of certain problems broached in that article, by the GNTK /Main Scientific and Technical Committee under the Council of Ministers USSR(or RSFSR) /, is completed.

2. Industrial Experiments on Decoppering Lead Slags When Using the Fuming Process

[This is a translation of an article written by A. V. Novozhilov and I. A. Stroitelev in Tsvetnyye Metally (Nonferrous Metals), No. 12, December 1959, pages 1-6; CSO:3001-N/7 (5).]

The present article expounds the results of industrial experiments with the decoppering of slags during fuming at the Ust'-Kamenogorsk Lead Plant. The slag-fuming installation of that plant is designed for dezincing molten lead slags usually containing 0.7-1.0 percent Cu and sometimes as much as two to five and more percent Cu. At present the partial decoppering of slags after fuming is conducted through the separation of matte in an electric settler. In this connection the dump slags retain 0.6-2.0 percent Cu (the mean content of copper for 1958 amounted to 0.9 percent), 0.1-0.3 grams of Au per ton, and 6-20 grams of Ag per ton. The mattes contain from 19 to 45 percent Cu.

To gain information on the forms of loss of copper, the dump slags were subjected to a chemicominalogical investigation. It was established that a slowly cooling slag melt segregates out into magnetite, wustite and fayalite. However, the slag does not undergo a complete decrystallization, even when cooled in a 3.7m³ ladle; the residual melt cools in the form of glass. As shown by chemical analyses of the magnetite, wustite and fayalite, copper does not enter into the crystal lattices of these minerals. Oxidized types of copper in the form of independent minerals have not been discovered in the investigated slags. The metallic copper appears in the slags only when they are deficient in sulfur.

Under a microscope the dump slag displays two different forms of sulfidic inclusions. One such form, measuring from a few microns to 0.1-0.2 and more mm in size, is represented by droplets of matte segregating from the slag after an additional settling period. The other sulfidic inclusions, as distinguished from the first form, are found only in the glass of the slag, where they remain in a fairly stable amount regardless of the duration of settling; their composition differs sharply from that of matte. In the slag cooled in the ladle these sulfidic inclusions had the following composition: 11.8 percent Cu; 49.9 percent Fe; 2.7 percent Zn; and 32.1 percent S, while the matte droplets had

the following composition: 47.5 percent Cu; 17.8 percent Fe; 2.9 percent Zn; and 26.3 percent S. Samples of the slag were melted inside corundum crucibles in a nitrogen atmosphere. After the melting of the slag and its quenching in water the above-mentioned sulfidic inclusions tend to disappear, in contrast to matte droplets, but afterward they reappear in the same slag after its secondary melting (at 1,200°C) and slow cooling. Hence it follows that these inclusions form as a result of sulfides that are soluble in molten slag but segregate in the process of the crystallization of that slag.

In the slag block cooled in the ladle, on the bottom of the shrinkage cavity forming in the thermal center where the crystallization of the melt terminates, we observed a sulfidic conglomeration (up to five-eight mm in height) having the same composition as that of the sulfidic segregations in the glass of that slag. This is but a superfluous proof of the solubility in the slag of sulfides, or more accurately the copper-bearing sulfidic fluid, and of the segregability of that fluid during the subsequent stage of crystallization of the slag. During a slow cooling of the slab, this liquid detaches itself in measure with its segregation in the shrinkage cavity.

The minimal content of copper in slags after prolonged settling of the slags in a nitrogen atmosphere, when they do not contain any matte inclusions observable under a microscope, usually ranges from 0.25 to 0.30 percent. This magnitude is probably close to the amount of copper soluble in slag.

A project has been drafted for decoppering slags through the after-charging of pyrite into a slag-fuming furnace. Inasmuch as the amount of soluble copper in slag depends basically on the composition of slag, therefore the purpose of sulfidization consists in "washing" the slag with sulfidic melt so as to decopperize it further and to sediment the droplets of matte.

The sulfidization of slags by pyrite concentrate was tested by laboratory experiments (by the crucible melting method). Upon using the sulfidizer in the amount of eight to nine percent of the weight of a slag containing 0.98 percent Cu, and after a settling time of 10 minutes, it proved possible to reduce the slag's content of Cu to 0.38-0.40 percent. An increase in the consumption of the pyrite concentrate to 16 percent and the prolongation of settling time to 60 minutes resulted in reducing the Cu content of slag by an additional 0.05-0.08 percent only (in absolute value).

The laboratory data served as the basis for conduct-

ing industrial tests of such sulfidization. The aftercharge of nodular pyrite (up to 150-200 mm mesh) into the slag-fuming furnace yielded negative results mainly because of the slowness of its dissolution in the slag. The charging of pyrite into an electric settler generally proved not to be very effective. The most acceptable charging method would be the blowing of sulfidizer fines into the furnace bath -- which requires the installation of special equipment. Therefore, the sulfidizer was charged in through the charging port of the furnace. December 1958 witnessed the industrial tests of the decopperization of slags based on the charging of pyrite concentrate and pyrite fines (up to 30 mm mesh) into the furnace by a scoop mounted on an overhead crane in the last 5-10 minutes before the slag begins to be tapped (Table 1). Every single blowing operation involved the charging of one scoop of sulfidizer, which, as depending on the melting stock of the furnace, ranged from 3.0 to 5.7 percent of the weight of the charged-in slag. The charging conditions prevented any further increase in the consumption of the sulfidizer.

Table 1

Composition of Pyrite Fines and Pyrite Concentrate,
in percent

Sulfi- dizer	Cu	Pb	Zn	S	Fe	Si ₂ O	Au grams/ /ton	Ag grams/ /ton
Pyrite Fines	0.3	0.34	1.03	42.4	36.5	7.8	2.0	28.0
Pyrite Concen- trate	0.62	2.16	3.32	40.2	33.8	12.06	15.0	154.0

The amount of the charged-in slag and matte obtained upon sulfidization was determined according to volume (by ladles). The matte was accumulated in an electric settler and removed thence once daily. Daily samples of the fumes were collected from the utilizing boiler, air preheater and gas flue. Altogether, 36 operations of the fuming of the slags of the shaft furnace and converter were conducted during the period of the sulfidizing experiments. Sulfid-

izing led to a considerable decrease in the losses of copper and noble metals in the slags. Thus, the charged-in slags contained 0.66-6.96 percent Cu, whereas the dump slags contained 0.3-0.85 percent Cu. The average content of copper in the dump slags during the 36 operations amounted to 0.52 percent, and that of Ag/Au -- 6.3 grams per ton (Tables 2, 3, 4).

During the above experiments, 0.16-3.4 percent S remained in the slags prior to the charging of the sulfidizer. After sulfidizing and settling the content of S in the slags normally averaged 1.6-2.0 percent S (mean average: 1.8 percent). Consequently, up to three to five percent of sulfidizer could be expended on "saturating" a slag with sulfur in the event the slag has a low content of sulfur. Therefore, the consumption of sulfidizer should be increased (to eight or nine percent, according to laboratory data). Pyrite concentrate appears to be the most suitable sulfidizer, inasmuch as, compared with nodular pyrite, it "dissolves" with great facility in the slag and it contains more gold and silver, which are efficiently recovered into the matte.

To reduce the burning out of sulfur, the sulfidizing of slags should be conducted within the ladle itself of the fuming operation. The addition of the sulfidizer did not exert any noticeable influence on the distillation of lead and zinc, because this addition was carried out (aftercharged) at the end of the operation.

The state of the caissons of the slag-fuming furnace during the period of sulfidizing experiments was fully satisfactory.

Upon mastering the decopperization of slags it will be possible to use the slag-fuming installation for the retreatment of copper-rich slags, inclusive of the converter slags, and solid copper-lead-zinc materials as well.

Table 2

Composition of the Mattes and Fumes Obtained
Upon the Sulfidization of Slags, Average for 36
Fuming Operations, in percent

Product	Cu	Pb	Zn	Scorbon	Fe	Au grams/ ton	Ag grams/ ton
Mattes							
Fumes:							
Coarse, from Utiliz-	16.18—	0.26—	2.12—	18.0—	46.0—	7.0—8.0	300—522
ing Boiler	18.84	2.12	4.0	19.0	50.0		
Coarse, from Air	2.17—3.6	—	—	—	—	1.0—1.5	49.5—100
Preheater	0.88—	—	—	—	—	0.25—1.0	2.95—
	1.21	—	—	—	—		74.4
Fine, from Gas Flue	0.41—	—	—	—	—	0.3	31.5
	0.45	—	—	—	—		

Table 3

Distribution of Copper and Noble Metals During
Sulfidization According to Product of Fuming,
for 36 Operations

Product of Fuming	Cu, in %	Au/Ag grams/ton
Dump Slag	30.9	18.0
Matte	60.8	67.4
Fumes	8.3	14.6

Table 4

Composition of Average Samples of Charged-In
and Dump Slag for 36 Fuming Operations, in
percent

Samples of	Cu	Pb	Zn	S	Fe	SiO ₂	CaO
Charged-In slag	1.48	2.5	12.8	3.4	32.9	18.3	6.7
Dump Slag	0.52	0.14	1.94	1.8	36.9	26.3	8.2

3. The Distribution of Rare Elements in the Products of Concentration of Copper and Copper-Zinc Ores

√This is a translation of an article written by A. I. Okunev, V. A. Oshman, and A. V. Petrov in Tsvetnyye Metally (Nonferrous Metals), No. 12, December 1959, pages 49-54; CSO:3001-N/7 (6). 7

In the light of the task of expanding the comprehensiveness of utilization of raw material, the behavior of rare elements in the ore-dressing processes is of a major practical and theoretical interest; this behavior depends both on the method of flotation employed and on whether this or that element present in the raw ore forms individual, independent minerals or joins the crystal lattice of some principal minerals of the ore.

Studies concerned with the problems of the distribution of rare elements during the concentration of copper-zinc ores are so few as to be counted on the fingers of one's hands (Bibl. 1,2).

To clarify more precisely the character of the distribution of rare and dispersed elements in the products of concentration, the samples of the copper-zinc ores of certain concentrator plants retreating the ores of various deposits were subjected to an investigation.

Concentration Yielding Bulk Copper-Zinc Concentrates and Pyritic Tailings

Tables 1, 2 and 3 cite the results of the distribution of elements, including the rare ones, among the products of concentration of the samples of three different ores -- A, B and C.

As shown by the figures cited in the above Tables, the recovery of rare elements into copper-zinc concentrates varies within the limits of only from 10 to 30 percent (except for cadmium and indium, whose recovery reaches 60-75 percent). Such a low recovery of the majority of elements stems both from the low yield of concentrates and the geochemical and mineralogical characteristics of the processed ores and the character of the association of rare elements.

Thus, the elements most richly represented in the copper-zinc concentrates are cadmium, indium and germanium. The content of selenium and tellurium in the concentrate is 1.3- 2.3-fold and the content of thallium -- 1.1-1.5-fold.

Table 1

Distribution of Elements Among the Products of Bulk Flotation (Ore A)

Material	Yield in %	Cu	Zn	S	Pb	Fe
Received: Ore	100.0	100.0*	100.0	100.0	100.0	100.0
Obtained: Copper-Zinc Concentrate	8.77	1.12 77.3 10.0	0.91 73.9 8.10	43.64 8.4 41.8	0.029 73.5 0.27	36.0 7.47 31.0
Zinc Concentrate	0.11	0.3 3.3	5.3 46.4	0.1 34.5	<0.1 0.02	0.03 9.0
Pyritic Tailings	91.12	22.4 0.28 (-1.3)	20.8 0.22 (-6.7)	91.5 44.4 (-1.0)	26.4 0.009 (-10.0)	92.50 37.0 (-1.2)

"Nevyazka" Middlings?

*The numerator shows the percentile distribution of elements among products; and the denominator -- content in percent (for rare elements -- in grams per ton). This remark applies to all other tables in this article also.

**Accepted according to difference.

[Continued next page]

[Continued from Page 20 - Table 1 - 7]

	SiO ₂	Cd	In	Ge	Al	Se	Te
Ore	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	12	40	1.0	0.8	4.7	5.0	41
Cu-Zinc Concentrate	2.9	69.0	79.0	23.3	13.8	11.7	16.5
	4.0	30.4	9.0	2.0	6.0	6.5	78
	0.1	3.0	5.0	1.4	0.1	0.1	0.1
Zinc Concentrate	0.7	1040	47	12.0	2.9	29.0	25
	97.0	28.0	16.0**	75.3	86.1	88.2	83.4
Pyritic Tailings	13.0	12	Traces	0.6	3.6	47	38
Middlings	(-1.7)	(+3.0)	—	(18.7)	(18.9)	(2.5)	(-1.2)

Distribution of Elements Among the Products of Bulk Flotation
(Ore B)

Material	Yield in %	Cu	Zn	S	Pb	Fe	SiO ₂	Cd	In	Ge	Tl	Se	Te
<u>Received:</u>													
Ore	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
		1.37	1.20	29.7	0.04	25.0	26.0	37	1.8	1.7	3.0	70	29
<u>Obtained:</u>													
Copper-Zinc Concentrate	11.6	83.8	80.2	16.0	36.9	15.0	1.1	62.0	76.0	34.0	18.0	13.0	23.0
		10.05	8.4	39.98	0.12	30.0	2.50	170	12.0	4.3	5.0	81	58
Pyritic Tailings	80.0	16.2	19.8	84.0	63.1	85.0	98.9	38.0	24.0	66.0	82.0	87.0	77.0
		0.28	0.30	30.56	0.03	25.0	33.0	15	<1	1.2	3.3	79	28
"Nevyazka" /Middlings?	+8.4	(-0.6)	(-1.1)	(+2.0)	(+7.3)	(+9.6)	(-2.7)	(+15.5)	—	(+14.6)	(-5.2)	(-2.5)	(-1.2)

*Accepted According to
difference

Table 3

Distribution of Elements Among the Products of Bulk Flotation (Ore C)

Material	Yield in %	Cu	Zn	S	Fe	SiO ₂	Cd	In	Ge	Tl	Se	Te
<u>Received:</u>												
Ore	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
		1.34	1.42	30.8	24.0	32.0	40	2.6	1.4	4.8	44	49
<u>Obtained:</u>												
Copper-Zinc Concentrate	9.5	90.0	69.0	11.0	8.2	2.4	69.8	66.3	26.6	15.8	24.1	24.8
		14.14	10.5	32.8	21.9	7.0	330	18.0	3.5	7.2	100	113
Pyritic Tailings	90.6	10.0	31.0	89.0	91.8	97.6	31.2	32.7*	73.4	84.2	75.9	75.2
		0.17	0.5	27.8	25.8	30.30	15	<1	1.0	4.0	33	36
"Nevyazka" / Middlings?	-0.1	(-11.2)	(-1.2)	(+8.1)	(-6.2)	(+12.7)	(-11.8)	—	(+12.7)	(+11.1)	(+10.8)	+11.5

*Accepted according to
difference

It follows from the foregoing that inasmuch as pyrite concentrates contain the major part of the rare and dispersed elements present in the raw ore, therefore the utilization of these concentrates during their retreatment into sulfuric acid should be comprehensive in nature.

A mineralogical characterization of the ores and of the obtained concentrates is cited below.

Ore A. It consists of pyrite (80-85 percent), chalcopyrite, sphalerite and other minerals (four to seven percent), and nonmetallic components (eight to 12 percent). The sulphides are represented in this ore in the following order of sequence as to their extent: pyrite and sphalerite; the minerals are represented mainly by chalcopyrite and tennantite. The nonmetallic materials are represented mainly by quartz and, to a small degree, barite.

The products of concentration include: copper concentrate (containing 12-15 percent chalcopyrite, 6-8 percent tennantite, 56-58 percent pyrite, 11-12 percent sphalerite, and 9-10 percent nonmetallic mineral materials); pyrite concentrate (containing 78-80 percent pyrite, 0.4-0.5 percent cupriferous minerals, and 0.35 percent sphalerite, with the remainder represented by nonmetallic mineral components); and zinc concentrate (containing 70 percent sphalerite, 14 percent pyrite, and 7-8 percent cupriferous minerals, with the remainder represented by nonmetallic components).

The character of the distribution of gold and silver is of interest to the analysis of the behavior of certain rare and dispersed elements. In the investigated case, at a content of 0.88 grams of Au and 14.0 grams of Ag per ton of ore, the recovery of these noble metals into the copper-zinc concentrate amounted to, respectively, 25 and 55 percent (with the concentrate containing, correspondingly, 2.36 and 81.4 grams per ton, respectively).

It is also of interest to note that the Zn:Cu and Zn:In ratio remains, as a rough approximation, constant; the changes in the S:Se and S:Te ratio are more complex -- they are smaller in the copper concentrate, greater in the pyrite concentrate, and reach a still higher extent in the zinc concentrate. The reason for this will be elucidated below.

Ore B. It is a mixture of different ore types: sulphur-pyrite, low-copper, cupriferous, and zinc-cupriferous. The minerals are represented in this order of sequence of their representation in the ore: pyrite, chalcopyrite, sphalerite, tennantite, arsenopyrite, galenite, bornite, covellite, and chalcocite. The nonmetallic minerals are represented by quartz, sericite, chlorite, rutile,

apatite, and gypsum.

The products of concentration include the following minerals: copper concentrate (containing 50 percent pyrite, 22 percent chalcopyrite, six percent tennantite and other cupriferous minerals, and 11-12 percent sphalerite); and pyrite concentrate (77 percent pyrite, 0.5 percent cupriferous minerals, 0.75 percent sphalerite).

To characterize the behavior of rare elements let us point out that while the content of Au and Ag in the raw ore amounts to 2.18 and 34.6 grams per ton, respectively, their recovery into the copper-zinc concentrate amounted to 53.2 and 60.2 percent, respectively (with the concentrate containing, correspondingly, 9.4 and 169 grams per ton, respectively). The recovery of gold and silver into the zinc concentrate had correspondingly amounted to 5.7 and 6.1 percent, respectively (with the concentrate containing 5.85 and 99.9 grams per ton, respectively).

The S:Se and S:Te ratio remains approximately the same in the products of concentration, except for the zinc concentrate where this ratio is approximately twice as high; this attests to the smaller degree of dissemination of selenium and tellurium in sphalerite as compared with pyrite, chalcopyrite and other minerals. The Zn:Cu ratio remains nearly constant in all products of concentration. The nature of the Zn:In ratio is more complex, considering that indium, being mainly associated with sphalerite, is partly also associated with chalcopyrite* which in turn results in a decrease of the Zn:In ratio in the copper concentrate and an increase of that ratio in the zinc concentrate.

Ore C. It also represents a mixture of several ore types. The products of its concentration contain: copper-zinc concentrate (30 percent pyrite, 25 percent chalcopyrite, eight percent tennantite and other cupriferous minerals, and 16 percent sphalerite, with the remainder represented by nonmetallic components); and the pyritic tailings of bulk flotation (53-55 percent pyrite, 0.75 percent sphalerite, 0.5 percent cupriferous materials, and 30 percent silica).

The raw ore C contains 1.8 grams of Au and 18 grams of Ag per ton, and the recovery of these metals into the copper concentrate amounted to 51 and 65 percent, respectively (with the concentrate containing 9.0 and 110 grams per ton, respectively).

Bulk Flotation of Ores with the Segregation of Copper, Zinc and Pyrite Concentrates and Quartz Tailings

Tables 4 and 5 present the distribution of elements, including the rare ones, among the products of concn. of samples of 2 ores--D,E.

Table 4

Distribution of Elements Among the Products of Selective Flotation (Ore D)

Material	Yield in %	Cu	Zn	S	Pb	Fe	SiO ₂	Cd	In	Ge	Tl	Se	Te
Received: Ore	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Obtained: Copper Concentrate	12.81	1.92	5.10	23.1	0.25	17.0	38.0	122	3.9	3.2	5.6	29	28
Zinc Concentrate	6.77	87.8	21.7	21.6	88.5	24.6	0.4	21.9	39.3	21.5	18.5	34.8	18.7
		13.4	8.5	59.2	1.86	31.7	1.10	230	12	5.6	8.3	79	42
		4.6	69.3	9.9	2.6	3.6	0.2	70.2	52.0	22.2	7.7	7.2	7.3
Pyrite Concentrate	27.77	1.31	51.4	34.0	0.10	8.8	1.50	1400	30	11.0	6.5	31	31
		4.4	5.1	49.0	3.0	60.6	9.1	4.1	8.7*	24.8	44.4	38.2	57.7
Quartz Tailings	52.65	0.32	0.93	40.9	0.03	56.2	13.0	20	<1	3.0	9.1	40	60
		3.2	3.9	19.5	5.9	11.2	90.3	3.8		31.5	27.4	19.8	16.3
Middlings	--	0.12	0.37	8.6	0.03	3.5	67.5	10		2.0	6.0	11	9
		--	(+1.6)	(-0.6)	(-5.1)	(+2.9)	(-3.7)	(-11.0)	--	(-3.2)	(-3.0)	(-1.0)	(-1.8)

*Accepted according to difference

Table 5

Distribution of Elements Among the Products of Selective Flotation (Ore E)

Material	Yield in %	Cu	Zn	S	Fe	SiO ₂	Cd	In	Ge	Tl	Se	Te
<u>Received:</u>												
Ore	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
<u>Obtained:</u>												
Copper Concentrate	10.89	1.85 92.0	1.7 27.8	37.7 11.8	30.0 12.0	16.3 0.8	74 21.6	3.2 32.3	1.0 9.6	4.7 13.2	38 33.1	61 32.5
Zinc Concentrate	1.73	16.87 1.5	4.50 47.4	39.9 1.5	31.10 0.4	1.10 0.2	133 52.7	9.0 39.3	1.0 8.7	5.4 0.9	108 2.3	139 1.8
Pyritic Tailings	85.4	1.79 6.5	48.43 24.8	31.3 86.7	6.20 87.6	2.0 99.0	2020 25.7	68 28.4	6.0 81.7	2.3 85.9	48 64.6	50 65.7
Middlings	+1.98	0.16 (-7.6)	0.50 (-3.5)	37.0 (+3.2)	29.0 (+5.7)	18.40 (+2.8)	20 (+10.0)	1.0 (+6.2)	1.1 (-15.0)	4.5 (+4.9)	27 (+6.4)	36 (+23.5)

The mineralogical characteristics of concentration products are cited below.

Ore D. Represented by pyrite, chalcopryrite, sphalerite, tennantite, and other minerals. Its concentration products include: copper concentrate (containing 50 percent pyrite, 22 percent chalcopryrite, six percent tennantite and other cupriferous minerals, and 11-12 percent sphalerite); zinc concentrate (75-77 percent sphalerite, 16 percent pyrite, 3.0-3.8 percent cupriferous minerals); pyrite concentrate (77 percent pyrite, 0.5 percent cupriferous minerals, 0.75 percent sphalerite); quartz tailings (five-seven percent pyrite, 0.25 percent cupriferous minerals, > one percent sphalerite, and approximately 85-90 percent nonmetallic components including 70 percent quartz).

Ore E. Represented 80-90 percent by a massive texture containing 70-80 percent pyrite, five-eight percent chalcopryrite, two-three percent sphalerite, two-three percent tennantite, and one-two percent galenite. Its non-metallic mineral components are represented by quartz (five-eight percent), sericite (two-three percent) and isolated grains of barite. To the extent of 10-12 percent the ore is represented by an impregnated texture containing 40-55 percent pyrite, two-three percent chalcopryrite, one-two percent sphalerite, one-two percent tennantite, and isolated grains of galenite. The nonmetallic minerals are: quartz (25-30 percent), sericite (three-five percent), talc (three-five percent), and isolated grains of barite.

The products of concentration of this ore contain: copper concentrate (42-44 percent pyrite, 36 percent chalcopryrite, six-seven percent tennantite and other cupriferous minerals, with the remainder represented by non-metallic components); zinc concentrate (72 percent sphalerite, 11 percent pyrite, five percent cupriferous minerals); and pyritic tailings (62-68 percent pyrite, 0.75 percent sphalerite, 0.3 percent cupriferous minerals).

The raw ore contains 1.8 grams of Au and 18 grams of Ag per ton, with the recovery of these metals into concentrate amounting to 51 and 65 percent, respectively (the concentrate containing 9 and 110 grams per ton, respectively). The content of gold and silver in the zinc concentrate is correspondingly 2.3 and 18 grams per ton, respectively.

The chemical analyses underlying the afore-cited tabulations of the distribution of rare and dispersed minerals were conducted in the Chemical Laboratory of the

*[See p. 24.] The content of indium in the monomineral fraction of chalcopryrite was assessed at 10-15 grams/ton, in the zinc blende -- at 90 grams/ton, and in pyrite -- at less than one gram/ton.

Unipromed' [Scientific Research Institute of Copper Industry].*
In this connection, both standard and new, original methods
of analysis were employed (Bibl. 3-6).

Conclusions

1. Not one of the investigated rare elements provides independent mineral formations. Cadmium and indium are associated with zinc blende and in the processes of concentration they follow zinc; selenium and tellurium mainly appear to replace the sulfur of sulfides and, roughly speaking, to be distributed analogously to sulfur (with some enrichment of copper concentrates). Thallium also is distributed analogously to sulfur. Germanium is uniformly distributed among all products; however in certain ores it displays an explicit propensity for associating with zinc blende.

2. Upon a broader introduction of the selective flotation of copper-zinc ores the recovery of cadmium, indium and germanium into concentrates could be drastically increased.

3. The recovery of thallium, selenium and tellurium may and should be organized in the process of the roasting of pyrite concentrates in sulfuric-acid plants, and during the retreatment of zinc concentrates as well.

4. The solving of the copper-zinc problem of the Urals will, in turn, solve to a major extent the problem of recovering rare elements from copper-ore raw material.

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*The chemical analyses were conducted under the guidance of V.A. Oshman in collaboration with M. V. Khokholkova, Yu. N. Spirina, N. T. Zagaynova, E. A. Mikhaylova, and others.

4. The Remelting of Cathode Copper

[This is a translation of an article written by B. V. Parfanovich in Tsvetnyye Metally (Non-ferrous Metals), No. 12, December 1959, pages 78-79; CSO:3001-N/7 (7).]

The remelting of cathode copper into wire bars and ingots is conducted in flame reverberatory furnaces or in electric furnaces. The latter exist in two types: induction furnaces with a steel core, and arc furnaces.

Smelting in reverberatory furnaces cannot be conducted continuously, and neither does it ensure the necessary quality of castings, as these then have a higher content of sulfur and oxygen; the content of oxygen may not be higher than 0.02 percent.

Considering the above drawbacks of the flame furnaces, increasing attention is devoted to the use of electric furnaces, which can yield copper ingots containing less than 0.011 percent oxygen.

In selecting the type of electric furnace, some specialists favor the induction type and others the arc type.

It used to be a practice abroad to install both these types of electric furnaces, but lately an explicit preference has been given there to arc furnaces, and the number of induction furnaces is limited to three or four installations dating from the previous years.

The power rating and number of the arc furnaces used for smelting copper cathodes are continually increasing, in contrast with the rating and number of induction furnaces. At present, according to incomplete published data, 25 electric furnaces have been established and have mostly a rating of 8,000 kva and a daily productivity of as much as 700 tons of copper.

The predominance of electric arc furnaces over induction ones is attributable to a number of signal advantages of the former over the latter, as can be seen from the following comparative data:

	Electric Furnace Type	
	Induction	Arc
Unit consumption of electrical energy, in kwt-hours/ton	280-310	220-250

[Continued next page]

(continued from Page 29)

	Electric Furnace Type	
	Induction	Arc
Resistance of lining, in days	60-90	300-400
Power factor of installation, without compensation	0.5-0.55	0.7-0.75
Realized power rating of one electric furnace, in kva	3,000	8,000
Waste of metal in burning, in percent	0.1-0.3	0.22-0.27

In addition to the above advantages, the arc furnaces with high power ratings have a simpler design execution and operate more reliably.

Below is cited a brief characterization of the arc furnaces operating in the abroad.

Power rating, in kva	2,000	4,000	6,000	8,000
Capacity, in tons	--	40	50	90
Diameter of bath, in meters	--	4.5	3.66	5.4
Working voltage, in volts	90-130	90-130	173-	--
Electrode diameter, in mm	--	457	350	--
Unit rating of bath, in kwt/m ²	--	250	600	220
Daily productivity, in tons/day	170	360	480	700
Unit consumption of electrical energy, in kwt-hours/ton	240	220	245	210

The electric arc furnaces for the remelting of copper cathodes are of a design analogous to the steel-smelting electric arc furnaces -- they are cylindrical, with three electrodes situated at the three ends of an equilateral triangle.

As distinguished from the melting process in the steel-smelting furnaces, the process of the melting of copper cathodes is conducted continuously and the metal, moreover, is continuously drained off from the furnace through an oblique taphole (siphon) located in the furnace wall. The tilting of the electric furnace is practiced only prior to a prolonged shut-down of the furnace for draining all metal or for removing the insignificant amount of slag.

The start-up of the electric furnace is conducted on

coke, whereupon the red-hot coke is overlain with copper cathodes which, on heating, form a molten bath, and subsequently the electric arc burns in between the electrodes and the molten-metal bath.

The cathodes are charged in through a side port in the furnace wall; on descending into the molten bath they melt and thus there is no direct contact between the electric arc or electrodes and solid cathodes. The absence of a contact between electrodes and solid charge prevents shortcircuiting and ensures a stable performance of the furnace, which makes it possible to install furnace transformers without reactors.

An adjustable and periodic pouring of metal into chill pans is achieved by installing an intermediate tilting mixer ["holding furnace"] into which pours continuously the metal from the melting furnace. Such intermediate mixers are constituted by an induction barrel furnace with a steel core, of a high power rating but low capacity; for instance, a mixer with capacity of 15 tons and rating of 150 kwt is attached to an electric melting furnace with a rating of 6,000 kva.

The unit consumption of electrical energy in the mixers amounts to 8-10 kwt-hours/ton. Inasmuch as copper oxides are not present in the induction mixers, therefore the resistance of the hearthstone lining is several times as high as in the induction melting furnaces, i. e., the principal shortcoming of these furnaces is absent here.

A number of high-productivity plants has installed, instead of induction furnaces, unheated ladles of the closed type (barrels), which has even further simplified the melting of copper cathodes in electric arc furnaces.

To prevent the oxidation of metal and the absorption of gases by metal, the chutes are of the covered type.

In view of the obvious advantages of the electric arc furnaces, our project-design organizations should re-examine the previously adopted decisions recommending the installation of steel-core electric induction furnaces for the remelting of copper cathode.

5. Regarding the Article by P. A. Pazdnikov
and B. V. Mal'tsev "Combining Dust-Coll-
ecting and Hydrometallurgical Processing
of Dust in a Single Installation at Cop-
per Smelting Plants"

✓This is a translation of an article written
by I. L. Peysakhov in Tsvetnyye Metally (Non-
ferrous Metals), No. 12, December 1959, pages
79-80; CS0:3001-N/7 (8).7

The authors of the above article place under discus-
sion the important problem of the scheme of retreating the
dusts of copper-smelting plants; these dusts contain con-
siderable amounts of zinc, lead, cadmium, and other valu-
able metals. However, that article contains errors which
open to doubt the relevancy of the authors' suggestions.
At the very beginning of the article they state that the
gas flues, chambers, cyclones and dry Cottrell filters
collect mainly coarse dust, while the fine dust (less
than 20 microns) is expelled through the stack.

Such a sweeping description of the effectiveness of
the dust-collecting equipment is untrue. It is known from
the literature (Bibl. 1, 2, 9) and practice that, in cer-
tain cases, cyclones collect dust particles of the order of
10 microns in size, while dry Cottrell filters often quite
satisfactorily collect particles measuring several microns
in size (Bibl. 3). Complications in the performance of
dry Cottrell filters operating without the preparation of
gas are usually observed when the gas contains fume partic-
les measuring less than one or two microns in size and when
SO₃ is absent (Bibl. 4).

The workers of industry and scientific-research
organizations know that if during the pyrometallurgical
retreatment of polymetal ores the dusts become enriched --
in comparison with the raw ore -- with such components as
lead, zinc, cadmium, etc., then this occurs as a result of
the formation of fumes the particle sizes of which are
generally smaller than one micron (Bibl. 3, 4). Further,
the authors state that "...the fine dust fractions can be
collected in bag filters with humidification of the gas;
however, wet dust collection appears to be more effective
(froth apparatuses, high-speed dust collectors, scrubbers

*"Tsvetnyye Metally," No. 8, 1959

with attachments, etc.)."

This assertion is utterly unfounded. Bag filters (Bibl. 3, 4) and often wet Cottrell filters provide a very high degree of purification of gases from fine-disperse dusts -- a degree such as could not as yet be achieved in froth devices and scrubbers (< 0.05 grams per standard cubic meter [of gas]) with respect to high-disperse dusts. (< 1 micron). As we noted above, it is precisely this dust that should be of primary interest when considering the problems touched upon by the authors of the article. These authors, proceeding from erroneous premises propose the application of a three-shelf froth device, assuming that this will ensure a 99-percent dust collection.

The originators themselves of the froth apparatuses (Comrade Pozin and others) assume (Bibl. 5) that particles smaller than three or five microns are poorly collectible. The NIIOGAZ (Bibl. 6) and VNIITsvetmet (Bibl. 7) [State Scientific Research Institute of Gas Purification and All-Union Scientific Research Institute of Nonferrous Metallurgy, respectively] confirmed that particles smaller than six microns in size (carbide dust), when a two-shelf apparatus was used, were collected only to the extent of 74-80 percent, in which connection the second shelf was not effective; the dusts of the air-draft expulsions from the melting of zinc-antimony melts were collected to the extent of 72 percent, and the dusts of lead production -- to the extent of 50-80 percent.

The results obtained with froth apparatuses provide a foundation for the NIIOGAZ, in certain cases, to view their effectiveness as comparable to that of cyclones. Therefore, we assume that the most valuable part of dusts, containing zinc, lead, cadmium, and rare metals, will be poorly collected in a froth apparatus.

The assertion of the article's authors that a tower with attachment has a resistance of 400-500 mm of the water column constitutes an explicit misunderstanding. Their proposal of the use of a three-shelf apparatus is, viewed from the standpoint of dust collection, totally unjustified, inasmuch as the literature states that the second and third shelves are nearly inoperative in relation to fine dusts, and this is only logical (Bibl. 5, 6).

Likewise, the authors interpreted erroneously the problem of sulfurous anhydride. First, there exist no foundations for assuming that it will be totally collected and transformed into sulfuric acid, because a major part of sulfurous anhydride will remain in the gases, and the expulsion into the atmosphere of cold, wet and, moreover, acid gases is contrary to sanitary considerations; this

has already been confirmed by the practice of the Chimkent Plant. Second, the collection of any considerable part of sulfurous anhydride would merely lead to an excessive consumption of lime to neutralize the acid. If, as stated by the article's authors, the gases contain 2.5 grams of dust per standard cubic meter, and if about two-thirds of these 2.5 grams, i. e., about 1.7 grams, are present in the form of the oxides and sulfides of metals, then their dissolution requires the consumption of about 2.5 grams of sulfuric acid, which is equivalent to 1.8 grams of $\text{SO}_2 \cong 0.60$ percent, whereas the gases contain about three percent SO_2 .

When compiling the material balance the authors assumed that the principal part of sulfurous anhydride is oxidized to sulfuric acid. Why do they make this assumption, and on the basis of what data?

Errors have likewise been made in compiling the heat balance. The authors assumed that by feeding the froth apparatus with a gas having the temperature of 250°C it is possible to heat the fluid (having a composition that differs little from water) to 80°C . It is known from theory (Bibl. 7-9) and practice that upon contact between gas and a fluid the latter can be heated only to the so-called wet-thermometer temperature, because, upon contact with gas, the fluid evaporates. For the case in question, that temperature will be approximately 55°C (taking into account the decrease in the pressure of vapors over the acid sulfitic solution), and in practice it will be several degrees lower.

Analogously the temperature of the gases in the outlet of the apparatus will differ from fluid temperature by only two or three degrees Centigrade (Bibl. 5), whereas the authors assumed that that difference will amount to 40°C .

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