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COMPLEX PRODUCTION OF VANADIUM, TITANIUM, AND IRON FROM TITANIFEROUS MAGNETITES IN THE CHINESE PEOPLE'S REPUBLIC

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The completion by Soviet and Chinese specialists of laboratory and commercial-scale investigations for organizing the production of ferroalloys from titanomagnetites has yielded sufficient technological data for the complex commercial utilization of these ores for smelting ferrovanadium and ferrotitanium.

A characteristic example of the creative collaboration of Soviet and Chinese specialists and also of the increased possibilities of research institutes and plants in the Chinese People's Republic in solving individual metallurgical problems is the completion of investigations and commercial-scale experimental smeltings for organizing the production of ferrovanadium and ferrotitanium in the Chinese People's Republic.

The technological cycle from raw materials to smelting of the ferroalloy represents a complex system of metallurgical reductions in connection with the need for the complex utilization of all the useful elements of the ore (titanomagnetites).

After an examination of the mine, concentration plant, and ferroalloy works which were destroyed by the Japanese usurpers during their retreat and which were not in operation for more than 10 years, a program of investigations and commercial scale experimental smeltings was scheduled which was confirmed by the Ministry of Heavy Industry of the Chinese People's Republic; moreover, it was decided that all the investigations and plant experiments were to be carried out by the efforts of the research organizations and plants of the Chinese People's Republic with the technical assistance of the author and the participation of Soviet and a large group of Chinese specialists.

A. Characteristics of the Ore and Concentrates

The chemical composition of the chief varieties of the utilizable titanomagnetites are characterized by the following percentages.

Ore	Fe total	V	TiO_2	SiO_2	P_2O_5	S
Rich	53.0	0.20	14.2	2.7	0.08	0.13
Average	36.0	0.15	8.2	18.1	0.125	0.20
Lean	20.0	0.06	5.6	26.0	2.60	0.32

An examination of these data indicates a direct relationship between the content of vanadium and titanium in the ore and the amount of iron: the greater the iron content, the more vanadium and titanium in the ore. Mineralogical analysis confirmed the presence in the ore of titanium, mostly in the form of ilmenite ($FeO \cdot TiO_2$) and of vanadium in the magnetite as a substitutent element ($FeO \cdot V_2O_3$).

A reverse relationship is observed with respect to harmful admixtures i.e., with increasing content of iron in the ore, the amount of silica, phosphorus, and sulfur drops sharply.

Although the ore is good, lumpy material, the direct utilization of rich ore in a blast furnace is not expedient, since this would lead to the irretrievable loss of titanium; for when it passes into the blast furnace slags, its extraction becomes practically impossible due to the low concentration and extremely small size of the crystals. In addition, direct smelting of the ore yields a below-standard cast iron with an increased content of sulfur and phosphorus.

For this reason, in all cases in which it is technologically possible, the titanomagnetites should first be subjected to concentration. Also, along with the elimination of the gangue and harmful admixtures, it is possible, in many cases, to obtain 2 concentrates: iron-vanadium (magnetic fraction) and iron-titanium (weakly magnetic fraction). Further rational processing of the concentrates makes it possible to utilize vanadium, titanium, and iron.

We are not dealing here with the case in which separation of titanium and vanadium by concentration methods becomes impossible, making it necessary to employ electric blast furnace smelting which yields rich titanium slags.

In connection with the limitation of reserves of rich ores, the Ministry of Heavy Industry of the Chinese People's Republic formulated a plan for processing ores containing 38% Fe; this is more complex than during the previous operation of the plant primarily with rich ores and withoug taking into account the need for the rational development of the entire deposit.

However, the application of improved methods of concentration (for example, flotation) will make it possible to obtain from comparatively lean ores quality concentrates suitable for processing into ferrovanadium and ferrotitanium.

Since a certain reserve of concentrates was found at the concentration works and the ferroalloy plant, it was decided to carry out the preliminary work and the commercial-scale experimental smeltings with old iron-vanadium technological questions and to enable the Chinese comrades to acquire experience in a field of technology new to them.

The iron-vanadium concentrates which were used in the experiments had the following composition, in %:

FeO_{total} 58-60, FeO 25, V 0.38-0.39, TiO_2 8-9, SiO_2 2-3, P_2O_5 0.08,

S 0.12-0.15, Al_2O_3 2.4-3.0, MgO 0.8-1.0, CaO 0.8-1.0, MnO 0.15, Cr_2O_3 0.3

Sieve analysis of the concentrate showed 10% of fraction larger than 0.20 mm and about 40% of fraction less than 0.06 mm.

Previously, the iron-vanadium concentrate of such a composition and grain size was forwarded to the ferroalloy plant for direct chemical extraction of vanadium in the form of iron vanadate which was smelted to give ferrovanadium. The concentrate was roasted in rotary furnaces 2 m in diameter and 32 m long.

The sludge remaining after the leaching out of the vanadium, which contained 60% Fe, was not utilized, since the Japanese specialists considered it impossible to smelt it in a blast furnace (because it contained up to 10% TiO_2). The total extraction of vanadium, as ferrovanadium, from the concentrate did not exceed 25%.

The iron-titanium (ilmenite) concentrate which was found at the concentration plant had the following composition, in %:

FeO_{total} 35.12, FeO 14.0, TiO_2 36.5, V 0.07, SiO_2 4.62, P_2O_5 0.35, S 131,

Al_2O_3 2.83, CaO 0.58, MgO 1.48, MnO, MnO 0.60, Cr_2O_3 0.03.

Such a concentrate is not suitable for smelting ferrotitanium of the standards established in the USSR, since it contains little TiO_2 and too much phosphorus and silica. Previously, the ilmenite concentrate was shipped to Japan.

B. Metallurgical Method of Processing Iron-Vanadium Concentrate

The metallurgical method of processing consists of smelting the previously caked concentrate in a blast furnace with the reduction of the iron and vanadium, the resulting vandium cast iron is blown down in a converter in which the vanadium is oxidized and passes into the slag which contains 15-20 times more vanadium than the original concentrate. The vanadium is extracted from the slag by a chemical method in the form of vanadium pentoxide which is used for smelting ferrovanadium.

Such a technology which permits the utilization of both vanadium and iron was developed and accomplished in the USSR (M. N. Sobolev, D. S. Belyankin, A. M. Samarin, and various other workers of institutes and plants [1, 2, 3]) was later utilized also in other countries.

However, the blast furnace smelting of the iron-vanadium concentrate having the characteristics listed in section A above can involve considerable difficulties in connection with the high content of titanium dioxide and sulfur and the need for preliminary caking of the raw material by briquetting or agglomeration (due to the fine grind of the concentrate).

The investigation of such a complex process and the yield of necessary data for designing the plant and concentration works required the completion of not only laboratory but also of production-scale experiments. For this reason, the experiments were carried out on a commercial scale.

The first experiments for the agglomeration of about 30 t of iron vanadium concentrate were carried out in June 1954 at the metallurgical plant, the agglomeration department of which was equipped with round pans of outmoded design with air feed from the bottom, For a large content of fine fractions in the concentrate, considerable losses of vanadium and iron were observed; these amounted to 20-25% in individual cases. Experiments confirmed that the method of agglomerating the iron-vanadium concentrate in pans with air feed from the bottom is not acceptable for commercial purposes.

In September 1954 agglomeration experiments were repeated in an experimental unit of another metallurgical plant, which consisted of 2 pans: a large one with a grate, 1,000 x 1,000 mm, and 300 mm high and a small pan, 340 x 340 mm, and of the same height (distance between the grate bars was 5 mm and the air was drawn from the top). The agglomerate was tested for strength in a cylindrical drum 980 mm in diameter and 530 mm long with 3 partitions 250 mm high. In testing, a 20-kg sample of the agglomerate was fed into the drum in lumps of 50 to 150 mm and after 100 revolutions of the drum, the sample was sifted through sieves with openings of 40, and 5 mm and the content of each class was determined.

In the case of an agglomerate from ordinary iron concentrates, the following classification, as a function of the content of the 0-5 mm fraction, was adopted at the plant:

content of 0-5 mm fraction, %	26	26-28	28
class of agglomerate	I	II	Reject

Only in a small portion of the 19 sinterings (including 3 on a large pan) was agglomerate of second class obtained; in the remaining cases, the fine fraction amounted from 35 to 41%. Thus, the experiments revealed the relatively small strength of the agglomerate, although they confirmed the possibility of obtaining agglomerate suitable for blast furnace smelting from contemporary, well equipped units. The compositinn of the experimental agglomerate is listed in Table 1.

TABLE 1

RESULTS OF EXPERIMENTAL SINTERINGS OF IRON-VANADIUM CONCENTRATE

Composition of charge, %					Thickness of charge layer, mm	Duration of sintering min-sec
concentrate	return	limestone	lime	coke breeze		
52.5	40.0	1.8	1.2	4.5	250-280	12 - 13
57.5	35.0	2.0	1.0	4.5		14
56.0	33.0	2.0	1.0	8.0		15 - 16

Chemical composition of agglomerate, %					Content of 0-5 mm fraction, %
Fe_{total}	FeO	V	TiO_2	S	
55.67	18.85	0.372	7.83	0.02	41.7
56.25	15.25	0.340	-	0.01	35.6
-	-	-	-	-	26.25

Since the experimental blast furnace smelting and the blowing down of the cast iron were intended to be carried out at the third plant, with distant transportation and repeated loading of the agglomerate, which would have resulted in the formation of a large amount of fines and losses, the possibility of briquetting the concentrate by high-temperature sintering was investigated simultaneously with the agglomeration experiments.

The charge for briquetting consisted of 98% concentrate and 2% lime and had a moisture content of 11-12%. Presses with pressures of up to 200 t and equipped with rotating tables were used to make 170 x 170 x 60 mm briquets. The briquets were sintered on trolleys in a flame furnace

70 m long and divided into 3 zones -- preheating, sintering, and cooling -- and fired with blast furnace gas.

The optimum sintering temperature for briquets from the iron-vanadium concentrate was determined to be 1,390-1,410°, with holding for 15 minutes at this temperature.

After sintering, the content of ferrous iron in the briquets decreased to 4-8% instead of 26-27% in the concentrate. The content of sulfur also decreased sharply. The briquets (Figure 1) had a considerably higher strength than the agglomerate and showed the following average chemical composition, in %:

Fe_{total}	V	TiO_2	SiO_2	P	S	Al_2O_3	CaO	MgO	MnO
58.58	0.385	8.50	1.90	0.027	0.010	2.43	1.16	1.01	0.15

About 2,400 tons of briquets were prepared for the experimental blast furnace smelting. Despite the repeated loading and additional grinding of the briquets prior to charging into the blast furnace, the total losses did not exceed 8%.

C. Smelting of Vanadium Cast Iron

In October and November 1954, commercial-scale experimental smeltings of vanadium cast iron were carried out in a blast furnace having a capacity of 73 cu m (Figure 2). From 2,162 tons of briquets, 1,250 tons of cast iron were obtained.

The condition of the furnace prior to the experiments was not entirely satisfactory, because in the course of almost 3 months it operated at low speed due to insufficient feed of raw materials in connection with a river flood (after the removal of 2 sows, there were still stoppages).

The furnace has 6 tuyeres of 100 mm and slag and cast iron tap holes. During the experimental smeltings, 3 air heaters were in operation. They had a heating surface of only 2,400 square miles with a comparatively low temperature of the blast -- up to 380°. The fourth Cowper stove with a heating surface of 1,200 m^2 was started only toward the end of the experiments.

On 25 October 1954, a third of the iron ore in the charge was replaced with briquets and over a period of 2 days cast iron containing an average of 0.77% Si, 0.39% Mn, and 0.179% V was obtained from such a charge. The operation of the furnace was smooth; the slag and cast iron came out normally.

On 27 October the charge was changed to 100% vanadium briquets. For 13 days the furnace operated normally, without complications. Changes in the charge were made only in order to correct the content of sulfur and alumina. The relative amount of slag varied from 1.0 to 1.26. The composition of the raw materials used is listed in Table 2 and the operating characteristics of the furnace for different charges are listed in Table 3.

TABLE 2

COMPOSITION OF RAW MATERIALS OF EXPERIMENTAL BLAST FURNACE SMELTING

Materials	Fe total	SiO_2	Al_2O_3	MgO	MnO	V	TiO_2	S	P
Briquet	58.58	1.90	2.43	1.01	0.15	0.385	8.50	0.010	0.027
Limestone	0.34	1.08	0.87	0.72	0.68	-	-	0.028	0.009
Dolomite	0.60	2.40	1.11	17.05	0.60	-	0.01	0.009	0.010
Blast furnace slag	0.85	33.98	15.48	2.25	1.10	-	-	0.720	0.008
Fireclay brick grog	1.10	51.82	44.94	0.72	0.06	-	0.90	0.005	0.116
Silica brick grog	0.88	94.81	0.91	0.52	0.04	-	-	0.020	0.023
Manganese ore	12.64	20.90	5.73	0.82	27.80	-	-	0.037	0.320
Ash of coke A*	8.60	42.48	34.95	1.31	0.21	-	2.82	-	0.010
Ash of coke B*	4.80	61.84	24.85	0.72	0.21	-	2.14	-	0.063

* Ash Coke A contained 6.5% ash and 0.59% S and 1.14% volatiles; coke B contained 14.00% ash, 0.59% S, and 1.11% volatiles.

The cast iron obtained during this period had the following percentage compositions:

C	Si	Mn	V	Ti
4.00	0.635	0.375	0.461	0.24

for a slag composition, in % (CaO:SiO_2=1.18):

SiO_2	Al_2O_3	CaO	MgO	TiO_2	FeO	MnO	V_2O_5
27.54	13.11	32.46	6.78	11.40	1.71	1.00	0.25

Material balance showed that the reduction of vanadium from the briquets to cast iron amounted to 68% and the losses in the slag were 23.7%. It can be assumed that, with a more modern blast furnace having a smaller amount of slag, a greater extraction of vanadium is possible.

Titanium dioxide passed primarily into the slag (85-90%). With a low temperature of the blast and with contaminating fluxes, there was observed a comparatively high movement of sulfur into the cast iron, i.e., up to 11-12% of its content in the charge. Optimum operation of the blast furnace with briquets was observed under the following conditions:

(a) composition of slag, in % (CaO:SiO_2=1.15-1.20):

TiO_2	Al_2O_3	MgO
12	13-15	6-8

(b) composition of cast iron, in %:

V	Si	Mn
0.45-0.50	0.50	0.40

TABLE 3

AVERAGE VALUES OF OPERATION OF BLAST FURNACE DURING SMELTING OF CAST IRON

				Consumption, kg/t of cast iron								Blast conditions		
Period	Average daily output, tons	coke	briquet	manganese ore	limestone	dolomite	slag blast furnace	slag bessemer	grog silica brick	grog fireclay brick	scrap from chutes and ladles	consumption cum/min	temperature, in °C	presure in mm Hg
28 Oct-3 Nov	58.2	1,181	1,738	12	-	37	732	-	45	-	30	165.7	333	185.7
3 Nov-7 Nov	65.4	1,087	1,793	28	129	352	282	-	-	167	16	161.5	326	186 5
7 Nov-10 Nov	82.6	1,029	1,731	41	378	423	-	-	130	220	18	156.0	374	170 0
10 Nov-16 Nov	44.9	1,218	1,792	57	420	375	-	35	168	166	40	142.0	372	186.2

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(c) temperature of blast up to 500°.

The average consumption of briquets per ton of cast iron amounted to 1.73 t and of coke 1.1 t.

During the last 4 days (from 11 November - 15 November), the furnace operation was cold due to drop in temperature of the blast. In order to restore the normal operation, the furnace was operated to smelt foundry cast iron and then from the 20 to 22 November, vanadium cast iron was again smelted with the remaining amount of briquets.

D. Blowing Down of Vanadium Cast Iron in the Converter

A 2-ton converter (Figure 3) was used for the experimental blowing down of the cast iron; also, a railroad line was laid and trolley ladles capable of holding 2.5 t of cast iron were made for transporting the liquid cast iron. Altogether, 761 of cast iron from the blast furnace (431 blowings) and 323 t of cast iron smelted in a cupola (182 blowings) were processed in the converter.

The converter lining was made of magnesite brick which served satisfactorily. The converter has 6 tuyers of 45-mm diameter for side blast. Side blast had various advantages over bottom blast for the oxidation of vanadium.

Since the cast iron obtained during the period of experiments in the blast furnace had a wide range of vanadium and silicon contents, it became possible to investigate the process under different conditions. Blowing down of the cast iron of the above indicated average composition during the period of normal operation of the furnace proceeded quite satisfactorily. The somewhat increased content of silicon had practically no negative effect on the process since the cast iron was tapped from the blast furnace after 2-2.5 hours and the converter could cool between the heats. This facilitated a decrease in the overheating of the hearth during the blowing down in connection with the increased content of silicon in the cast iron.

The rates of burning of vanadium and silicon during the blowing down of the cast iron, as a function of their content, are shown in Figure 4.

The temperature of the hearth during the blowing down of heat 2,556 varied as follows (Figure 4):

Time from start of blowing down, in minutes	1	2	3	4	5
Temperature of metal °C	1,160	1,175	1,180	1,195	1,205

Time from start of blowing down in minutes	6	7	8	9	10
Temperature of metal °C	1,240	1,265	1,285	1,300	1,345

Chemical analysis of the sample of vanadium slag, obtained by averaging 13 average daily samples after the elimination of metal globules, gave the following results, in %:

V_2O_5	FeO	SiO_2	TiO_2	MnO	Fe_{met}	MgO	Al_2O_3	S	P
14.62	35.67	24.40	9.00	8.71	2.51	1.12	0.47	0.136	0.045

One may note the close agreement between the actual content of vanadium pentoxide in the average sample of the slag (14.62%) and the calculated average weight. (14.90%). Chemical processing of high-vanadium slag to vanadium is much more economical than processing of the concentrate.

The content of metal globules in the converter slag amounted on the average, to 11%. In order to grind the slag and remove from it the metallurgical globules, the trolley with the slag was shipped to the experimental concentration unit of the metallurgical works; the remaining slag was turned over to the ferroalloy plant.

E. Extraction of Vanadium from the Slag

In order to extract vanadium from the slag, the following operations are carried out:

(a) grinding and removing metal globules, after which only a small amount of dispersed metal inclusions remains in the slag;

(b) oxidation roasting of the slag with salts of alkali metals at 750–850° to convert the vanadium into the soluble form (for example, $NaVO_3$);

(c) leaching out the soluble compounds of vanadium with hot water with subsequent treating of the slurries with a weak acid solution for additional extraction of vanadium;

(d) precipitating vanadium pentoxide from the solutions by heating them to the boiling point;

(e) fusing vanadium pentoxide and preparing it in lump form for smelting ferrovanadium.

The roasting of the slag is of the greatest importance, for the extent of vanadium extraction depends largely on it. The roasting is carried out in an oxidizing atmosphere for converting trivalent vanadium into pentavalent vanadium.

Investigations have shown that the vanadium in the slag is in the form of spinel [1, 2]. The chemical mechanism of the roasting can be characterized by the following equation:

$$(V_2O_3) + 2NaCl + 3/2\ O_2 = 2NaVO_3 + Cl_2.$$

However, this process does not go to completion and the roasted slag also has vanadium compounds that are soluble only in acid.

At the experimental grinding unit of the concentration plant of the metallurgical works, the wet process with the closed cycle is used for

For a content of about 1% silicon and 0.47% vanadium (heat 2,556), the blowing down of the metal to a content of 0.02-0.03% vanadium dragged on up to 10 minutes even in the small converter. This confirms the need for allowing, under normal operating conditions, a silicon content in the cast iron not exceeding 0.3-0.4% and a vanadium content of 0.45-0.55%. The optimum temperature for the start of the blowing down(filling) for such a cast iron can be considered as 1,200 ±20°C.

In the course of 13 days 550 t of liquid metal with 2,536 kg vanadium, were poured into the converter and 26,245 kg of vanadium slag with 2,184 kg vanadium were obtained. The extraction of vanadium from the cast iron into the slag amounted to 86.1% and the yield of semiproduct (metal) was 90% of the cast iron poured into the converter (86% in ingots and 4% in gates).

During a comparatively cold operation of the blast furnace, a batch of cast iron was tapped with an average vanadium content of 0.342% and, during blowing down, its extraction into the slag amounted to 85.4%. The corresponding rates of burning of vanadium and silicon are shown in Figure 4 (heat 2,675).

The blowing down of 323 t of cast iron with preliminary melting in a cupola is of definite interest. Experiments were carried out for the extraction of vanadium from the solid cast iron which was unavoidably obtained during tapping from the blast furnace in connection with the limited capacity of the ladles. The question of the extent of combustion loss of vanadium in the cupola was also clarified.

During the melting, 125-130 kg coke, 20-25 kg limestone, and 5-10 kg feldspar were added per 1,400 kg of the charge in the cupola. If the vanadium cast iron contains little silicon, then the combustion loss of vanadium may be as high as 20-25%, which was observed in the given case and especially in the first heats. However, when the Si content in the cast iron reached up to one percent, the combustion loss of vanadium dropped to 10-12% and even lower. Silicon and titanium are protective elements for vanadium, which is confirmed by their considerable oxidation during the melting of the cast iron.

The overall results of the blowing down of vanadium cast iron are listed in Table 4.

TABLE 4

RESULTS OF THE BLOWING DOWN OF VANADIUM CAST IRON

Period	Characteristics of period	Added to converter		Obtained		Extraction of vanadium %
		cast iron	Vanadium in cast iron, kg	slag, tons	vanadium in slag, kg	
28 Oct -10 Nov	Normal operation of blast furnace	550	2,536	26.2	2,184	86.1
11 Nov -15 Nov	Cold operation of blast furnace	135	462	4.8	395	85.5
15 Nov -20 Nov	Melting of cast iron in cupola	323	1,130	15.2	962	85.1
20 Nov -22 Nov	Non-normal operation of blast furnace	76	285	3.9	242	84.9
	Total	1,084	4,413	50.1	3,783	85.7

grinding, classification, and magnetic separation. Almost pure slag without globules passes from the classifier into the run-off. Hence one can exclude magnetic separation during good control. An inherent shortcoming of the wet grinding is the need for drying the slag.

Experiments on the extraction of vanadium from the slag were first carried out on a laboratory scale, with roasting in a muffle furnace for one kg of charge and then on a semicommercial scale with equipment of the roasting and chemical departments.

The semicommercial furnace (Figure 5), which is fired by city gas, has a length of 8,650 mm and an effective diameter of 600 mm; a combustion chamber was built in (Figure 6). For 24 revolutions of the furnace per hour its loading capacity amounts to 2-2.5 per 24 hours. Along the length of the furnace there are 6 openings for the thermocouples which are used for checking the results of the systematically employed optical pyrometer (Figure 6). The temperature in the furnace varies along the length from 850 to 430° (including the locations of the discharge of the roasted charge and of the exit of the gas).

Three covered openings have been built along the length of the furnace to allow sampling of the charge.

The roasted charge is discharged directly into a tank containing water at 50°.

In preliminary laboratory experiments on the aqueous leaching out of vanadium, the results obtained were good.

The optimum extraction of vanadium (80-85%) is attained by adding to the slag 12% of table salt and by roasting at 800-830°. When sodium sulfate is added instead of table salt, it is necessary to raise the temperature by 100-150°; in addition, the charge is sintered into dense spheres and the extraction of vanadium drops.

During a 3-fold aqueous extraction of the roasted charge with NaCl and vacuum filter, the residual content in the slurry of vanadium pentoxide did not exceed 0.25%. If weak solutions after the second and third leaching out are utilized for the first leaching out, then the concentration of vanadium pentoxide in the main aqueous solution can be raised to 45-50 g/l.

For acid leaching out a 6% solution of sulfuric acid was used.

The precipitation of vanadium pentoxide took place at 95° and a solution acidity of pH=0.5 during 2-2.5 hours; also, the content of V_2O_5 in the run-off waters was brought down to 0.3 g/l. (for an original concentration of 30-40 g/l.).

In the experimental order, an investigation was made of the precipitation of vanadium in the form of ammonium vanadate and from the residual solutions, in the form of calcium vanadate.

As a result of the processing of about 22 t of slag, we obtained over 3.9 t of crude vanadium pentoxide which was discharged for experimental smelting of ferrovanadium and other utilization. A portion of the vanadium pentoxide was remelted at about 800° into a lumpy product of the following percentage composition:

V_2O_5	Fe_2O_3	SiO_2	Al_2O_3
91.0	1.40	0.40	1.39

F. Laboratory Smelting of Ferrovanadium

Silicothermal smelting in a closed furnace gives a good yield of ferrovanadium. Since no such furnace was available and the amount of fused vanadium pentoxide was comparatively small the first experimental laboratory smeltings of ferrovanadium were carried out by the aluminothermal method.

It was established that it is possible by the aluminothermal method to obtain from the fused vanadium pentoxide a ferrovanadium with practically any content of vanadium. The organization of smelting of standard ferrovanadium (35-40%) does not present any difficulties. However, in case of the use of iron vanadate, the alloy prepared by this method contains not over 33% vanadium.

G. Chemical Processing of Iron-Vanadium Concentrates

Considering the possibility of starting the ferroalloy plant before organizing the metallurgical processing of the iron-vanadium concentrate, we also carried out extensive laboratory investigations of chemical processing of this concentrate in order to determine the possibility of improving the previously obtained results. About 100 roasting and leaching out experiments were conducted under different conditions with charges of one kg and also the same number of experiments with charges of 100 g.

The results have shown that better extraction of vanadium is attained by roasting the concentrate with sodium sulfate at a temperature of 1,100-1,150°. In order to increase the concentration of vanadium pentoxide in the solutions to that required for precipitation, it is necessary to leach out 7-8 times with the same solution.

The precipitation of iron vanadate is possible with a smaller concentration of vanadium in the solutions, but as was pointed out earlier the ferrovanadium from the iron vanadate cannot contain more than 33% vanadium.

H. Improving the Quality of the Ilmenite Concentrate

In order to organize the smelting of standard ferrotitanium, laboratory and semicommercial investigations were conducted to improve the quality of the iron-titanium (ilmenite) concentrate.

It was intended to raise by concentration the content of titanium dioxide to 40% and higher (instead of the existing 35%) and to lower the content of P_2O_5 to 0.07% (instead of 0.35%) and of silica to 2.5% and lower (instead of 4.6%). The sulfur can be removed to the required limit by means of oxidation roasting of the concentrate at a temperature of about 1,200°.

After the laboratory investigations, experiments were carried out with a commercial batch of the concentrate (over 200 t).

The titanium-containing mineral in the ore is mostly ilmenite and also the small amount of rutile, titanomorphite.

The minerals of the veinstone include mostly magnetite, apatite, calcite, and chlorite. During grinding of the original ore to 100 mesh (0.15 mm), a greater portion of the ilmenite, a magnetite, and veinstone form free grains. During the investigation, a check was made of 6 methods for improving the quality of the concentrate:

(a) flotation;
(b) magnetic separation with a high-voltage magnetic field;
(c) magnetic separation with a low-voltage magnetic field;
(d) magnetic separation with a high-voltage magnetic field and flotation;
(e) both types of magnetic separation
(f) both types of magnetic separation with the addition of flotation.

The best values of extraction were obtained by flotation (Table 5).

TABLE 5

RESULTS OF ADDITIONAL IMPROVEMENT OF ILMENITE CONCENTRATE

Method of improvement	Percentage Composition of concentrate II			Extraction of Ti, %
	TiO_2	P_2O_5	SiO_2	
Flotation	40.07	0.06	3.25	85.40
	39.67	0.07	2.73	86.63
	40.56	0.05	2.40	84.01
	39.16	0.08	3.50	88.14
	40.72	0.075	2.83	91.52
Magnetic separation with high voltage magnetic field and flotation	39.61	0.13	3.61	83.86
	40.47	0.14	3.50	87.35
Both types of magnetic separation and flotation	40.71	0.19	3.90	78.99
	40.97	0.13	2.64	83.26

With a laboratory concentration unit, products of the following percentage composition were obtained:

Product	Yield	TiO_2	P_2O_5	SiO_2	V_2O_5	Fe	S
Concentrate II	67.4	40.56	0.05	2.40	0.14	36.78	0.26
Commercial product	7.5	29.57	-	5.92	-	-	-
Tailings	25.09	23.06	-	7.99	-	-	-

The processing of a commercial batch of the concentrate by the method of flotation (A) and flotation with classification (B) gave the following results, in %.

	Method of processing	
	A	B
Yield of concentrate II	57.0	43.8
Extraction of titanium	64.8	52.3
Content in concentrate II:		
TiO_2	40.15	43.02
P_2O_5	0.069	0.10

A certain portion of the concentrate was processed only on the classifier with preliminary grinding in a large ball mill. The yield of secondary concentrate with 43.02% TiO_2 and 0.08% P_2O_5 amounted thereby to 42.1% with a titanium extraction of 52%.

The resulting concentrate was subjected to roasting in a rotary furnace in order to remove the sulfur. The secondary concentrate should be considered suitable for smelting standard ferrotitanium. Laboratory aluminothermal smeltings of secondary concentrate confirmed the full possibility of obtaining such ferrotitanium.

Conclusions

1. For the complex utilization of vanadium, titanium, and iron of titanomagnetites, it is expedient to concentrate the ore, with the separation of 2 concentrates, and to utilize a metallurgical method of processing the iron-vanadium concentrate.

2. The technological data obtained from the completion of commercial and laboratory experiments make it possible to carry out the design of ferroalloy and metallurgical plants and of a concentration works with the utilization of ores of average quality.

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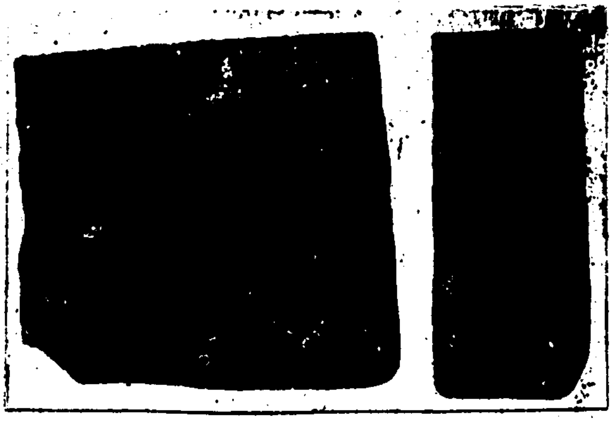

Figure 1. External view of briquet of iron-vanadium concentrate

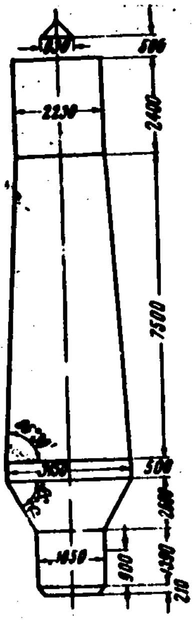

Figure 2. Profile of blast furnace in which vanadium cast iron was smelted.

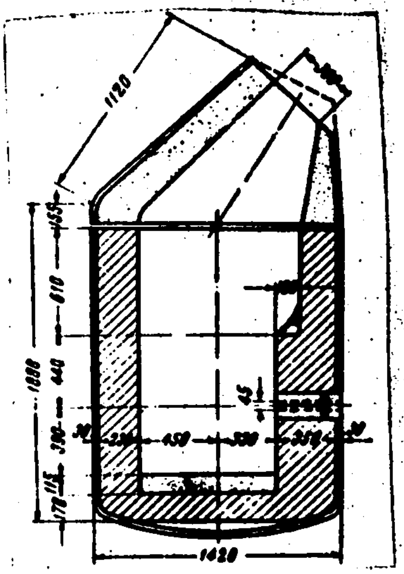

Figure 3. Converter for blowing down cast iron

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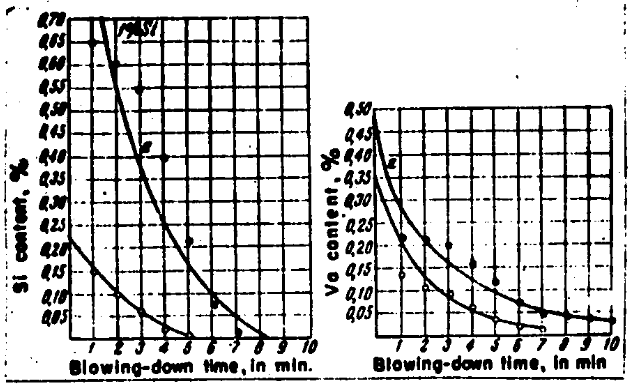

Figure 4. Rates of combustion of vanadium and silicon during the blowing down of the heats. a, 2556; b, 2675.

Figure 5. Semicommercial furnace for roasting vanadium slag

Figure 6. Combustion chamber of the furnace for roasting the slag

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