

Indicative Vanadium Deportment in the Processing of Titaniferous Magnetite by the Roast-Leach and Electric Furnace Smelting Processes

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An evaluation of vanadium deportment in the processing of titaniferous magnetite (titanomagnetite) by roast-leach vanadium primary production and electric furnace smelting vanadium co-production processes was conducted. The data for the roast-leach process were generated from a matrix of tests, followed by a bulk-scale test conducted at Mintek, whereas that of the electric furnace smelting option was acquired from the open slag bath furnace (OSBF) and shaking ladle process flowsheet data of Evraz Highveld Steel and Vanadium Corporation (EHSV). The evaluation showed that the cumulative vanadium extraction as V_2O_5 from titanomagnetite through the roast-leach approach was 82%, whereas that of OSBF-shaking ladle flowsheet at EHSV was 71% at the stage of the vanadium-rich slag. The cumulative vanadium extraction in the smelting process would be 58% at the final stage of V_2O_5 extraction from the vanadium slag through the prevalent roast-leach process if the data from the Mintek roast-leach process of titanomagnetite are adopted due to unavailability of commercial vanadium slag processing data in the open literature. At this stage, the roast-leach process appears to be superior compared with electric furnace smelting in terms of overall vanadium recovery; however, it should be noted that the smelting approach is a co-production process in which iron is another valuable product that is used in the steel making.

INTRODUCTION

Titaniferous magnetite (titanomagnetite) is a valuable source of iron and potentially titanium, and is the primary source of vanadium in the world (Fischer, 1975; Roskill, 2010). Vanadium is largely used in the steel industry in the form of ferrovanadium. Vanadium pentoxide (V_2O_5) is a valuable chemical compound in the catalysis and other chemical industries, and is also a versatile intermediate product for other products like ferrovanadium and Fe-Ti-Al-V-Si alloys (Goldberg et al., 1992; Moskalyk and Alfantazi, 2003). Vanadium extraction from titanomagnetite and its intermediate products, in particular V_2O_5 , can be conducted through numerous process technologies (Gupta and Krishnamurthy, 1992; Rohrmann, 1985; Song et al., 2014; Steele and Wilson, 1966; Taylor et al., 2006; Zhao et al., 2014). Amongst these, direct roast-leach and electric furnace smelting processing of titanomagnetite for vanadium primary and co-production, respectively, are established and predominantly used in commercial operations (Gupta and Krishnamurthy, 1992; Jones, 1992; Rohrmann, 1985; Taylor et al., 2006).

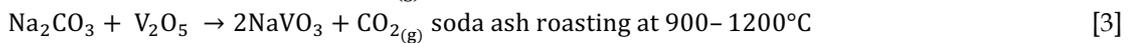
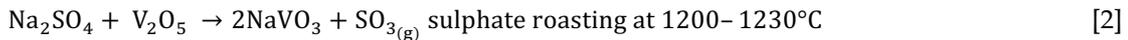
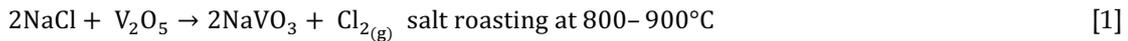
This study was formulated to evaluate the cumulative vanadium deportment in the operations of these prominent vanadium extraction process technologies. Data for the roast-leach process were generated from bulk-scale testwork conducted at Mintek, whereas those for the electric furnace smelting option were acquired from the published operational data of Evraz Highveld Steel and Vanadium Corporation (EHSV).

Overview of Roast-Leach Process

Vanadium primary production, as V_2O_5 , through the direct roast-leach processing of titanomagnetite, typically entails three stages, namely; roasting, leaching, and precipitation and calcination (Steele and Wilson, 1966; Rohrmann, 1985; Gupta and Krishnamurthy, 1992; Moskalyk and Alfantazi, 2003; Mefos, 2006).

Roasting

The purpose of the roasting stage is to convert vanadium species in the titanomagnetite to leachable phases. In this process, the titanomagnetite is blended with $NaCl$ / Na_2SO_4 / Na_2CO_3 and subjected to high-temperature treatment, as shown by Equations [1] to [3]. The general product of roasting is the water-soluble sodium vanadate ($NaVO_3$).



Leaching

Sodium vanadate is typically leached using water as lixiviant; however, the presence of Ca, Mg, Fe and Al in the titanomagnetite can cause the formation of water-insoluble vanadates, and culminate in low vanadium extraction efficiencies. The formation of the insoluble vanadates is effectively addressed by optimising roasting and leaching conditions. Moreover, in a case where Na_2CO_3 is used during roasting, the vanadium can be recovered from an insoluble calcium vanadate to the soluble sodium vanadate, as shown in Equation [4]:



Precipitation and calcination

Vanadium is recovered from the pregnant leach liquor by precipitation with an excess of ammonium chloride or sulphate solution to form an ammonium (poly)vanadate precipitate. The most common precipitate is ammonium metavanadate (AMV). AMV is typically calcined to remove crystalline water and ammonia and produce a high-purity V_2O_5 material.

The roast-leach processing of titanomagnetite is typically conducted using the soda ash option, *i.e.*, Na_2CO_3 roasting, followed by water leaching and AMV precipitation and fusing at 850°C to produce high purity V_2O_5 flakes (Rohrmann, 1985; Roskill, 2010). The by-products of Na_2CO_3 roasting are relatively less problematic and the vanadium extraction efficiencies are generally superior. Roast-leach processing of titanomagnetite is established and used in several commercial processes, including the Vantra (Vanchem) and Windimurra processes (Mefos, 2006; Taylor et al., 2006). New commercial operations based on the roast-leach flowsheet, such as Bushveld Minerals Ltd (Bushveld Minerals Ltd., 2016) and Largo Resources Ltd (Largo Resources Ltd., 2016), are under development and are expected to start operating in the near future.

Overview of Electric Furnace Process

Following pioneering work conducted by Dr William Bleloch in 1949 (Rohrmann, 1985), in which he successfully demonstrated the smelting of Bushveld Complex titanomagnetite using submerged arc furnace (SAF) technology, the smelting of titanomagnetite for the extraction of vanadium in a pig iron metal was started in the 1960s by EHSV and New Zealand Steel (NZS; processes iron sands). The EHSV operation was based on SAF technology, whereas NZS opted to use open slag bath furnace (OSBF) technology (Kelly, 1993; Steinberg et al., 2011). EHSV has since converted their SAFs to OSBFs to improve their process control and maximise vanadium recoveries to the metal phase (Steinberg, 2008). During the development of electric furnace titanomagnetite smelting technology, it was believed that the use of conventional blast furnace technology would be unfeasible owing to possible formation of high-melting reduced titanium species that would require a higher coke rate and would descend into the cohesive zone as unmelted solids. However, some years later, blast furnace

technology was successfully commissioned for the smelting of titanomagnetite in China by Panzhihua Iron and Steel Corporation (Pangang) and later by Chengde Iron and Steel (Chengde), and in Russia by Nizhny Tagil Iron and Steel (NTMK) (Evraz NTMK, 2016; Roskill, 2010; Steinberg et al., 2011; Zhang et al., 2007).

The typical titanomagnetite smelting operation, as used by EHSV, entails the feeding of titanomagnetite, coal reductant and flux, *i.e.*, dolomite and quartz, into a pre-reduction kiln. The pre-reduced material is conveyed to a furnace where it is smelted to produce a vanadium-bearing pig iron metal and a by-product titania (TiO₂)-rich slag. The by-product slag is routed to the slag waste dump. The vanadium-bearing pig iron is tapped into a shaking ladle where it is subjected to controlled oxygen potential for strategic oxidation of vanadium into the slag phase. The iron metal from the ladle is transferred via the tapping spout into the basic oxygen furnace (BOF) for the production of steel. The vanadium slag, containing as high as 25% V₂O₅, is then removed by up-ending the ladle above a slag pot. The vanadium slag is crushed and the iron entrainment is removed by means of an overhead magnet. Part of the slag is marketed to the ferroalloy or steel industry for the direct production of ferrovandium (Rohrmann, 1985; Roskill, 2010; Steinberg et al., 2011).

The rest of the slag is marketed to producers of high-purity V₂O₅ chemical for use in the chemical and catalysis industries and for the production of other vanadium products like ferrovandium and other alloys (Jones, 1992; Mefos, 2006; Rohrmann, 1985; Roskill, 2010; Steinberg et al., 2011). Extraction of V₂O₅ is typically conducted by the roast-leach process discussed above. In several roast-leach operations, the vanadium slag is blended with titanomagnetite ore fines to lower the vanadium concentration in the roast feed to about 7% to prevent sintering of the vanadium slag at the required roasting temperatures (Rohrmann, 1985). Soda roasting, followed by water leaching and precipitation of vanadium into AMV using ammonium sulphate, is the most commonly used method today owing to the production of by-products that are relatively less environmentally hostile (Mefos, 2006).

MATERIALS

Titanomagnetite mined from the main magnetite layer (MML) of South Africa's Bushveld Complex was used in the study. The particle size of the roast-leach titanomagnetite feed was D₉₀ 75 µm. The typical bulk chemical composition of the titanomagnetite feed to the EHSV process was acquired from literature (Steinberg et al., 2011), whereas that of the roast-leach process was determined at Mintek. The bulk chemical compositions of the feed titanomagnetites to the two processes are given in Table I. They have similar chemical composition, with an average V₂O₅ concentration of 1.7%. The elemental compositions of the feed titanomagnetites are also included in Table I to facilitate evaluation of vanadium deportment in elemental form.

Table I. Bulk chemical compositions of the feed materials (mass %)

	Fe ₃ O ₄	MgO	Al ₂ O ₃	SiO ₂	CaO	TiO ₂	V ₂ O ₅	Cr ₂ O ₃	MnO
Mintek lab-scale tests	78.4	1.68	3.62	1.28	0.30	11.8	1.73	0.34	0.26
EHSV operation	76.5	1.60	4.80	2.00	0.10	12.7	1.65	0.40	0.30
	Fe	Mg	Al	Si	Ca	Ti	V	Cr	Mn
Mintek lab-scale tests	55.1	1.01	1.92	0.60	0.21	7.07	0.97	0.23	0.20
EHSV operation	54.8	0.96	2.54	0.93	0.07	7.61	0.92	0.27	0.23

All reagents used in the roast-leach processing of titanomagnetite were of analytical grade. De-ionised water was used as a lixiviant. The bulk chemical compositions of samples were determined by inductively coupled plasma optical emission spectroscopy (ICP-OES; Varian Vista-PRO CCD simultaneous instrument).

EXPERIMENTAL PROCEDURES

Roast-Leach Experiments

The roast-leach testwork conducted at Mintek adopted the soda ash roasting and water leaching approach because this option is reportedly most common in industrial applications (Mefos, 2006).

Roasting

The roasting conditions were deduced from the best roasting conditions established at Mintek in the past for processing a similar titanomagnetite material. Titanomagnetite was mixed with Na₂CO₃ in a mass ratio of 100:5. Roasting was conducted in a rotary kiln that has a sample mass capacity of about 10 kg. Six batches of titanomagnetite were blended with a saturated Na₂CO₃ solution (prepared by dissolving 500 g Na₂CO₃ in 1.5 L deionized water per batch). Each batch, containing 15% moisture, was thoroughly blended, and subsequently oven-dried overnight at 105°C.

The dried batch samples were separately fed into a cold rotary batch kiln furnace equipped with a steel reactor shell. The furnace was heated at a rate of 10°C/min to 1150°C and kept at this temperature for 60 min before being cooled to room temperature. The standard rotation speed of the rotary kiln was 3 rpm. The batch products were mixed together to make a composite of the bulk titanomagnetite roast of about 65 kg in mass. A 12 kg mass of the roast material was sampled from the bulk composite to investigate the best leaching conditions. The remainder of the sample was stored for use as feed in the bulk testwork.

Leaching

The effects of particle size distribution of the feed (D₁₀₀ of 25 µm and 1000 µm), pH of the slurry (natural pH and 7.8) and pulp density (30 m/m% and 65 m/m%) on the leaching efficiency of V and detrimental species (Al, Si and Cr) were investigated. The pulp density was expressed as the solid contents in mass percent (m/m%), as shown in Equation [5]. In all these studies, the reaction kinetics were evaluated over a total period of 60 min. The testwork was conducted in a cylindrical reactor equipped with a thermocouple, pH probe, hot plate and overhead stirrer with an impeller.

$$\text{Pulp}_{\text{density}} = \% \text{ solids} = \frac{\text{Mass}_{\text{solids}}}{\text{Mass}_{\text{solid}} + \text{Mass}_{\text{lixiviant}}} \times 100 \quad [5]$$

The general procedure for the leaching testwork entailed the slurring of the feed roast (700 g) in pre-heated (70°C) deionised water to achieve a target pulp density. The slurry was agitated by means of the overhead stirrer that was set at 250 rpm. For kinetic studies, a 20 mL slurry volume was withdrawn from the reactor by means of a pipette. The sample withdrawal intervals were 10, 15, 30, 45 and 60 min. Each subsample was centrifuged and the clear solution was subjected to bulk chemical analysis by means of ICP-OES. The remaining slurry was returned back to the reactor for subsequent solids accounting.

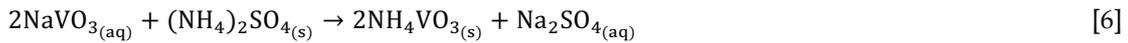
At the end of the 60 min reaction time, the slurry was allowed to cool and subsequently filtered using a Buchner funnel. The volume of the filtrate and the mass of the residue were recorded. The residue filter cake was re-slurried and washed three times, initially with alkalized deionised water (deionised water alkalized to the pH of the slurry) and thereafter with deionised water at a mass ratio of wash liquor to wet cake mass of 1:2.5. The filtrate recovered after 60 min and the washed and dried (oven-drying at 50°C overnight) residue were subjected to bulk chemical analysis.

The conditions achieved from the best vanadium leaching were adopted for processing a bulk titanomagnetite roast of about 55 kg in a leaching tank. The water leach filtrate of the bulk test was subjected to evaporation to concentrate the vanadium content in the filtrate. The evaporator was made of stainless steel and coated with glass lining. To complete the evaporation, the evaporator tank was closed and fitted with a suction point where a vacuum line from the condenser was fitted. The filtrate in the evaporator was heated to boiling point (about 90°C) by means of an oil heat exchanger. The evaporated solution was cooled in the condenser. After reaching a constant volume, the vacuum was stopped and the solution was drained into a dedicated container. The pregnant solution was

allowed to cool further under natural conditions. A sample of the solution was subsequently subjected to bulk chemical analysis for recovery evaluation.

Precipitation and calcination

The conditions for AMV precipitation and calcination were not investigated in this testwork programme; instead the base test conditions were deduced from the previous testwork conducted at Mintek. AMV precipitation was only conducted on the bulk sample. This entailed transferring the concentrated solution into a reactor that was equipped with an overhead stirrer, pH and temperature meters. The pH of the solution was controlled to about 7.8 using 5.1 M H₂SO₄ solution. A 200% stoichiometric amount of ammonium sulphate was added into the solution as dry powder to precipitate vanadium according to Equation [6]. The test was allowed to run for 4 h under conditions of constant pH and temperature of 7.8 and 100°C, respectively.



At the end of the test, the reactor contents were filtered using a Buchner funnel. The filtrate from this single-stage precipitation was recovered and subjected to chemical analysis. The wet solids were washed three times by re-slurring, initially with alkalized deionised water to the pH of the slurry, and thereafter with deionised water with a mass ratio of wash liquor to wet cake mass of 1:10. The washed AMV cake was dried in an oven at 50°C overnight.

As represented by Equation [7], the cake was calcined in a muffle furnace at 300°C in air overnight to decompose AMV to produce the valuable V₂O₅ product and gaseous NH₃ and H₂O by-products:



Electric Furnace Smelting Option

Vanadium deportment in the electric furnace smelting of titanomagnetite was investigated in the EHSV operational flowsheet. The available EHSV data cover the vanadium deportment and extraction until the vanadium slag production stage (Steinberg, 2008; Steinberg et al., 2011). There is a general secrecy concerning the mass balance data for the extraction of V₂O₅ from the EHSV vanadium slag; however, because it is known that the vanadium slag is generally processed through a soda roast-leach process (Rohrmann, 1985; Roskill, 2010), the V deportment data obtained from the Mintek roast-leach testwork were adopted to estimate the V deportment in the processing of EHSV vanadium slag. The discrepancy that would be brought about by differences in the starting V₂O₅ concentration between the Mintek roast-leach titanomagnetite feed and the EHSV vanadium slag (which is generally diluted with titanomagnetite prior to processing) was deemed acceptable because this study was only indicative in nature.

RESULTS AND DISCUSSION

Roasting

Roasting of 10 kg batches of titanomagnetite and soda mixture in the rotary kiln gave an average 3% mass loss that was attributed to residual material attached to the walls of the reactor and CO₂ emission. The roast sample formed soft lumps that were broken down inside a plastic bag by means of a rubber hammer to a D₁₀₀ of 1000 μm. Chemical analysis of the titanomagnetite roast composite was conducted on a pulverized sample and the results, in both the oxide and elemental form, are shown in Table II.

Table II. Bulk chemical composition of the titanomagnetite roast composite (mass %)

Fe ₃ O ₄	MgO	Al ₂ O ₃	SiO ₂	CaO	TiO ₂	V ₂ O ₅	Cr ₂ O ₃	MnO
79.7	1.53	3.36	1.24	0.17	11.1	1.48	1.20	0.25
Fe	Mg	Al	Si	Ca	Ti	V	Cr	Mn
55.4	0.92	1.78	0.58	0.12	6.65	0.83	0.82	0.19

Leaching

The objective of the leaching stage was to maximize the vanadium leaching from the roasted material. Vanadium extraction efficiency was calculated as shown in Equation [8]. Leaching of the detrimental elements, such as Al, and Cr, was also evaluated, in particular during the pH studies. The best leaching conditions that were adopted in the bulk testwork were determined from the test that reported maximum vanadium extraction with minimum leaching of the detrimental elements.

$$\% V_{\text{extraction}} = \frac{\text{Mass } V_{\text{leachate}}}{\text{Mass } V_{\text{feed solids}}} \times 100 \quad [8]$$

Effect of particle size distribution

The effect of particle size distribution (PSD) was studied at D₁₀₀ of 1000 µm and D₉₀ of 25 µm, in which case the pulp density and pH were kept constant at 65 m/m% and 7.8, respectively. The results, given in Figure 1, show that the best vanadium extraction was generally achieved when the coarser material (D₁₀₀ of 1000 µm) was used as feed. Lower V extraction from the finer grind size was attributed to possible oxidation during milling, which might have resulted in the formation of insoluble phases. It also appears that, in both particle size ranges, V extraction from 30 min and above increases with increasing the reaction time up to about 100% extraction after 60 min. The error in the V extractions above 100% were attributed to measurement uncertainty as a consequence of V being a minor element in the filtrate. The V extraction results suggest that there is no need to mill the feed prior to leaching.

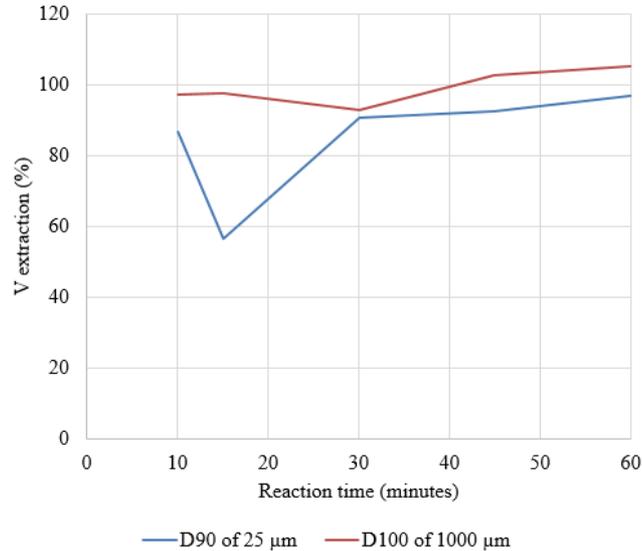


Figure 1. Effect of PSD on the vanadium extraction from the titanomagnetite roast.

Effect of pH

The effect of pH on V extraction from the roasted material was studied at natural pH (not controlled) and controlled pH of 7.8. In this study, the PSD and the pulp density were kept constant at D₁₀₀ of 1000 µm and 30 m/m%, respectively. As established in the historical work conducted at Mintek, the leaching behavior of detrimental elements can be controlled by decreasing the pH of the slurry. Leaching behaviors of Al, Si and Cr were also monitored at the studied pH conditions. The results are shown in Figure 2. The best V extraction was achieved when the pH was controlled to 7.8 throughout

the studied reaction times. The leaching of Si (averaging zero and not plotted) and Al were very low at both pH points. The Cr extraction of about 14% at both pH points is worrisome, considering that the Cr and V concentrations in the feed are similar. The Cr content in the subsequent AMV and V_2O_5 products has to be monitored to establish if it is within the specified grade.

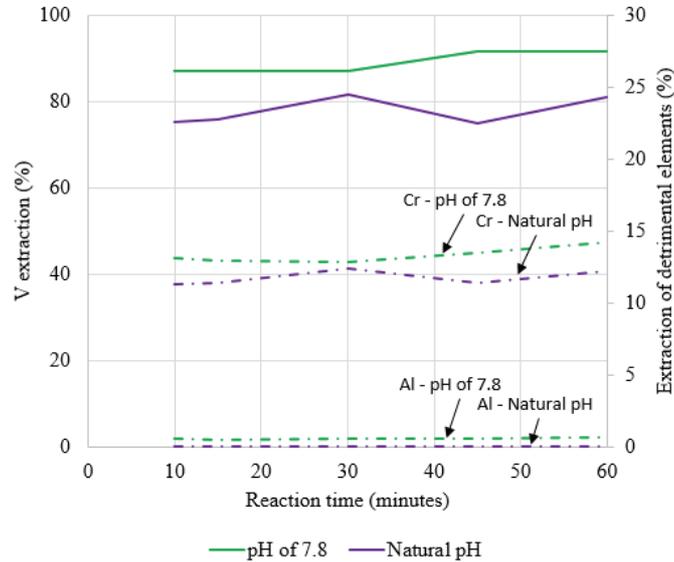


Figure 2. Effect of pH on the vanadium extraction from the titanomagnetite roast.

Effect of pulp density

The effect of pulp density on the V extraction from the titanomagnetite roast was studied at 30 m/m% and 65 m/m%, while the PSD and pH were kept constant at D_{100} of 1000 μm and 7.8, respectively. The results, depicted in Figure 3, show that the best V extraction was generally achieved at the highest pulp density of 65 m/m%. In other words, the best vanadium extraction was attained in the system with relatively higher solids' content. These results suggest that the extraction process limiting reagent in the leach slurry is the solids' content, as opposed to being typically the lixiviant concentration in the case of acid and basic leaching systems. Indeed, the water lixiviant offers (a relatively cheap and) an active vanadium leaching medium.

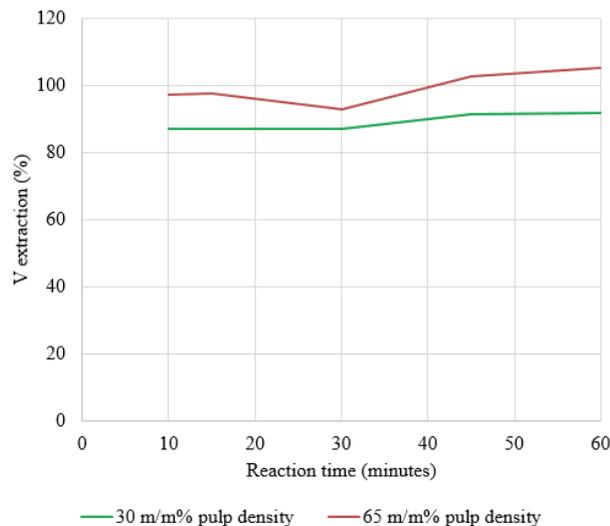


Figure 3. Effect of pulp density on the vanadium extraction from a titanomagnetite roast.

VANADIUM DEPARTMENT

Roast-Leach Process Flowsheet

To study vanadium department in the Mintek roast-leach process flowsheet, the bulk titanomagnetite roasted material was subjected to the best water leaching conditions as established above, followed by precipitation of vanadium as AMV and calcination to produce V_2O_5 . The conditions that were used in the processing of the bulk titanomagnetite roast (with $-1000 \mu\text{m}$ particle size) entailed leaching in a leaching tank with deionised water at 70°C for 60 min in a pulp density of 65 m/m% and at a pH of 7.8. The leach liquor was subjected to evaporation of water to concentrate the vanadium in the solution, followed by precipitation of V as AMV using 200% stoichiometric addition of $(\text{NH}_4)_2\text{SO}_4$ powder, and calcination of the AMV at 300°C overnight.

The roast-leach process flowsheet is shown in Figure 4. The cumulative vanadium department in the respective stages was calculated in relation to the starting V concentration in the original feed. The vanadium extraction at a specific operational stage and the cumulative vanadium extraction are shown in bold black font and bold red font in brackets, respectively. The material build-up on the kiln reactor walls, thus resulting in residual V at this stage, was overlooked in the quantification of the cumulative vanadium extraction efficiency because in a continuous large-scale process, the residual V is anticipated to approach and subsequently reach a minimum. As shown in the flowsheet, significant V losses occur during the leaching and AMV precipitation stages. The indicative cumulative vanadium extraction from the MML titanomagnetite to the final V_2O_5 product was 82%. The bulk chemical composition of the subsequent V_2O_5 product is shown in Table III. The high Cr content previously observed in the leaching optimisation does not seem to be problematic in the final product. The relatively high Al concentration in the V_2O_5 product will likely be less problematic as well provided the V_2O_5 is used to produce FeV by the established aluminothermic reduction.

Table III. Bulk chemical composition of the V_2O_5 product (mass %)

Al_2O_3	SiO_2	CaO	V_2O_5	Cr_2O_3
1.00	0.52	0.23	97.5	0.41

Electric Furnace Smelting Option

Vanadium department in the EHSV process was investigated for the recently developed and advantageous OSBF process coupled with the shaking ladle process. The vanadium department in this EHSV process flowsheet is shown in Figure 5. The bulk chemical composition of a typical EHSV vanadium slag is shown in Table IV. The vanadium extraction in the OSB furnace was 77% and that of the shaking ladle was 92% (Steinberg, 200; Steinberg et al., 2011). This implies that the cumulative vanadium extraction in the EHSV operations to produce the vanadium slag is 71%. The vanadium slag is processed by Vametco, Vanchem and Treibacher to produce vanadium products such as V_2O_5 (Rohrmann, 1985; Roskill, 2010). The vanadium slag is typically processed by the roast-leach process to produce V_2O_5 (Mefos, 2006; Rohrmann, 1985). Because data for the vanadium slag processing flowsheet are not readily available, data obtained from the bulk processing of the titanomagnetite by the roast-leach process at Mintek was adopted to obtain an indicative vanadium extraction. The cumulative vanadium extraction as V_2O_5 through the electric furnace smelting approach was thus 58%.

Table IV. Bulk chemical composition of a typical EHSV vanadium slag (mass %)

MgO	Al_2O_3	SiO_2	CaO	TiO_2	V_2O_5	Cr_2O_3	MnO	FeO
4.0	5.0	15.0	5.0	9.0	24.0	5.0	3.0	29.0
Mg	Al	Si	Ca	Ti	V	Cr	Mn	Fe
2.4	2.6	7.0	3.6	5.4	13.4	3.4	2.3	22.5

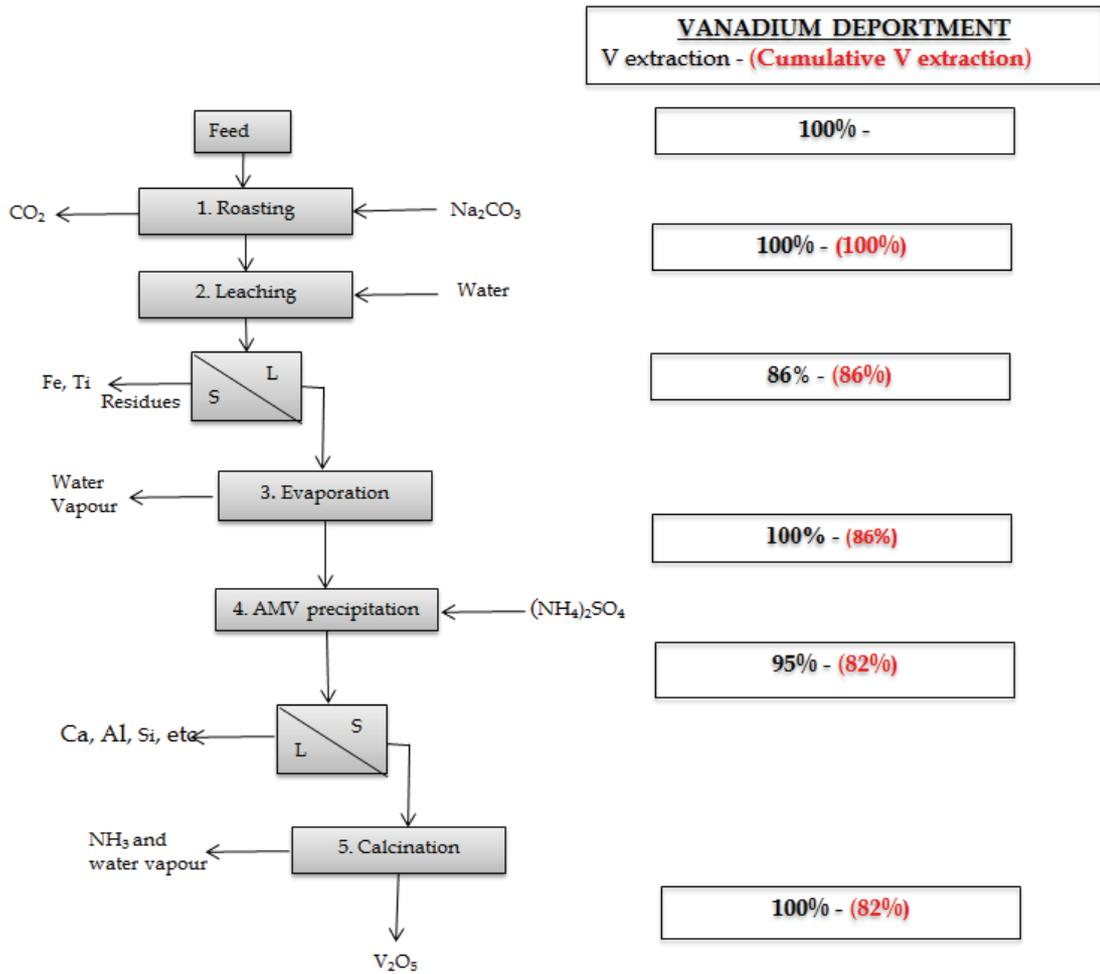


Figure 4. Vanadium department in the Mintek titanomagnetite roast-leach processing flowsheet.

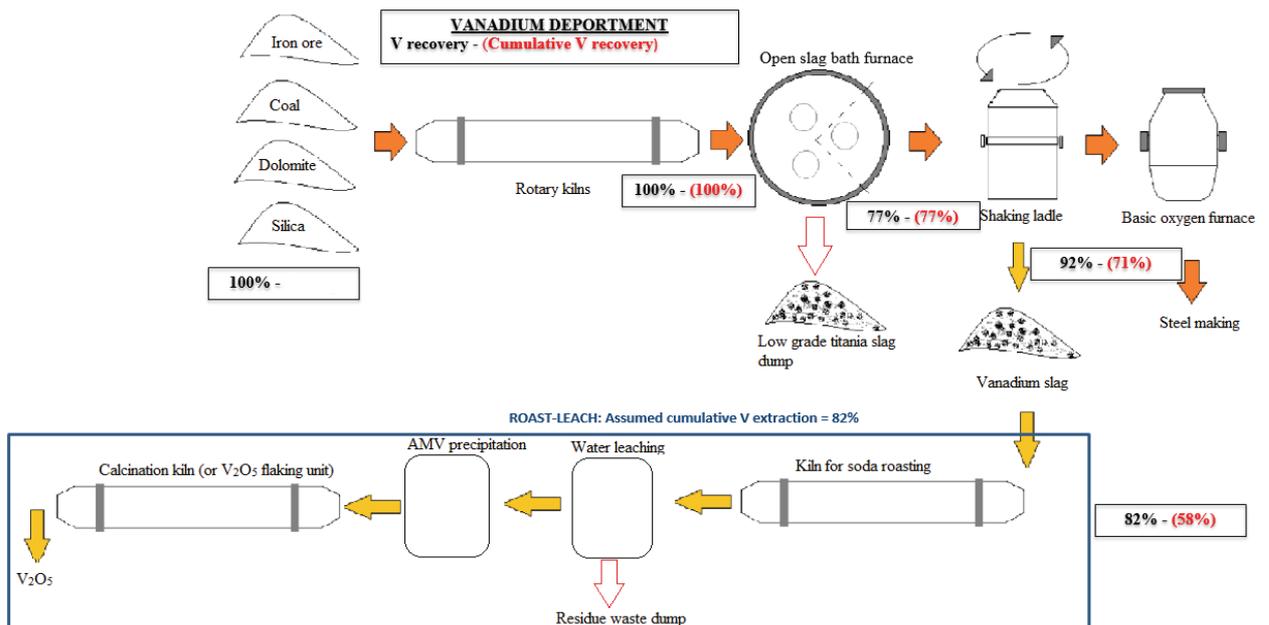


Figure 5. Vanadium department in the EHSV titanomagnetite OSBF smelting flowsheet.

CONCLUSIONS AND RECOMMENDATIONS

From this testwork, the cumulative vanadium extraction as V₂O₅ from titanomagnetite through the roast-leach vanadium primary production approach was 82%. The cumulative vanadium extraction from titanomagnetite through the OSBF and shaking ladle vanadium co-production process at EHSV was 71% up to the stage of the vanadium slag. The cumulative vanadium extraction for this process would be 58% at the final stage of V₂O₅ extraction from the vanadium slag by the roast-leach process, if the data from the Mintek roast-leach process of titanomagnetite are adopted, because of the unavailability of commercial vanadium slag processing data in the open literature. In terms of cumulative vanadium extraction, the roast-leach process appears to be superior if compared with electric furnace smelting on this basis. However, it should be noted that electric furnace smelting is a co-production process in which iron is another valuable product that is used in steel making and vanadium is not the primary product.

A techno-economic evaluation of the roast-leach vanadium primary production and electric furnace smelting vanadium co-production processes needs to be completed to fully understand the most beneficial processing route, taking into account all the products. Further testwork to improve the scope of products from the processing of titanomagnetite by the two processes, *i.e.*, investigate the iron and titanium production in the roast-leach process and titanium in the electric furnace smelting approach, would further enhance the quality of the interpretation of the data.

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REFERENCES

- Bushveld Minerals Ltd. (2016). Vanadium product – product summary. <http://www.bushveldminerals.com/vanadium.aspx> [accessed 18 March 2016].
- Evraz NTMK (2016). http://www.evraz.com/products/business/constructional/nizhny_tagil/ [accessed 18 March 2016].
- Fischer, R.P. (1975). Vanadium resources of titaniferous magnetite deposits. *Geological Survey Professional Paper*, 926-B. United States Government Printing Office, Washington. pp. B1–B10.
- Goldberg, I., Hammerbeck, E.C.I., Labuschagne, L.S. and Rossouw, C. (1992). International strategic minerals inventory summary report – Vanadium. *U.S. Geological Survey Circular* 930-K. United States Government Printing Office, Washington. pp. 1–41.
- Gupta, C.K. and Krishnamurthy, N. (1992). *Extractive Metallurgy of Vanadium*. Elsevier Science Publishers. pp. 151–320.
- Jones, R.T. (1992). The availability and marketing of vanadium. *Proceedings of the 6th International Ferroalloys Congress (INFACON 6)*. Vol. 1. South African Institute of Mining and Metallurgy, Johannesburg. pp. 37–41.
- Kelly, B.F. (1993). Ironmaking at BHP New Zealand Steel Limited, Glenbrook, New Zealand. *The Sir Maurice Mawby Memorial Volume*. Vol. 2. Australasian Institute of Mining and Metallurgy, Victoria, Australia. pp. 348–353.
- Largo Resources Ltd (2016). Production process and profile. <http://www.largoresources.com/English/projects/maracas/default.aspx> [accessed 18 March 2016].
- Mefos, G.Y. (2006). Recovery of vanadium from LD slag. A state of the art report, Part 1 – Facts and metallurgy of vanadium. Document D 816 (MEF06001). Swedish Steel Producers Association. Stalkretsløppet, Jernkontorets Forskning. pp. 12–24.
- Moskalyk, R.R. and Alfantazi, A.M. (2003). Processing of vanadium: a review. *Minerals Engineering*, 16, 793–805.
- Rohrman, B. (1985). Vanadium in South Africa. *Journal of the South African Institute of Mining and Metallurgy*, 85 (2), 141–150.

- Roskill (2010). *Vanadium: Global Industry Market and Outlook*. 12th Edition. Roskill Information Services Ltd., London. pp. 7-133.
- Song, W., Li, H., Zhu, F., Li, K. and Zheng, Q. (2014). Extraction of vanadium from molten vanadium bearing slag by oxidation with pure oxygen in the presence of CaO. *Transactions of the Nonferrous Metals Society of China*, 24, 2687-2694.
- Steele, T.W. and Wilson, A. (1966). A study of the salt-roasting process for the extraction of vanadium from iron ore. *Proceedings of the 36th Congress International De Chimie Industrielle*, Brussels, Belgium. pp. 1-6.
- Steinberg, W.S. (2008). Development of a control strategy for the open slag bath furnaces at Highveld Steel and Vanadium Corporation Ltd. *Master's Dissertation*, University of Pretoria. <http://repository.up.ac.za/bitstream/handle/2263/26090/dissertation.pdf?sequence=1> [accessed 2 June 2016].
- Steinberg, W.S., Geysler, W. and Nell, J. (2011). The history and development of the pyrometallurgical processes at Evraz Highveld Steel and Vanadium. *Journal of the Southern African Institute of Mining and Metallurgy*, 111 (10), 63-76.
- Taylor, P.R., Shuey, S.A., Vidal, E.E. and Gomez, J.C. (2006). Extractive metallurgy of vanadium-containing titaniferous magnetite ores: a review. *Minerals and Metallurgical Processing*, 23 (2), 80-86.
- Zhang, L., Zhang, L.N., Wang, M.Y., Li, G.Q. and Sui, Z.T. (2007). Recovery of titanium compounds from molten Ti-bearing blast furnace slag under the dynamic oxidation condition. *Minerals Engineering*, 20, 684-693.
- Zhao, L., Wang, L., Qi, T., Chen, D., Zhao, H. and Liu, Y. (2014). A novel method to extract iron, titanium, vanadium, and chromium from high-chromium vanadium-titanomagnetite concentrates. *Hydrometallurgy*, 149, 106-109.

