The Effect of Talc Content on Flotation Mass Pull of Oxidised PGMs Ore. A Case Study of the Great Dyke of Zimbabwe Altered PGMs Ore

Nancy. R. Mabwe[1], Willie Nheta[2], Vusumuzi Sibanda[3]

***Abstract*—**Mineralogical composition of an ore plays a critical role in mineral beneficiation. In this study, flotation test work was done on oxidised platinum group metals (PGMs) ore after milling to achieve a grind of 60% passing 75 microns. XRF and ICP- OES analysis of the ore showed a high percentage of magnesium, that is, 18.8% and 25.9% respectively. The XRD and SEM analysis also confirmed that the oxidised PGM ore had a high proportion of talc. Rougher flotation produced a concentrate with a mass pull of 30%. This was an indication of the presence of unintentionally recovered talc. Subsequent cleaning of the rougher concentrate resulted in a significant decrease in the mass pull from 30% to 15%. These results show that there is a strong link between talc content and flotation performance. The effect of comminution on the liberation of talc needs to be critically analysed to improve flotation performance.

***Keywords*—**Flotation, Mass Pull, Oxidised PGMs ore, Talc.

I. Introduction

Flotation can best be described as a complex physico-chemical mineral processing method that depends on the surface chemistry of the mineral particles [1]. Full liberation of valuable particles from the waste in a single step is not possible; therefore, multi-step processes are employed to attain better liberation. The effect of mineralogy on the development and optimisation of the flotation process flowsheets is crucial since ores are composed of aggregates of minerals with different textures and varying degrees of non-homogeneity [2]. A previous study on Upper Group 2 chromitite reef samples showed that mineralogy, physical properties and milling behaviour play a crucial role in sustainable ore beneficiation [3]. This research provokes continued research in order to establish the influence of mineralogy on the comminution and flotation behaviour of the Zimbabwean oxidised ore sample.

The rapid depletion of high-grade ores, which is the case in the PGMs industry in Zimbabwe, where average head grades of the ores have decreased significantly [4], has led to increased research efforts on alternative processing techniques. The Great Dyke in Zimbabwe, which is host to the second largest platinum group metals (PGMs) in the world, is a geological feature that is linear and runs north-south through the middle of the country and passes on the western side of Harare. It runs for an approximate length of 550 km and is made up of hills as high as 460m. Sulphide dissemination of the Main Sulphide Zone (MSZ) hosts economic concentrations of the platinum group elements (PGEs) and mining of this zone is mainly at Mimosa, Ngezi and Unki mines. There also exists in the MSZ zone oxidised near-surface ores that have great potential with a large resource base of about 250 Mt. Unfortunately, processing of the oxide ores by conventional techniques has not yielded positive results with recoveries less than 50% attained. This is attributed to the polymodal distribution of the PGEs compared to the bimodal distribution in pristine, MSZ. The oxidised ores originate from sulphidic base metal sulphides that have disintegrated, partially releasing the PGEs and base metals and substituting these minerals with hydroxides or iron oxides.

Since the processing challenges are attributed to the complex nature of the oxidised MSZ ores, there is a need for novel processing techniques which depend on proper knowledge of the mineralogical composition of the ores [5]. Table I illustrates the difference in mineralogy between the pristine main sulphide zone ores and the weathered ores from the main sulphide zone of the Great Dyke of Zimbabwe. The table shows the altered behaviour of the oxidised PGMs ore as they are composed of a large amount of sperrylite (PtAs2) and cooperite in combination with nickel and palladium sulphides having the formula ((Pt, Pd, Ni)S) [6].

Nancy. R. Mabwe[1] is with the Mineral Processing and Technology Research Centre, Department of Metallurgy, University of Johannesburg, Doornfontein, South Africa

Willie Nheta[2] is with the Mineral Processing and Technology Research Centre, Department of Metallurgy, University of Johannesburg, Doornfontein, South Africa

Vusumuzi Sibanda[3] is with the School of Chemical & Metallurgical Engineering, University of the Witwatersrand, Johannesburg, South Africa

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TABLE I
MINERALOGY CHARACTERISATION OF PRISTINE MSZ ORE AND ALTERED PGMS ORE FROM THE GREAT DYKE MAIN SULPHIDE ZONE [6]

Ore type	Discrete PGM grains (n)	(Pt,Pd)(Bi,Te)*	$PtAs_2$	(Pt,Pd,Ni)S	Pt and Pt-Fe alloys	PGE-sulph-arsenides	Others
Pristine MSZ sulphide	801	50.1	19.0	8.5	2.4	11.9	8.7
Oxidized MSZ sulphide	1293	11.4	57.2	28.3	3.1	0	0

II. MATERIALS AND METHODS

A. Materials

The oxidised PGM ore sample obtained from a Platinum mine in the Great Dyke of Zimbabwe was utilised in this study.

B. Sample Preparation

The sample was first dried in air and then crushed to a 1 mm size. A blender was used for homogenisation, then the sample was split using a rotary splitter into 1 kg aliquots. 1 kg samples were wet milled in a rod mill at a solid-to-liquid ratio of 1: 1 for 10, 30 and 60 minutes. The milling curve shown in Fig. 1 showed that the optimum milling time was 38 minutes in order to achieve 60% passing 75 microns, the target fineness for flotation.

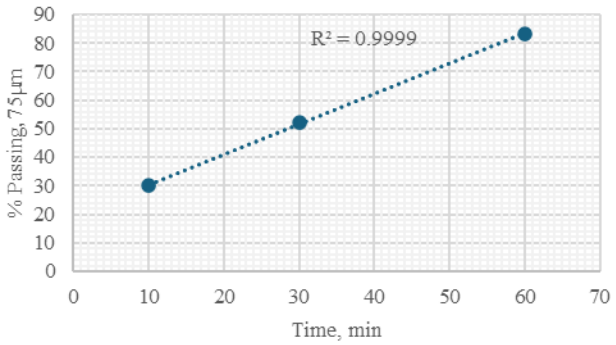

Fig. 1 Milling curve to achieve flotation target fineness

The feed sample was characterised using powder X-ray diffraction (XRD) and scanning electron microscopy (SEM).

Inductively coupled plasma optical emission spectrometry (ICP-OES) and X-ray fluorescence (XRF) analysis techniques were employed for 4E (platinum, palladium, rhodium and gold) head-grade analysis.

C. Flotation Tests

The process water used for all the flotation test works was carefully prepared to resemble process water that is typically employed at PGMs concentrators. The water was prepared by adding ionic salts to distilled water. The preparation formula is presented in Table II[7].

TABLE II
THE FORMULA USED TO PREPARE PLANT WATER [7]

Salt name	Quantity (g)
$MgSO_4 \cdot 7H_2O$	12.30
$Mg(NO_3)_2 \cdot 6H_2O$	2.14
$Ca(NO_3)_2 \cdot 4H_2O$	4.72
$CaCl_2 \cdot 2H_2O$	2.94
NaCl	4.12
Na_2CO_3	0.60

A hydroxamate oxide collector was employed, as other researchers have proved its effectiveness in improving the flotation recovery of PGMs from oxidised PGMs ore, with recoveries as high as 74.7% 3E achieved. This is attributed to the ability of hydroxamates to form compounds with the base metals, in particular iron [6]. The flotation reagent suite employed for the test works is presented in Table III.

TABLE III
FLOTATION REAGENT SUITE USED FOR THE FLOTATION EXPERIMENTS

Reagent name	Purpose	Reagent dosage
Carboxymethyl cellulose (CMC)	Depressant	750 g/t
Sodium Isobutyl Xanthate (SIBX)	Primary collector	150 g/t
AM 28 (Hydroxamate collector)	Oxide co- collector	300 g/t
Dowfroth	Frother	40 g/t
Copper sulphate	Activator	50 g/t

Flotation test works were done for 7 minutes at an impeller speed of 1200rpm. Mass pull from the first run was calculated using the formula:

$$Mass\ pull = \frac{Concentrate\ (g)}{Feed\ (g)} * 100 \quad (1)$$

The concentrate (rougher concentrate) obtained from the initial flotation run was cleaned using the same reagent suite to get the cleaner concentrate and the mass pull was also calculated.

III. RESULTS AND DISCUSSION

A. Head Assay

The 4Es feed grade was 2.3 g/t obtained using inductively coupled plasma optical emission spectrometry (ICP-OES).

PGMs concentrations in the oxidised PGMs run-of-mine were measured using the XRF and ICP-EOS analysis techniques. Tables IV and V present the analysis results.

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TABLE IV
XRF ANALYSIS OF THE OXIDISED PGMS ORE

No.	Component	Result	Unit	Element line	Intensity (cps/μA)
1	Mg	18.8	mass%	L:Mg-Kα	4.33778
2	Na	ND	mass%		
3	Al	3.75	mass%	L:Al-Kα	4.32198
4	Si	40.8	mass%	L:Si-Kα	147.74578
5	S	0.136	mass%	L:S-Kα	1.72218
6	P	ND	mass%		
7	K	0.199	mass%	M:K-Kα	0.03570
8	Ca	6.50	mass%	M:Ca-Kα	2.13255
9	Sc	(0.0283)	mass%	M:Sc-Kα	0.01325
10	Ti	0.569	mass%	M:Ti-Kα	0.42817
11	V	0.0440	mass%	M:V-Kα	0.05330
12	Cr	0.160	mass%	M:Cr-Kα	1.59344
13	Mn	0.474	mass%	M:Mn-Kα	1.10304
14	Fe	26.3	mass%	M:Fe-Kα	94.50789
15	Co	0.0944	mass%	M:Co-Kα	0.49356
16	Ni	0.869	mass%	M:Ni-Kα	3.69052
17	Cu	0.488	mass%	M:Cu-Kα	2.74298
18	Zn	0.0280	mass%	M:Zn-Kα	0.22449
19	Sb	0.052	mass%	H:Sb-Kα	0.07462
20	Sn	0.0558	mass%	H:Sn-Kα	0.74101
21	Cd	ND	mass%		
22	Ag	0.0019	mass%	H:Ag-Kα	0.02225
23	Pd	ND	mass%		
24	Rh	ND	mass%		
25	Ru	ND	mass%		
26	Nb	ND	mass%		
27	Mo	0.0069	mass%	H:Zr-Kα	0.04205
28	Zr	ND	mass%		
29	Pt	ND	mass%		
30	Pb	0.0033	mass%	M:Pb-Lα	0.02770
31	Tl	ND	mass%		
32	Au	ND	mass%		
33	W	ND	mass%		
34	Ir	ND	mass%		
35	Ba	0.0088	mass%	H:Ba-Kα	0.07618
36	Rb	0.0011	mass%	M:Rb-Kα	0.04052
37	Cs	ND	mass%	H:Cs-Kα	0.00001
38	Ta	ND	mass%	M:Ta-Lα	0.00582
39	W	ND	mass%	M:W-Lα	0.00002
40	Os	ND	mass%	M:Os-Lα	0.00043
41	Tc	ND	mass%		
42	Sr	0.0077	mass%	M:Sr-Kα	0.31426

TABLE V
ICP- EOS ANALYSIS OF THE OXIDISED PGMS ORE

Al_2O_3 (%)	CaO (%)	Co (%)	Cr_2O_3 (%)	Cu (%)	Fe_2O_3 (%)	MgO (%)	MnO (%)	Ni (%)	PbO (%)	SiO_2 (%)	TiO_2 (%)	V_2O_5 (%)	Zn (%)	Au (ppm)	Pd (ppm)	Pt (ppm)
3.72	4.33	<0.05	0.409	0.13	12.3	25.9	0.206	0.21	<0.06	53.3	0.184	<0.09	<0.05	0.205	0.92	1.425

The low amounts of nickel and copper from both the XRF and ICP- EOS analysis showed the oxidised PGMs ore had reduced amounts of the base metal sulphide minerals. Also, the reduced amount of sulphur as measured by the XRF significantly contributes to reduced recoveries during flotation when the sulphide reagent suite is employed. Sulphidic base metal minerals such as pyrrhotite, chalcopyrite and pentlandite are targeted due to the close association of the minerals with PGMs [8].

Flotation test works done on the ore after milling to achieve an optimum grind of 60% passing 75 microns resulted in a rougher concentrate with a high mass pull of 30% and PGMs grade of 16,8 g/t. The findings are in good agreement with those of [9], who confirmed that a high mass pull is due to hydration alteration of the ore and oxidation of the sulphide matrix. XRF and ICP- OES analysis of the ore showed a high percentage of magnesium, that is, 18.8% and 25.9% respectively. This is an indication of the presence of talc, whose unintentional recovery has an adverse effect on the flotation process [10]. Researchers [2] attributed high mass pull to the presence of orthopyroxene, which also floats naturally. Subsequent cleaning of the rougher concentrate resulted in a significant decrease of the mass pull from 30% to 15% with a PGMS grade of 50,96 g/t, an observation that needs to be critically analysed both theoretically and experimentally. These results show that there is a strong link between the mineralogy of the ore and ultimate flotation performance.

B. Bulk Mineralogical Analysis

The feed sample mineralogy was assessed to determine the main crystalline phases. The results are presented in Figs 2

and 3.

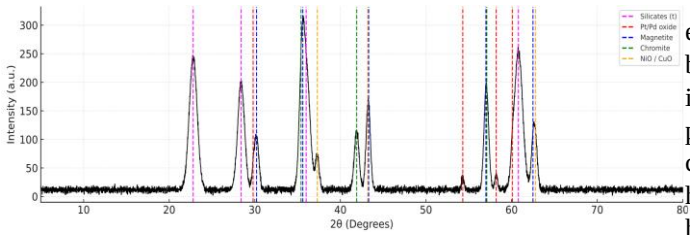

Fig. 2: XRD Diffractogram of oxidised PGMs ore

The strong silicate band (talc/pyroxenes) signature in the ore, as shown in Fig. 2, is clear evidence of talc's abundance and hydration alteration of the ore. Talc reports to the flotation concentrate since it is naturally hydrophobic and has a platy morphology, thus reducing flotation efficiency**.** Fine talc increases froth stability and water recovery. This results in gangue entrainment, diluting PGMs grades. Talc adsorbs collectors, reducing their selectivity and depressing true PGMs recovery. The presence of talc also raises pulp and froth viscosity, impairing bubble–particle collision/attachment kinetics and making froth drainage poorer. This consequently results in high mass pull as more gangue reports to the concentrate, increasing overall solids recovery.

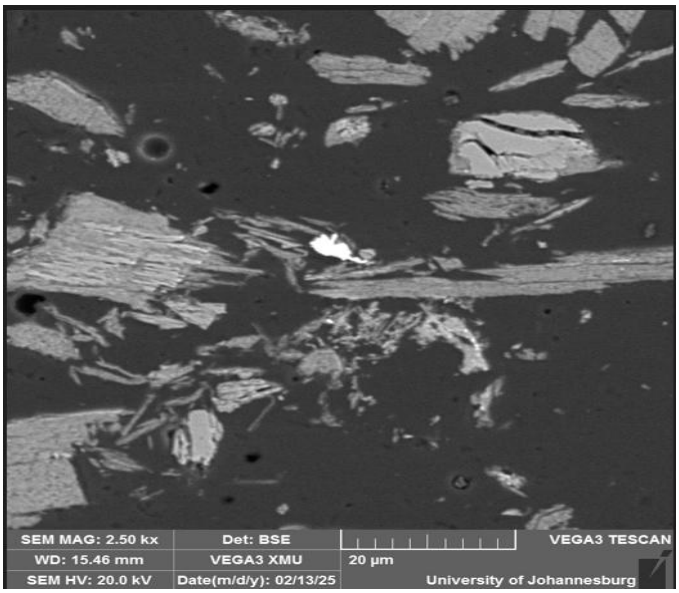

Fig 3: SEM image of oxidised PGMs ore

Fig. 3 also shows that talc is the main gangue phase in the oxidised PGMs ore. The silicate gangue matrix (talc, pyroxene) appears as pale grey/white zones that dominate the fine fraction. The darker spots or grains represent the disseminated oxides and the secondary alteration products (iron oxides/ hydroxides) are also represented as dark stains. The negative effects of talc on the flotation process can be minimised by prefloating the talc ahead of roughers, adding depressants, using low frother dosages and optimising pH and ionic strength to reduce slime coating and improve selectivity.

IV. Conclusion

The body of work presented in this study shows enough evidence that talc content has an effect on the flotation behaviour of the oxidised PGMs ore under study. Talc, which is naturally floating, is therefore a probable reason for the poor flotation performance, among other factors. High talc content increases froth stability and entrainment, resulting in higher mass recovery but a lower concentrate grade. This highlights the need for targeted talc management strategies—such as pre-flotation, selective depressants, and tighter froth control—to lower mass pull, improve concentrate grade, and reduce downstream processing load. Mass pull can also be controlled by monitoring operating conditions and reagent regime.

Cleaning of the rougher concentrate proved useful in reducing the high mass pull. Continued research is encouraged to determine the comminution behaviour of the ore in relation to abundant minerals in the ore for sustainable processing of the oxidised ores.

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