PAPER 10

Formulation and Plant Trial of a Mixed Collector Suite For Eland Platinum

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ABSTRACT

The role of mixed collectors in enhancing the metallurgical performance of sulphide flotation processes was reviewed in 2008, and later published as an integrated theory by Lotter and Bradshaw, 2009. The theory proposed that xanthates, dithiophosphates and dithiocarbamates could be synergistically combined in an optimal formula to deliver improved grade and recovery. The approach uses a five-step structure, viz.

1. A mineralogical study of the ore to be treated, identifying and quantifying the economic minerals (with hosting and association) and gangue species present.
2. Selection of candidate collectors from this mineralogical information and from the known pulp conditions of that operation (e.g. Eh, pH).
3. Design and execution of a factorial experiment to estimate and test the main effects and interactions, and to model the response surface for an optimum formulation [Box et al., 1978].
4. Use of a High-Confidence Flotation Test platform in the factorial to reduce overall errors and allow for clearer observations [Lotter, 1995; Lotter and Fragomeni, 2010].
5. Statistical design of a plant scale trial to measure and validate the gains [Napier-Munn, 1995; Lotter et al., 2009].

Eland Platinum, which was commissioned in November 2007, appointed a team to apply this theory to formulate an optimized mixed collector suite for use in the operation. The West Pit ore (UG2 - Normal ore) was sampled at site and brought to Xstrata Process Support, Sudbury, Canada, for mineralogical and metallurgical testing in 2008. Microprobe and QEMSCAN measurement characterized the type, grain size and textural association of the discrete Platinum Group Minerals (PGM) present. From the mineralogical data, it was clear that a mixture of dithiocarbamate and two xanthates would be a likely reagent suite selection. A factorially designed set of High-Confidence Flotation Tests studying these was performed, concluding that an optimal suite using certain amounts of these collectors was a viable formulation that would advance the grade and recovery performance at Eland above the standard PIBX by approximately 2.13% 4E PGE. The plant trial, which used a form of the replicated block design after Napier-Munn, 1995, and Lotter et al., 2009, exceeded this estimate with an overall PGE recovery gain of 2.48% (absolute basis) and a concentrate grade gain of 26.4 g/t PGE (absolute basis), as well as reducing chrome in concentrate from 2.04 to 1.62% (absolute basis). These gains proved to be statistically significant using the analysis of variance method, and have validated the theory.

INTRODUCTION

Post-Commissioning History

The Eland concentrator, located some 20 km east of Brits, Northwest Province, South Africa, was commissioned in November 2007, and treats a range of UG2 ore types. The ore resource and concentrator had been developed by the junior miner “Eland Platinum”, who had bought the mining rights to this property from Amplats. Xstrata plc subsequently bought Eland Platinum in

November 2007. The prospectus stated that life-of-mine PGE recoveries of around 75% were expected in a saleable concentrate grading 170 g/t PGE and less than 2% Cr_2O_3. Designed milling capacity was 250 Ktpm.

The flowsheet follows a generic MF2 (mill, float, mill, float) arrangement typical of most Bushveld UG2 circuits, with the primary and secondary grinds set at 45 and 80% mass passing 75 microns. Both grinding stages are in open circuit, so as to limit overgrinding of chromite and thus to limit unwanted entrainment of this species in the concentrate. Separate cleaning of the first and second rougher flotation arisings releases two independent increments to final concentrate. At commissioning, the selected collector was sodium normal propyl xanthate (SNPX). During 2008, this was changed to potassium isobutyl xanthate (PIBX) for improved recovery. Total collector dose is approximately 300 g/t milled. From commissioning in November 2007 to July 2009, Eland had used a single collector system.

Process Mineralogy

Some of the earliest acknowledgements of the impact of mineralogy on milling and flotation were published by Gaudin, 1939. The beginnings of modern Process Mineralogy start with Henley, 1983, at the time when QEM*SEM was being developed in Australia by the Commonwealth Scientific and Industrial Research Organisation (CSIRO). This hybrid discipline was developed on the foundations of sampling, quantitative mineralogy, and mineral processing. It soon developed synergies between these disciplines that formulated mineral processing implications prior to any mineral processing testwork being performed [Lotter et al., 2002]. Numerous examples of equivalent development and applications have since been published, for example Lotter et al., 2003; Fragomeni et al., 2005; Charland et al., 2006; Dai et al., 2008; McKay et al., 2007; Triffett et al., 2008. The overall effect is to develop predictive properties from the mineralogy, and to optimize the milling and flotation flowsheet strategies from this information to a better level than would be delivered by traditional empirical testing. By focusing on an improved concentrator flowsheet, we are addressing recovery improvement at the source of the largest paymetal loss by which saleable metals are produced from sulphide ores [Cramer, 2001]. In the case of the Eland post-commissioning optimization project, the Process Mineralogy group of Xstrata Process Support (XPS) was appointed to develop an understanding of the main Eland orebody and its processing implications.

Mixed Collectors

If sustainable paymetal recovery gains can be made and proven from a reagent suite change, these gains represent the highest rate of return for any project in a concentrator, since they do not require capital funds in order to be implemented. Mixed collectors for sulphide flotation are well-known for their synergy and extra flotation performance, but their practice is not new. One of the earlier researchers in this field published his work in 1958 out of Moscow, concluding that when a weak collector was added to a strong one, recovery gains of 2-5% were achieved, often with a lower overall dose of collector [Glembotskii, 1958]. Lotter and Bradshaw, 2009, reviewed the subject and concluded that the following performance gains were independently reported from optimized mixed collector suites:

1. Improvement in rate of flotation [Plaskin et al, 1960; Adkins and Pearse, 1992].
2. Improvement in coarse particle recovery [Plaskin et al, 1954].
3. Reduction in dosage requirement [Plaskin et al, 1960, Bradshaw, 1997].
4. Best results were obtained at an optimum ratio of constituents [Mingione,1984; Critchley and Riaz, 1991;Valdiviezo and Oliveira, 1993, Bradshaw, 1997, Deng et al., 2010].

This review was deliberately confined to the use of xanthates, dithiocarbamates and dithiophosphates, since these demonstrate synergies, are in common use and are supplied in a common price range. It was also concluded that there was an appreciation of the value of a mixed collector system, but the collector suite that had been configured had not been fully optimized. This shortfall may be dealt with by the following steps:

1. A mineralogical study of the ore to be treated, identifying and quantifying the economic minerals (with hosting and association) and gangue species present.
2. Selection of candidate collectors from this mineralogical information and from the known pulp conditions of that operation (e.g. Eh, pH).
3. Design and execution of a factorial experiment to estimate and test the main effects and interactions, and to model the response surface for an optimum formulation [Box et al., 1978]. It is key to recognize that the interactions drive the mixed collector synergy [Bradshaw, 1997], and that these are best identified and quantified in a factorially designed experimental arrangement [Bradshaw et al., 1992].
4. Use of a High-Confidence Flotation Test platform in the factorial to reduce overall errors and allow for clearer observations [Lotter, 1995; Lotter and Fragomeni, 2010].
5. Statistical design of a plant scale trial to measure and validate the gains [Napier-Munn, 1995; Lotter et al., 2009].

Thus, a mixed collector project appeared suitable as a starting point in Eland's optimization programme, provided that the above approach was followed.

High-Confidence Flotation Testing

High-Confidence Flotation Testing (HCFT) was developed in Amplats at their Divisional Metallurgical Laboratory (DML), Rustenburg, South Africa, between 1987 and 1995. This was done in two stages: the first, a heuristic expert system (1987-1992), and the second, a more fundamental study (1993-1995), so as to advance the quality control system and improve reproducibility and scalability of laboratory test data for the Platinum Group Element (PGE) set [Lotter and Munro, 1994; Lotter, 1995]. It was subsequently adapted to base metal applications in Canada [Lotter and Fragomeni, 2010] with several refinements that improved flexibility, but retained the 95% confidence level and reproducibility. The principles are based on a sound sampling arrangement to ensure a representative sample of ore to be tested, a rigorous crushing, blending and subsampling system to produce comparable replicate test charges of ore, and a system of replicate flotation tests with quantitative limits set on metal balances and concentrate masses. Averaging of the accepted replicates invokes the powerful effects of the Central Limit Theorem, reducing random errors and placing the residual errors in a normal distribution. With this arrangement it is possible to detect small differences in flotation performance, ideal for a factorially designed test programme in which main effects and interactions need to be more clearly observed.

Factorial Test Design

There are abundant references available on the subject of factorial test design. Proper use of this technique typically produces optimum results in a shorter time than traditional one-at-a-time testwork. More importantly, factorial design identifies and quantifies the interactions between the variables, which the one-at-a-time approach does not [Box et al., 1978]. Furthermore, the main effects and interactions are estimated from the differences between plane averages of the cube, rather than from individual test points. In the context of mixed collectors, it is the interactions that dominate the synergy, thus factorially designed flotation testing for mixed collectors is an obvious choice. In the Eland study, a three-variable, two-level factorial with a mid-cube, conducted in duplicate with a replicated mid-cube, was chosen. For such an arrangement, this means that the main effects and interactions are estimated from 16 individual observations in each case. This improves the robustness of the estimated effects.

Plant Scale Trials

Small but economically significant performance gains across a concentrator operation present the problem of measurement and proof. This problem is generic for suphide ore concentrators across the world. Historically the mineral processing industry has left the small gains to the realm of the unprovable, and has preferred to pursue larger performance gains because of their easier proof. The cost of this approach has left a much larger total business opportunity on the table, since there are many more small performance gain opportunities than there are large ones. The leverage of statistics into mineral processing re-opens the discussion of these small recovery gains because the test designs can deal with the central problem, viz. small performance changes within noisy data and associated with autocorrelation.

Earlier publications have clearly set the example of how this may be done, using the replicated blocking method [Napier-Munn, 1995 at Bouganville Copper Limited, and Lotter et al., 2009 at the Xstrata Nickel Raglan concentrator, for example]. It is a generic sampling rule that samples must be taken on an independent basis. That is, if a suite of samples are taken in a time series, there must be no correlation between their measurements. Otherwise, the sampling outcomes will be biased [Box et. al., 1978; Gy, 1978]. It is the preferred arrangement that a plant trial works outside autocorrelation by 'blocking' the effects of autocorrelation by appropriate test design. This arrangement provides for independent sampling, an important statistical premise.

Specific Objectives

The specific objectives of this project were:

1. To identify the appropriate candidate collectors for the Eland Platinum operations
2. To characterize these in laboratory scale flotation testwork, and formulate an optimal mixture for plant trial

3. To design and execute a suitable plant trial to demonstrate the metallurgical benefits
4. To validate or reject the theory of mixed collectors as set out by Lotter and Bradshaw, 2009.

EXPERIMENTAL PROCEDURE

Sampling and Mineralogy of West (Normal) Ore

Approximately 600 kg of run-of-mine Eland West Pit stockpile ore was manually sampled in May 2008 and dispatched to XPS. The bulk sample was crushed and blended, then spin-riffled into 2 kg replicate test charges. Random test lots were taken for fire-assaying to populate the External Reference Distribution. Polished sections of this crushed ore were also prepared and studied by EPMA microprobe and QEMSCAN.

Selection of Candidate Collectors and Design of Factorial

Three collectors were chosen for the formulation of the mixed collector to be put on trial at Eland. Based on the mineralogy, these were xanthate 1, xanthate 2, and a dithiocarbamate (DTC) [Fuerstenau, 1978; Bradshaw, 1997, Lotter and Bradshaw, 2009]. The alkaline natural pH of the Eland milled ore slurry ($8 < pH < 9$) was noted. Copper sulphate was completely avoided because of its known tendency to activate silicate gangue flotation [Lotter et al., 2008]. Table 1 sets out the arrangement of the three-variable, two-level factorial. The reagent dosages are shown as grammes per tonne milled. All reagents were taken at 100% nominal activity, and were dosed in a split arrangement between first and second flotation stages. Each test point was conducted in triplicate.

Table 1: Factorial format of test design

Test No.	Xanthate 1, g/t		Xanthate 2, g/t		Dithiocarbamate, g/t	
1	-	60	-	60	-	60
2	+	90	-	60	-	60
3	-	60	+	90	-	60
4	+	90	+	90	-	60
5	-	60	-	60	+	90
6	+	90	-	60	+	90
7	-	60	+	90	+	90
8	+	90	+	90	+	90
MC	M	75	M	75	M	M

Flotation Testing

A two-stage grinding and flotation laboratory procedure (MF2) was developed so as to mimic the Eland operations rougher and scavenger flowsheet. The primary and secondary grinds were set at 45 and 80% mass passing 75 microns to produce a total of three rougher concentrates. The timing of the concentrates was C1: 0-3 minutes; C2: 3-15 minutes; and C3: (after secondary grinding): 15-30 minutes. This arrangement is shown in Figure 1.

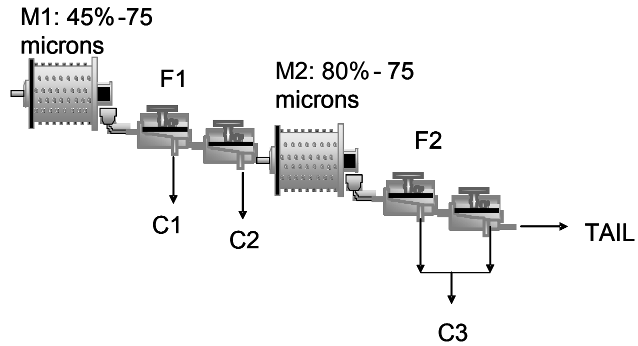

Figure 1: MF2 layout for flotation tests

The collectors were dosed in a split arrangement between the first and second flotation stages to a total of 240 g/t in the baseline test. Depressant Sendep 369 was dosed to a constant total of 30 g/t in two equal stages. Frother Senfroth 200XP was added at two stages. Replicate test ore charges of 2.0 kg were used.

Design of Plant Trial

The Eland operation treats at least five ore types in a campaign format from an opencast mine with some overlap of ore types as the ore supply changes from one stockpile to another. The trial focused on the Normal (West Pit) ore, since this was the ore type sampled and characterized at XPS. A form of replicated block design was used, in which standard SIBX and trial Exp 820 were dosed in an "on-off" format with data blocks of variable length as dictated by the supply of the Normal ore. The first two days of operation after the changeover were disqualified from being used as true data since:

1. The collector reagent inventories in the plant had to be purged;
2. It was necessary to break the autocorrelation that exists in short-term rougher float feed PGE grades.

Use is made of Analysis of Variance (ANOVA) as published by Box et al., 1978, with appropriate modifications as indicated by the specific case details at the Eland plant trial. The basis of this approach is as follows:

1. The data are gathered by ore type and by collector type. The changeover day is disqualified in all cases for the purposes of measurement. In this paper, we will discuss the process response to Normal Ore (West Pit).
2. Within this ore type, the data for each collector type are analysed with ANOVA to determine if the separate campaigns may be pooled.
3. Using accepted pooled data within this ore type and varying by collector type, ANOVA is used to determine whether the change of collector type produced a significantly different metallurgical response.

Having established significant differences in concentrate and tailings grades from ANOVA, the tailing grade is modelled from feed and concentrate grade, collector type and grind. A coefficient is estimated for each variable. Adjustment is made for feed and concentrate grades and grind, and the tailing grades are modeled for each collector type. Data were collected from August to October 2009.

RESULTS

West (Normal) Ore Sample

Geology and Mineralogy

A site visit was conducted in May of 2008 by XPS personnel in order to initiate a sampling and testwork program for Xstrata Alloys at the Eland Pt operation, Brits, South Africa. The mineralogical programme is aimed at examining mineralogical variability in the UG2 between West (Normal Facies) and East (Split Facies). The following observations and conclusions result from the mineralogical analysis of the West (Normal) ore.

Chromite, plagioclase, orthopyroxene, and chlorite dominate the bulk modal mineralogy in relative order of abundance. Base metal sulphides as well as rare alloy and metallic phases are present in trace amounts in all composites ranging from 0.1% to 0.4% of the bulk modal mineralogy. Base metal sulphide average grain sizes are very fine grained and range from 10 µm to 45µm in the various composites. Alteration phases such as serpentine, talc and chlorite content range from 2% to as high as 8% suggesting minor alteration throughout. Approximately 90% of all PGM grains identified are PGE sulphide species. Cooperite (PtS), Braggite (PtPd(Ni)S), Malanite (Pt(RhCu)S) and Laurite ((Ru(Rh)S) are the dominant species. Ag chlorides, Au alloys and trace amounts of Pt-Pd tellurides and arsenides make up the remaining 10% of the total PGM distribution. PGM mineral association data indicates that the majority (~65%) of PGM grains are associated with silicates while only ~25% of the PGM phases are associated with base metal sulphides and the remainder associated with chromite. This is somewhat unique for "normal" UG-2 reef which typically has higher proportions of PGM associated with base metal sulphides. Average PGM grain sizes are very fine ranging from 2µm to 7µm within the various composites. Composite 4 (West facies, LC reef, altered) has the coarsest average PGM grain size at 7µm. A summary of the discrete PGM measured by QEMSCAN is shown in Figure 2.

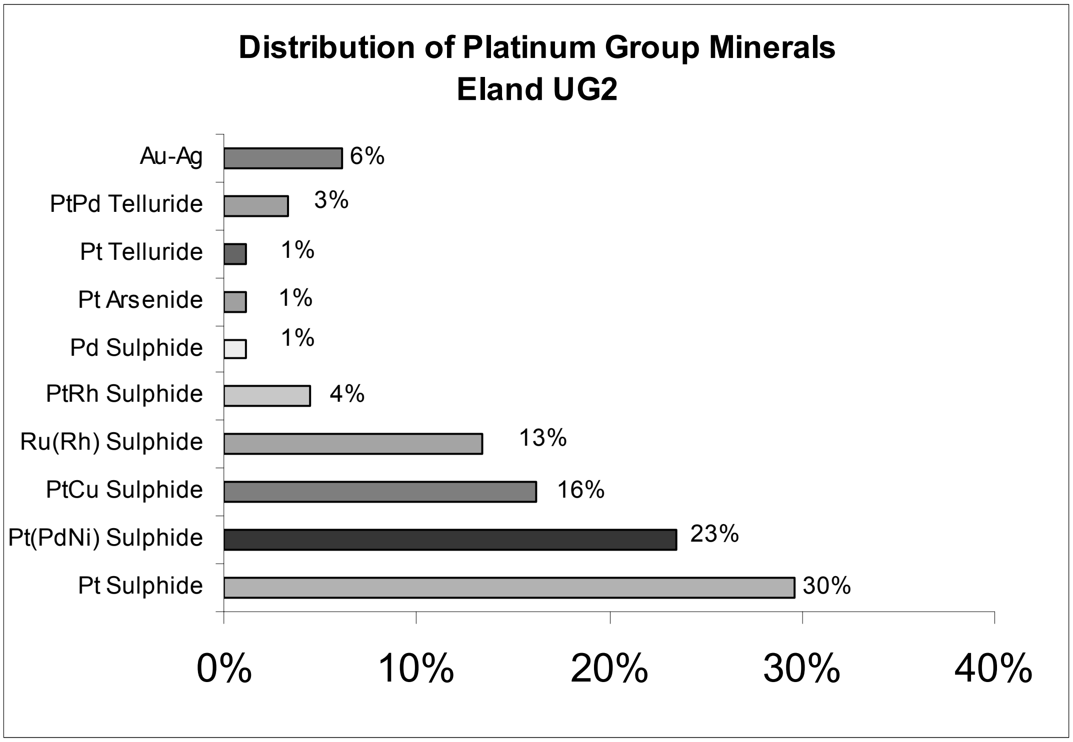

Figure 2: PGM mineral distribution from all ten composites combined based on 179 PGM grains

Selection of Candidate Collectors

From the above mineralogical information, it is clear that the target minerals to be floated are discrete PGM as discrete sulphides probably in a silicate-hosted texture (approximately 60-65%) and the balance as discrete PGM occurring as blebs attached to base metal sulphides such as pentlandite. This would be achieved by a factorial testing two different xanthates and a dithiocarbamate.

Execution of a Factorial Experiment with High-Confidence Flotation Testing

Tables 2-6 summarise the flotation test results (where 4E represents the group of four economic elements: platinum, palladium gold and rhodium).

Table 2: ERD – West (Normal) Ore

S(%)	Cr (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Rh (g/t)	4E (g/t)
0.03	16.86	1.98	0.76	0.02	0.36	3.12

Table 3: Baseline flotation tests

	SIBX g/t	Platinum		Palladium		Rhodium	
Mean of three replicates		Tails Grade (g/t)	Recovery (%)	Tails Grade (g/t)	Recovery (%)	Tails Grade (g/t)	Recovery (%)
	220	0.63	71.10	0.22	74.10	0.16	57.20
		Mass pull (%)	Overall Concentrate Grade (g/t)				
			Pt	Pd	Rh		
		8.59	14.44	5.08	1.46		

Table 4: Summary of factorial test data-Tailings and recoveries

Test Point	Xanthate 1: g/t	Xanthate 2: g/t	DTC g/t	Platinum		Palladium		Rhodium	
				Tails Grade g/t	Recovery %	Tails Grade g/t	Recovery %	Tails Grade g/t	Recovery %
1	60	60	60	0.63	70.64	0.22	72.83	0.16	57.48
2	90	60	60	0.64	71.77	0.23	75.83	0.14	61.55
3	60	90	60	0.62	72.52	0.21	74.68	0.16	61.25
4	90	90	60	0.58	73.52	0.21	74.90	0.15	62.17
5	60	60	90	0.66	70.10	0.24	70.90	0.16	57.77
6	90	60	90	0.63	71.49	0.23	73.19	0.16	60.80
7	60	90	90	0.60	73.05	0.22	74.55	0.15	64.00
8	90	90	90	0.63	71.79	0.22	73.26	0.16	61.74
MC	75	75	75	0.63	72.11	0.22	74.08	0.15	61.29

Table 5: Details of mid-cube replicates

Test Point	Xanthate 1: g/t	Xanthate 2: g/t	DTC g/t	Platinum		Palladium		Rhodium	
				Tails Grade g/t	Recovery %	Tails Grade g/t	Recovery %	Tails Grade g/t	Recovery %
MC1	75	75	75	0.62	72.56	0.22	73.94	0.13	64.67
MC2	75	75	75	0.70	70.40	0.24	73.49	0.16	58.83
MC3	75	75	75	0.59	71.63	0.22	73.51	0.16	61.71
MC4	75	75	75	0.62	73.23	0.22	74.70	0.14	65.41
MC5	75	75	75	0.59	73.01	0.21	73.76	0.15	61.55
MC6	75	75	75	0.64	71.85	0.23	75.12	0.17	55.64
MC	75	75	75	0.63	72.11	0.22	74.08	0.15	61.29
Standard Deviation				0.041	1.048	0.010	0.672	0.015	3.650
Standard Error				0.017	0.428	0.004	0.275	0.006	1.490

Table 6: Summary of factorial test data - Concentrate grades and mass pulls

Test Point	Xanthate 1: g/t	Xanthate 2: g/t	DTC g/t	Total Mass Pull %	Overall Concentrate Grade g/t		
					Pt	Pd	Rh
1	60	60	60	8.30	16.76	6.53	2.35
2	90	60	60	8.44	17.66	7.93	2.44
3	60	90	60	8.62	17.23	6.57	2.70
4	90	90	60	8.65	16.92	6.47	2.63
5	60	60	90	7.64	18.71	6.92	2.65
6	90	60	90	7.93	18.33	7.29	2.89
7	60	90	90	9.65	15.11	5.90	2.42
8	90	90	90	8.62	16.87	6.40	2.56
MC	75	75	75	8.59	17.48	6.94	2.64

Comparison to Baseline

A simple comparison of the grades and recoveries in the factorial cube against the baseline data gives an initial indication of how much the metallurgical performance has moved as a result of the various combinations of the three candidate mixed collectors. Figures 3-4 show these for platinum recovery and grade, with the baseline shown as a dotted line. Upper and lower confidence limits of the baseline mean are shown at the 95% confidence level as two standard errors. Recovery gains that exceed the two standard error level are in the range of 1.01-2.42% Pt; 0.58-1.73% Pd; and 3.6-6.8% Rh. For overall concentrate grade, these are 3.0-4.3 g/t Pt (or 20.7 – 29.8% relative); and 0.9 – 1.4 g/t Rh (or 61 – 98% relative). Palladium did not show any significant concentrate grade gain. These gains are consistent with the generic features of optimal mixed collector suites that have been published elsewhere in other work.

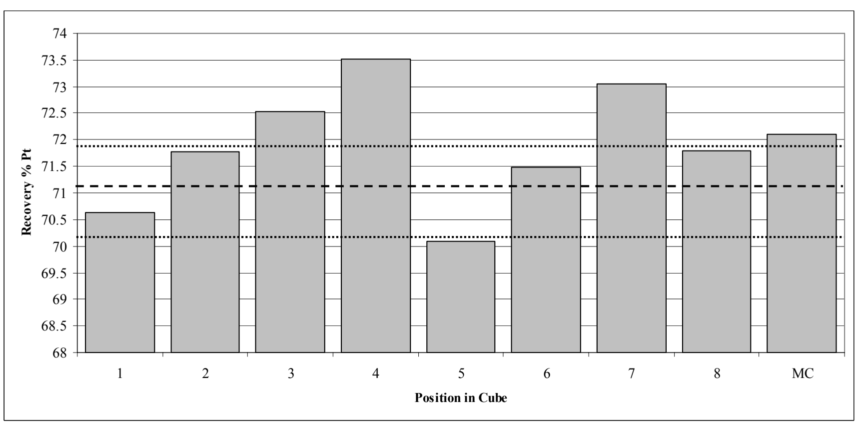

Figure 3: Comparison of platinum recoveries in factorial cube with the baseline (SIBX) : Result of 71.1%

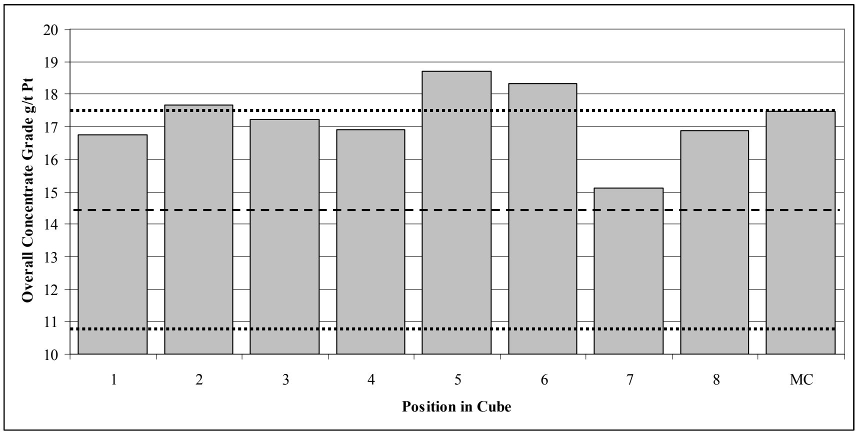

Figure 4: Comparison of platinum overall concentrate grades in factorial cube with the baseline (SIBX) -Result of 14.4 g/t

Analysis of Factorial

The following section contains detailed analysis of the factorial responses including mass pull and individual PGE responses.

Mass Pull

In UG2 flotation, mass pull across the rougher and scavenger flotation is a well-known indicator of performance. The main effects and interactions are summarized in Table 7.

Table 7: Main effects and interactions on mass pull

Detail	**Effect/Interaction**	
	% Mass Pull	**Test value of t**
Xanthate 1	-0.14	-0.59
Xanthate 2	**+0.61**	**2.58**
DTC	-0.04	-0.17
Xanthate 1/Xanthate 2	-0.36	-1.52
Xanthate 1/DTC	-0.24	-1.02
Xanthate 2/DTC	**+0.54**	**2.29**
Xanthate1/Xanthate 2/DTC	-0.30	-1.27

The mid-cube replicates showed a sample mean of 8.23% mass pull with a standard error of 0.236% mass pull. At the 95% level of confidence, any of these effects would have to exceed two standard errors (i.e. 0.48% mass pull) in order to be significant. Inspection of Table 7 shows that the main effect of Xanthate 2 (+0.61%) and the interaction between Xanthate 2 and DTC (+0.54%) both pass this level with t values of 2.58 and 2.29 respectively. The interaction between Xanthate 2 and DTC shows that at the high Xanthate 2 level, the DTC concentration has a stronger effect on the mass pull.

Table 8: Summary of main effects and interactions on recovery

Detail	**PLATINUM**		**PALLADIUM**		**RHODIUM**	
	Recovery %	**Test Value of t**	**Recovery %**	**Test Value of t**	**Recovery %**	**Test Value of t**
Standard Error	**0.428**		**0.275**		**1.49**	
Xanthate 1	+0.56	+1.31	**+1.06**	**3.85**	+1.45	0.97
Xanthate 2	**+1.72**	**+4.02**	**+1.15**	**4.16**	**+2.89**	**1.94**
DTC	-0.50	-1.17	**-1.59**	**-5.78**	+0.47	0.32
Xanthate 1/Xanthate 2	-0.70	-1.63	**-1.59**	**-5.78**	-2.11	1.41
Xanthate 1/DTC	-0.50	-1.17	**-0.55**	**2.00**	-1.07	0.72
Xanthate 2/DTC	-0.10	-0.23	**+0.69**	**2.52**	0.69	0.46
Xanthate 1/Xanthate 2/DTC	-0.62	-1.45	-0.21	-0.76	-0.54	-0.36

The standard error of the platinum recovery in the mid-cube replicates was 0.428%. Table 8 shows that the single significant effect is the Xanthate 2, achieving a gain in recovery of +1.72% Pt across the range of doses tested. The standard error for the palladium recovery in the mid-cube replicates is 0.275%. All main effects and interactions are significant except Xanthate 1/Xanthate 2/DTC. The standard error of the rhodium recovery in the mid-cube replicates was 1.49%. The main effect of the Xanthate 2 is +2.89% recovery, probably significant. The high standard error at 1.49% limits the t-value to 1.94, so we may only tentatively state that the Xanthate 2 has a main effect on Rh recovery, and should use caution in this interpretation.

Selection of a Mixed Collector Formula

These data were analysed so as to formulate a prototype mixture of the three candidate collectors for a plant trial. The criteria were grade and recovery of the individual PGE, and which combination of collectors offered most consistent benefit across the set of 4E. An optimal formulation was found, and its results compared to the baseline SIBX results. Whilst separate endeavours to advance the operations secondary grinding strategy were initiated, the composition of the mixed collector was named "Exp 820" by reagent manufacturers Senmin. Additionally, enquiries were to be made to learn more about factorial modelling techniques for future project work. It was expected that, in terms of 4E PGE data such as the mine used on a daily basis for measurement, that use of Exp 820 would increase 4E recovery by approximately 2.13% (disregarding the gold as an insignificant feed grade of 0.02 g/t).

Statistical Design of a Plant Trial

A week-long plant trial, nicknamed a "sighter" trial, was run so as to determine if there were any untoward effects of this prototype mixed collector. These results are summarized in Table 9.

Table 9: July 2009 "Sighter" trial data from Exp 820

Date	Shift	4E PGE Grades, g/t			Cr_2O_3%	Reagent
		Feed	Concentrate	Tails	Concentrate	
19 July	N	3.27	162.20	0.96	2.19	SIBX
18 July	M	2.88	257.83	0.81	2.28	SIBX
	A	3.33	144.03	0.92	2.44	SIBX
	N	2.82	128.20	0.88	1.77	SIBX
19 July	M	2.65	123.57	0.99	1.88	SIBX
	A	3.18	162.27	0.93	1.67	SIBX
	N	3.34	131.17	0.90	1.60	SIBX
22 July	M	2.85	141.57	0.91	2.47	SIBX
	A	**2.68**	**177.40**	**0.83**	**2.18**	**Changeover**
	N	3.10	206.20	0.84	1.64	Exp 820
23 July	M	3.27	122.97	0.73	1.97	Exp 820
	A	3.18	123.00	0.77	1.91	Exp 820
	N	2.76	120.70	0.78	2.46	Exp 820
24 July	M	2.66	126.90	0.78	1.64	Exp 820
	A	2.69	124.20	0.69	1.61	Exp 820
	N	2.44	109.90	0.68	2.03	Exp 820
25 July	M	3.37	120.93	0.85	1.72	Exp 820

Averages

SIBX	**3.04**	**156.4**	**0.91**	**2.04**
Mixed Collectors (Exp 820)	**2.93**	**131.9**	**0.76**	**1.87**

From these results, there were clearly recovery advantages for the Exp 820 trial collector. The formal on-off plant trial was then confirmed and executed. The results are summarized in Tables 10 and 11.

Table 10: Normal ore

Sept	**Ore Type**	**Collector**	**Oct**	**Ore Type**	**Collector**	**Nov**	**Ore Type**	**Collector**
18	N	Exp 820	9	N	Exp 820	9	N	Exp 820
19	N	Exp 820	10	N	Exp 820	10	N	Exp 820
20	N	Exp 820	11	N	Exp 820	11	N	Exp 820
21	N	Exp 820	12	N	Exp 820	12	N	Exp 820
22	N	Exp 820	13	N	Exp 820			
24	N	Exp 820	15	N	SIBX			
			16	N	SIBX			
			18	N	SIBX			
			19	N	SIBX			
			24	N	SIBX			
			25	N	SIBX			
			26	N	SIBX			
			27	N	SIBX			
			28	N	SIBX			
			29	N	SIBX			

SIBX Observations 10 days
Exp 820 Observations 15 days

Table 11: Normal ore, pooled raw data for SIBX and Exp 820

Collector	Feed Grade (g/t) PGE	Conc Grade (g/t) PGE	Tails Grade (g/t) 4E	Chrome in Conc (%)	Grind % -75 Microns
SIBX (Oct)	3.30	159.30	1.00	2.04	77.8
Exp 820 (Sept, Oct and Nov)	2.98	183.64	0.95	1.77	77.9

Closer inspection of the Exp 820 data suggests that the November results were different from those for September and October. Grades of 3.00, 175.04 and 1.11 g/t for feed, concentrate and tailings were recorded as the November average for Normal ore. These data have been pooled together with the September and October data in the above table. Separation of the three runs on Exp 820 into September, October and November as distinct treatments showed tailings grades averaging 0.90; 0.93; and 1.11 g/t respectively. Testing of these individual treatments using the Analysis of Variance produced the following results (Table 11).

Table 12: ANOVA: Comparison of the Three Exp 820 Data Blocks- Tailings

Source of Variation	Sum of Squares	Degrees of Freedom	Mean Square
Within Treatments	0.82	42	0.020
Between Treatments	0.28	2	0.14
Total about the Grand Average	1.10	44	0.160

Thus F= 0.14/0.02 = 7.00.

Referring to the tables of the F distribution at the 0.01% (99%) level, Fc= 5.18 at 2 and 42 degrees of freedom. Accordingly, it may be stated that the final tailings grades produced from Normal ore in November are significantly different to those from September and October. Unfortunately there are no data for treating normal ore using SIBX in November. The November trial data were thus excluded.

Now pooling the September and October data as a reference treatment using Exp 820, the following results (Table 13) are obtained and are analysed in Table 14.

Table 13: Normal ore, pooled data for SIBX and Exp 820

Collector	Feed Grade (g/t) PGE	Conc Grade (g/t) PGE	Tails Grade (g/t) 4E	Chrome in Conc (%)	Grind (% -75 microns)
SIBX	3.30	159.30	1.00	2.04	77.8
Exp 820	2.97	185.79	0.91	1.62	77.8

Table 14: ANOVA: Comparison of the SIBX and Exp 820 Data Blocks: Tailings

Source of Variation	Sum of Squares	Degrees of Freedom	Mean Square
Within Treatments	1.613	63	0.02561
Between Treatments	0.1467	2	0.0738
Total about the Grand Average	1.7597	65	0.09941

Thus F= 0.0738/0.02561 = 2.86.

Referring to the tables of the F distribution at the 0.1% (90%) level, Fc= 2.39 at 2 and 63 degrees of freedom. Accordingly it may be stated that the final tailings grades produced from Normal ore by Exp 820 are significantly lower than those produced by SIBX. The distributions of tailing grades for each collector type are shown in Figure 5.

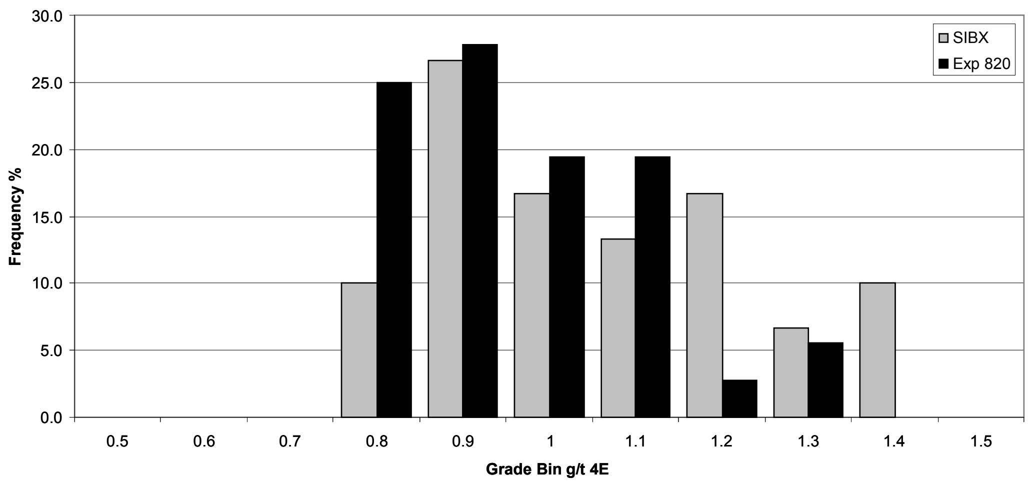

Figure 5: Histograms of final tailings grades by collector type, normal ore

Table 15: ANOVA: Comparison of the SIBX and Exp 820 data blocks - Final concentrate

Source of Variation	Sum of Squares	Degrees of Freedom	Mean Square
Within Treatments	102827.9	63	1632.19
Between Treatments	15348.99	2	7674.494
Total about the Grand Average	118176.89	65	9306.684

Thus F = 7674.49/1632.19 = 4.70

Referring to the tables of the F distribution at the 0.05% (95%) level, Fc= 3.15 at 2 and 63 degrees of freedom. Accordingly it may be stated that the final concentrate grades produced from Normal ore by Exp 820 are significantly higher than those produced by SIBX, as shown in Figure 6.

It is also noted that the feed grades for the SIBX and Exp 820 data blocks are different, with the SIBX feed higher at 3.30 g/t (Exp 820: 2.97 g/t). There is usually a correlation between feed grade and tailings grade. The plots for this relationship are shown in Figure 7, and for overall grade and recovery, in Figure 8.

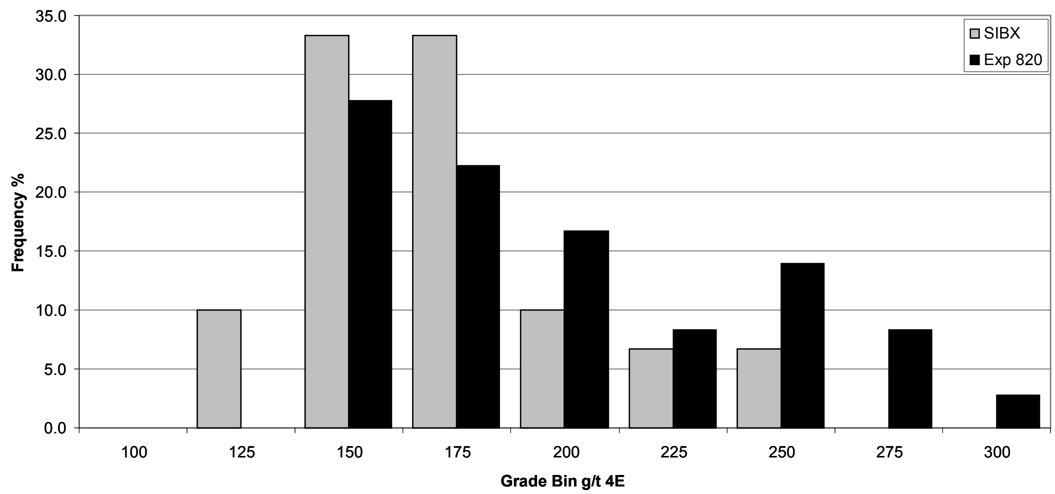

Figure 6: Histograms of final concentrate grade by collector type - Normal ore

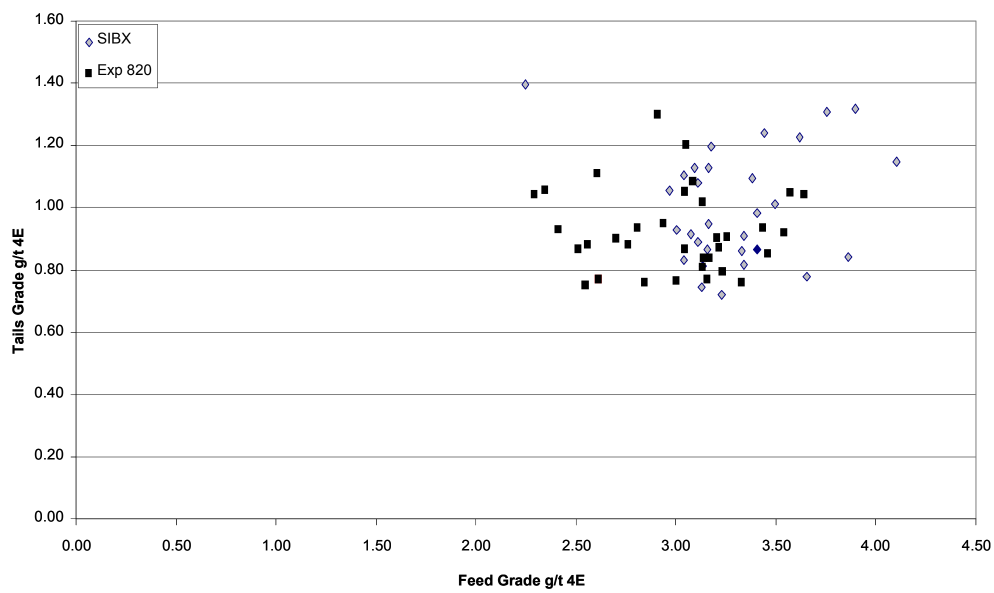

Figure 7: Feed and tailing grade relationships, Normal Ore

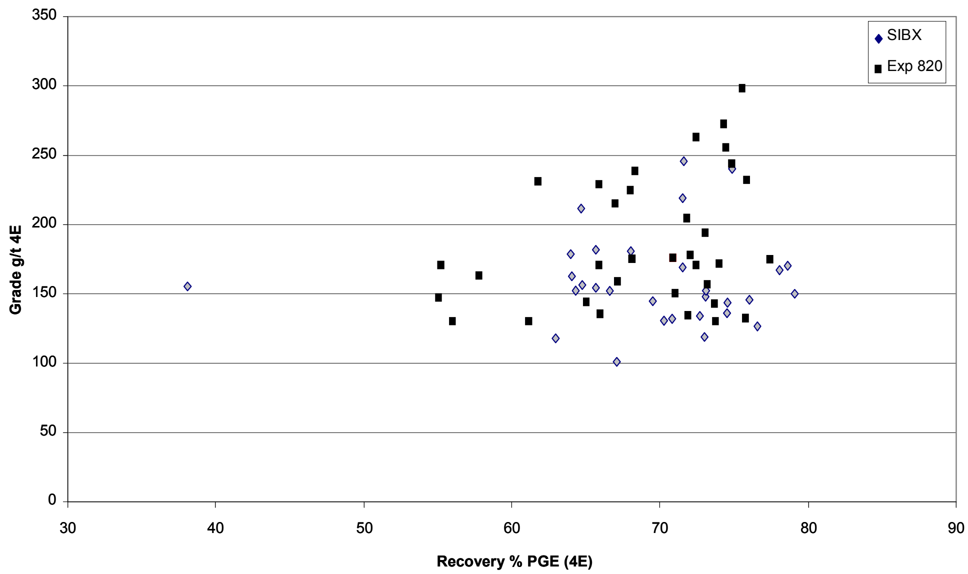

Figure 8: Grade / recovery domains for SIBX and Exp 820, Normal ore

A model was run from the statistical software SAS to estimate coefficients and an overall significance for the relationship between feed, concentrate and tailings grade, grind, and reagent type. The model obtained was

Tail = 1.551 +0.008Feed -0.000839Conc -0.06664Type -0.0056Grind [1]

Where

Tail	=	final tailings grade, g/t PGE
Feed	=	feed grade, g/t PGE
Type	=	Collector type [0:= SIBX; 1:= Exp 820)
Grind	=	Final Grind, % Mass Passing 75 microns

The overall model F was 2.35 with a tail area of 0.06, i.e. significant at the 94% confidence level. Using the data file mean value of feed (3.12 g/t), and respecting the difference of concentrate grade of +26.39 g/t in favour of the Exp 820, this resolves to

Tail = 0.996 – 0.06664Type -.022 for Exp 820 [2]

And

Tail = 0.996 – 0.06664Type for SIBX [3]

The modelled tailing values by collector type, after normalizing for different feed grades, will then be:

SIBX:	0.99	g/t
Exp 820:	0.91	g/t

These modelled values agree closely with the actual test data means of 1.00 g/t (SIBX) and 0.91 g/t (Exp 820). Referring now to the two-product formula, we estimate the recoveries attributable to collector type for the same feed grades:

$$R_{SIBX} = \frac{(3.12 - 0.99)}{(159.3 - 0.99)} x \frac{159.3}{3.12} x100 \qquad = \quad 68.70\%$$

$$R_{Exp820} = \frac{(3.12 - 0.91)}{(185.79 - 0.91)} x \frac{185.79}{3.12} x100 \qquad = \quad 71.18\%$$

Thus a recovery gain of 2.48% PGE has been realized with the Exp 820 mixed collector for equivalent feed grades, with an improved concentrate grade by 26.39 g/t, at better than the 94% confidence level, for the Normal ore.

Inspection of the grade of Cr_2O_3 in final concentrate in Table 18 shows mean grades of 2.04 and 1.62% for SIBX and Exp 820, respectively. It is understood that the grade of Cr_2O_3 in final concentrate affects furnace operations and is described in the toll treatment contract, so the lower this level is, the more favourably will the toll-treating smelter regard the treatment of the Eland concentrate. An apparent drop of (2.04 – 1.62) = 0.42% will now be tested for significance using ANOVA.

Table 16: ANOVA: Cr_2O_3 in final concentrate - Normal ore

Source of Variation	Sum of Squares	Degrees of Freedom	Mean Square
Within Treatments	11.377	63	0.181
Between Treatments	2.943	2	1.472
Total about the Grand Average	14.314	65	1.653

Thus F = 1.472/0.181 = 8.15

Referring to the tables of the F distribution at the 0.05% (95%) level, Fc= 3.15 at 2 and 63 degrees of freedom. At the 0.01% (99%) level, this is 4.98, and at the 0.001% (99.99%) level, 7.76. Accordingly it may be concluded that the Exp 820 has significantly reduced the grade of Cr_2O_3 in final concentrate by 0.42% absolute at better than the 99.99% confidence level.

CONCLUDING REMARKS

The mixed collector theory advanced by Lotter and Bradshaw, 2009, has been validated by this work. The metallurgical performance gains to be delivered from an optimised mixed collector suite have been demonstrated at both laboratory and plant scale. The Eland operations now have a 2.48% PGE recovery gain (compared to an estimated 2.2% gain from laboratory scale testwork), together with an improvement in concentrate grade equivalent to a 16% relative improvement as well as a reduction in chrome in final concentrate. The agreement between laboratory and plant scale PGE recovery (2.2 vs 2.5%) is close, and reflects the type of scale-up that may be achieved with suitable representative ore sampling, mineralogy, quality controls and high-confidence flotation testing.

FURTHER WORK

Ongoing Endeavour

A second phase of mixed collector formulation has been started in 2010. This will build on the successes reported here for the first prototype. In this second phase, use will be made of the formulation of optimal mixtures after the work of Cornell, 2002, and adding another collector as an additive to the mixed collector suite.

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