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METALLURGICAL TEST REPORT

FOR

Cardinal Resources Limited

REPORT NO. Rpt 034

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PROCESS DEVELOPMENT TEST WORK FOR GOLD RECOVERY FROM NAMDINI GOLD PROJECT IN GHANA

CONFIDENTIALITY

Abbreviations

MC – Master composite

V – Volcanoclastics

G – Granitic

D – Diorites

PAX – Potassium Amyl Xanthate

CRl - Cardinal Resources limited

ECD – Equivalent circular diameter

LDL – Lower detection limit

UDL – Upper detection limit

OPEX – Operating expenditure

CAPEX – Capital expenditure

UFG – Ultrafine grinding

EXECUTIVE SUMMARY

The Namdini ROM Au was highly amenable to beneficiation by froth flotation with a rougher flotation recovery of up to 95% achieved but partially refractory during cyanidation with Au leach recoveries of ~64,1% at the optimum grind of 80%-75µm; pure O_2 increased Au leach recovery by ~5,6%. The bulk of the Au was associated with or hosted by pyrite (~75.4%) while an additional 10.6% was hosted by other sulphides of average particle size ~70ECD(µm); the Au particles identified were fine being less than 13.5ECD(µm) hence poor liberation (~31.8% was locked and only 27.5% liberated). Ultra-fine milling the concentrates at 10kWh/t improved Au dissolution by ~12.2% to attain a total leach recovery of ~80.2% while roasting the concentrate increased Au dissolution to ~86.5%. The high cyanide consumption of between 1.35kg/t and 1.60kg/t could affect project economics while the low demand for lime will have a positive impact on OPEX.

Based on the results obtained, the author recommends 2 flowsheets both incorporating a front end rougher flotation stage targeting an overall flotation recovery of ~95%, mass pull of ~8% and concentrate grade of between 15g/t, Au and 25g/t, Au. The first option (Low CAPEX) would then require UFG of the concentrate to 80%-10µm and cyanidation to obtain a recovery of ~80% which implies an overall recovery of 80%x95%=**76%**. The second option will incorporate biological degradation of sulphides targeting an overall leach recovery of ~95% (as it is possible to liberate Au in the sub-microscopic size range (<3µm)) to achieve an overall Au recovery of ~90%.

TECHNICAL SUMMARY

Item	Description	Units	MC	V	G	D
	CHARACTERISATION TEST RESULTS					
1	Head assay results, Au	g/t	1,42	1,88	0,88	1,59
2	Head assay results, S^{2-}	%	1,2	0,9	0,9	2,3
3	Specific gravity (SG)		2,74			
4	Moisture content	%	0,1			
5	Spi	kWh/t	9,57	8,83	8,83	9,13
6	BBWi	kWh/t	14,9			
	Mineralogy					
7	Au associated with S (pyrite @75.4%) ROM	%	86.6			
	Au associated quartz	%	5,8			
	Free surface Au	%	5,6			
8	Liberated Au @80%-75µm of ROM	%	27,5			
	Locked Au @ 80%-75µm of ROM	%	31,8			
9	Au grain size @80%-75µm: %-3µm	%	61,2			
	FLOTATION TEST WORK RESULTS					
10	Optimum grind for flotation and cyanidation	%	80%-75µm			
11	Optimum performance (Test 7 results): Au recovery	%	94,2			
	Optimum conditions: Mass pull	%	5,4			
	Optimum conditions: Ro conc grade	g/t	22,72			
	CYANIDATION TEST WORK RESULTS ON FEED SAMPLE					
12	Au leach recovery - conventional cyanidation	%	64,1	63,3	46,8	66,7
	Au leach recovery - $PbNO_3$ @ 1,6kg/t	%	66,2			
	Au leach recovery - Pure O_2	%	69,7			
	Au leach recovery - H_2O_2	%	67,6			
	Au leach recovery - Air injection	%	64,1			
	CYANIDATION TEST WORK RESULTS ON FEED SAMPLE					
12	Au leach recovery - conventional cyanidation	%	67,8			
	Au leach recovery - $PbNO_3$ @ 1,6kg/t	%	69,1			
	Au leach recovery - Pure O_2	%	65,9			
	Au leach recovery - H_2O_2	%	65,7			
	Au leach recovery - Air injection	%	64,4			
	Maximum Au leach rec on Float conc + UFG @10kWh/t	%	80,2			
	Maximum Au leach rec on Float conc + roasting	%	86,5			
	DIAGNOSTIC LEACH RESULTS ON ROM					
13	Au available to direct cyanidation (no carbon)	%	61,3			
	Au available to direct cyanidation (CIL)	%	64,1			
	Preg-robbed Au	%	2,8			
	Au liberated after mild oxidative pre-leach	%	8,0			
	Au liberated after severe oxidative pre-leach	%	27,3			
	Au available after complete roasting	%	0			
	Au undissolved (associated with quartz)	%	0			
	DIAGNOSTIC LEACH RESULTS ON FLOTATION CONCENTRATE					
14	Au available to direct cyanidation (no carbon)	%	66,7			
	Au available to direct cyanidation (CIL)	%	67,8			
	Preg-robbed Au	%	1,1			
	Au liberated after mild oxidative pre-leach	%	3,0			
	Au liberated after severe oxidative pre-leach	%	19,0			
	Au available after complete roasting	%	1,4			
	Au undissolved (associated with quartz)	%	8,7			
	REAGENT CONSUMPTIONS					
15	NaCN consumption	Kg/t	1,55	1,6	1,35	1,6
16	Lime consumption	Kg/t	0,53	0,26	0,29	0,42

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1.0. INTRODUCTION

Cardinal Resources limited (CRl) has had continued exploration success over a long period at its flagship Namdini gold project in Ghana; the company has 3 other projects in Ghana namely Bolgatanga, Kungongo and Subranum; the Kungongo tenement lies within the Bolgatanga project. Namdini represents a genuine new gold discovery with previous high grade intersections of 67 metres at 3.10 g/t, Au and 45 metres at 7.73g/t, Au and exhibits wide zones of mineralisation open in many directions. Figure 1 illustrates the location of the projects.

Figure 1: Map showing position of projects within Ghana for CRl

Satisfied with the exploration results and the resource established to date, Mr Paul Abbott the Exploration Manager for CRl engaged Suntech Geomet Laboratories to carry out preliminary metallurgical test work on gold samples obtained from their Namdini Gold deposit. The samples tested are identified herein as;

- ✓ Volcanoclastic – coded (V) samples,
- ✓ Granitic lithologies – coded (G) samples and
- ✓ Diorites – coded (D) samples

The objective of the metallurgical test work was to determine if flotation will successfully produce a gold-rich concentrate that can be further processed/ treated

for recovery of gold into bullion via standard CIL and/or by any other means. It was also envisaged to establish the mineralogical and metallurgical characteristic of the 3 ore samples as well as those for the composite of the 3. This metallurgical test report provides details of the metallurgical test work carried out to date and the results obtained at this stage.

2.0. SAMPLE RECEIPT AND VERIFICATION

2.1. Sample receipt

Gold samples from the Namdini Gold deposit were received at Suntech Geomet laboratory on the 1st of July 2016 packed in 18 strap-locked and sealed cargo boxes. Upon receipt, pictures were taken in order to show the condition of the samples at the time of receipt; Figure 2 illustrates the cargo boxes containing the samples received.

Figure 2: Cargo boxes containing Namdini gold samples

In total, 18 boxes were delivered at Suntech by DHL; these were accompanied by documents specifying the details and weights of the samples. At the time of receipt, no signs of damage were observed and seals on all boxes were still intact. The cargo boxes were individually opened and pictures taken to show the state of the contents at the time of receipt; Figure 3 illustrates the contents.

Figure 3: Namdini Gold samples inside the cargo boxes

2.2. Sample verification

Each of the 18 cargo boxes contained 2 strap-locked plastic sample bags; all bags from each of the V, G and D samples were opened and pictures of the contents taken in order to show their state at the time of receipt; Figure 4 illustrates the contents of selected sample bags.

Figure 4: Contents of the plastic sample bags containing Namdini drill cores

The sample bags contained quarter drill core samples tagged with relevant sample IDs. The contents of every bag were verified and quantified (for comparison with client weights);

Table 1 presents the comparative sample masses of the individual sample bags as weighed by the client and Suntech while Table 2 presents the combined sample masses for each of the 3 lithology.

Table 1: Comparative masses for the V, G and D samples from Namdini

Item	Sample ID	Sample mass (kg)		Var (%)
		Namdini	Suntech	
1	V1	9,4	9,4	0%
2	V2	8,8	8,8	0%
3	V3	9,1	9,2	1%
4	V4	9,2	9,2	0%
5	V5	8,9	8,9	0%
6	V6	9,5	9,4	1%
7	V7	7,1	7,2	1%
8	V8	8,3	8,3	0%
9	V9	8,3	8,4	1%
10	V10	7,4	7,4	0%
11	V11	13,0	13,0	0%
12	V12	16,6	16,6	0%
13	V13	10,1	10,1	0%
14	V14	10,2	10,2	0%
15	G1	8,3	8,3	0%
16	G2	8,5	8,4	1%
17	G3	8,4	8,6	2%
18	G4	8,6	8,8	2%
19	G5	8,4	8,4	0%
20	G6	8,3	8,2	1%
21	G7	8,1	8,1	0%
22	G8	8,2	8,4	2%
23	G9	7,9	8,0	1%
24	G10	7,9	7,8	1%
25	G11	8,2	8,2	0%
26	G12	7,8	7,8	1%
27	G13	8,6	8,6	0%
28	G14	8,3	8,2	1%
29	G15	9,6	9,6	0%
30	G16	12,2	12,1	1%
31	D3	9,7	9,6	1%
32	D4	9,6	9,6	0%
33	D5	9,7	9,8	1%
34	D6	10,1	10,0	1%
35	D7	10,0	10,1	1%
36	D9	9,5	9,5	1%

Table 2: Total sample weights for the Namdini gold samples

Item	Sample ID	Sample mass (kg)		Var (%)
		Namdini	Suntech	
1	Volcaniclastics (V sample)	136,0	136,1	0%
2	Granitic lithologies (G sample)	137,3	137,5	0%
3	Diorites (D sample)	58,5	58,6	0%
Total mass, kg		331,8	332,2	0%

The V, G and D samples weighed ~136.1kg, ~137.5kg and ~58.6kg respectively. No signs of contamination were observed; samples were stored in a contamination free environment prior to test work.

3.0. TEST WORK PROCEDURES AND RESULTS

3.1. SCOPE OF TEST WORK

At this stage of metallurgical testing, it was envisioned to establish:

i. the best reagent suite for optimal recovery and upgrading of Au into a concentrate suitable for further processing,
ii. the acceptable coarsest grind for optimal flotation recovery of Au,
iii. the mineralogical make-up of the feed ore and concentrate via QEMScan and diagnostic leach evaluations and
iv. the amenability of Au to recovery by conventional cyanidation among other things from both the ROM ore and from flotation concentrate.

3.2. SAMPLE PREPARATION ON LITHOLOGY SAMPLES

Sample preparation was undertaken as illustrated in Figure 5.

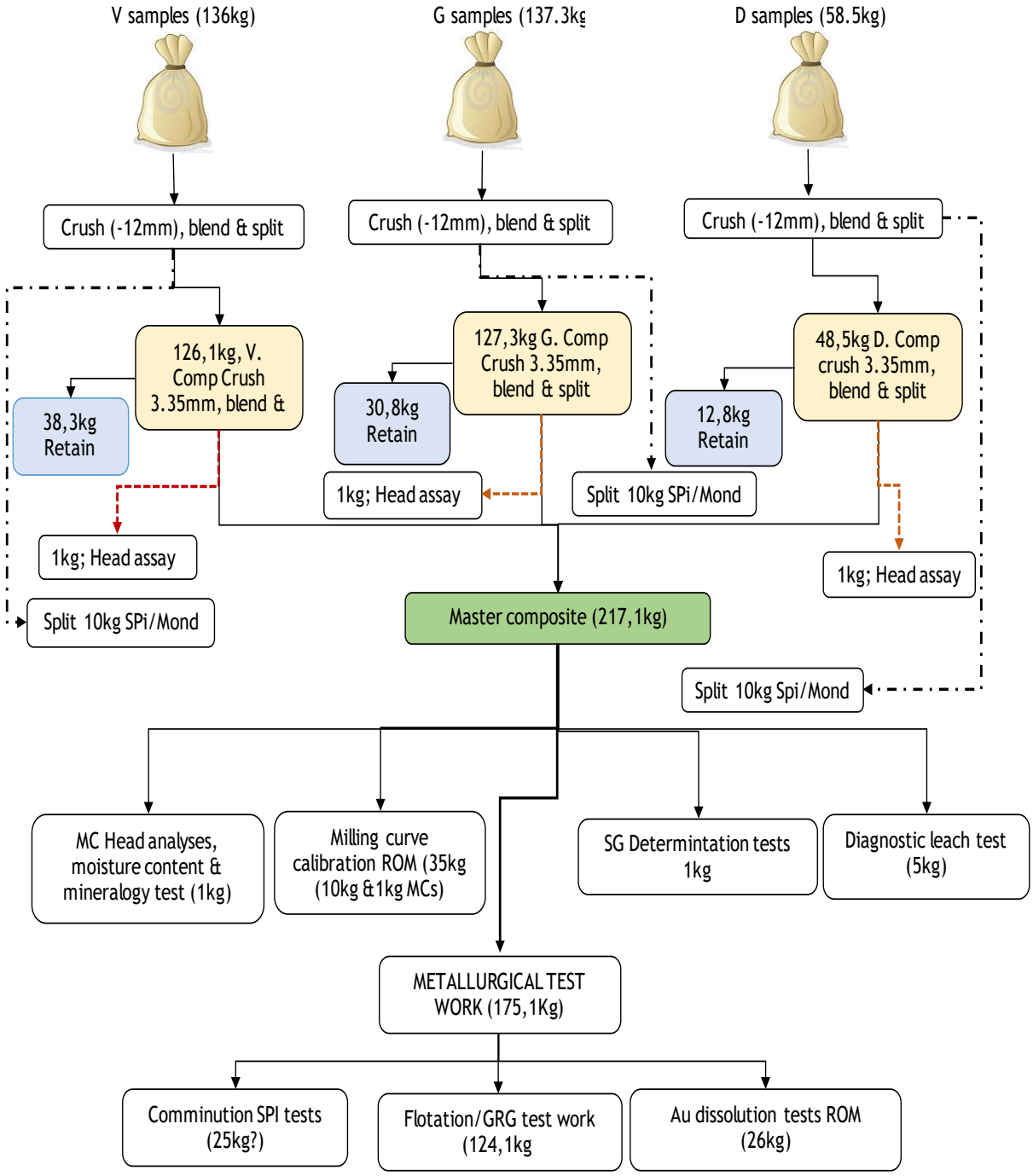


Figure 5: Sample preparation procedure for Namdini samples

Each of the 3 lithology samples were separately stage crushed to 100%-12mm, thoroughly blended and split by coning and quartering to remove ~10kg aliquots from each lithology for SPi/Mod Bond test work. The remaining samples from each lithology were separately stage crushed to 100%-3.35mm, blended and split for compositing as presented in Table 3.

Table 3: Compositing ratios for preparation of the master composite

Item	Sample ID/Lithology	Amount of sample		Total Qty (Kg)	Comp dist (%)
		Retained (kg)	Composited (kg)		
1	V samples	38,3	86,8	136,1	40,0%
2	G samples	30,8	95,5	137,5	44,0%
3	D samples	12,8	34,7	58,6	16,0%
4		81,9	217,0	332,2	100%

The compositing ratios were 10:11:4 for the V, G and D samples respectively while the corresponding samples remaining after compositing amounted to ~38.3kg, ~30.8kg and ~12.8kg. Approximately 10kg aliquots were removed from each of the lithology for Spi/Mod Bond test work while the remaining samples were stored in a contamination free environment awaiting further test work.

3.3. SAMPLE PREPARATION ON MASTER COMPOSITE SAMPLE

Representative aliquots of the V, G & D samples (stage crushed to 100%-3.35mm) were blended (in ratios presented in Table 3) to make a master composite (MC) weighing ~217.0Kg for the metallurgical test work. The blended sample was further split into representative aliquots of various masses in preparation for the metallurgical test work.

3.4. CHARACTERISATION TEST RESULTS

A selection of characterisation tests were carried out on representative aliquots of the master composite and the V, G & D samples in order to establish their metallurgical characteristics; results obtained are presented in the following sections.

3.4.1.1. Head assay results

Representative aliquots of each of the V, G, D and the master composites were sub-sampled from the bulk composite by riffle and rotary splitting, dried at 105^{o}C and pulverised to ~90%-75µm prior to Au and other analyses. The pulverised samples were further rotary split into 2x150g portions for Au analyses at Suntech and Mintek

laboratories. The sample analysed at Suntech was further rotary split into representative triplicate aliquots weighing ~10g each for Au analysis by fire assaying. The aliquot submitted at Minteck was again rotary split into 2x~50g portions for Au analysis by fire assaying while the remaining sample was further split into ~2g portions for comprehensive analyses by multi-acid digestion and ICP finish while ~1g each was used for S^{2-} and C analyses by Leco and combustion respectively in line with the analytical schemes presented in Table 4.

Table 4: Analytical schemes used on Namdini Au samples

Elements	Analytical scheme	Detection limit
Au	Fire assay, fusion lead collection with ICP Finish	Lower limit 0.05g/t
Ag, As	Atomic Absorption Spectroscopy (AAS)	Lower limit 5mg/Kg
Total sulphur	Total combustion ("Leco")	0.01%
Organic carbon	Combustion after extraction	0.01%
Multi-elements	Inductively Coupled Plasma-Optical Emmission Spectroscopy (ICP-OES)	Lower limit 0.05%
Matrix dependant	XRF analysis: All elements at detectable concentrations	Lower limit 0.05%

Table 5 presents the method detection limits as well as the quality control standards that were used in fulfillment of the requirements for the SANAS ISO 17025 standards as well as the rejection/acceptance criterion adopted for Au analyses.

Table 5: Detection limits & rejection criteria applied for Namdini Au analyses

Method detection limits	Unit		g/t, Au
	LDL		0.02
	UDL		100.0
Quality control standards	AMIS 307	Acceptable assay value	**0.43**
		Acceptable assay range	0.37 - 0.49
	AMIS 168	Acceptable assay value	**1.20**
		Acceptable assay range	1.06 - 1.34
	AMIS 335	Acceptable assay value	**3.83**
		Acceptable assay range	3.55 - 4.11
	AMIS 369	Acceptable assay value	**26.36**
		Acceptable assay range	23.96 - 28.76
	AMIS 245	Acceptable assay value	**88.42**
		Acceptable assay range	83.44 - 93.40

The head assay results obtained are summarised in **Table 6** while the results of comprehensive analyses are presented in

Table 7.

Table 6: Head assay results for the Namdini lithology & composite samples

Item	Description	V Lithology	G lithology	Diorites, D	MC - Suntech	MC - Mintek
		Assay, Au g/t	Assay, Au g/t	Assay, Au g/t	Assay, Au g/t	Assay, Au g/t
1	Aliquot 1	1,96	1,01	1,45	1,25	
2	Aliquot 2	1,77	0,78	1,70	1,51	1,27
3	Aliquot 3	1,91	0,86	1,62	1,50	1,50
4	**Average**	**1,88**	**0,88**	**1,59**	**1,42**	**1,39**
Compositing ratios		**40**	**44**	**16**	**100**	**100**

The V sample contained the highest concentration of Au when compared to the other 2 lithology with an average assay head value of ~1.88g/t Au; the corresponding Au assay values for the G and D samples were ~0.88g/t Au and ~1.59g/t Au. By calculation, the back calculated head grade of the MC sample was 1.40g/t Au while the assayed head grade of the master composite was ~1.42 g/t Au. These 2 figures confirmed each other with reasonable limits with a variance of only 1.8%.

3.4.1.2. Results of comprehensive chemical analyses on head

Representative aliquots of the V, G, D and the master composite head samples were cut out from the bulk by riffle and rotary splitting, pulverised to 90%-75µm and analysed for multiple elements including $S_{package}$ and $C_{package}$ (Leco), Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, La, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sn, Sr, Te, Th, Tl, Ti, U, V, W, Y, Zn, Zr using multi-acid digestion with ICP multi-element ICP-OES finish; the results obtained are presented in

Table 7.

Table 7: Results of comprehensive chemical analyses on head samples

Item	Sample description	Ag	Al	As	B	Ba	Be	Bi	Ca	Cd	Co	Cr	Cu	Fe	Ga	In	K	Li	Mg	Mn	Mo	Na	Nb	Ni
		ppm	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	%	ppm	ppm	%	ppm	ppm
1	Volcaniclastics (V)	<3	7,21	545	<20	408	<1.2	<15	5,7	<2.5	31	206	55	7,98	15,4	<10	1,42	17,7	2,57	840	9,23	1,39	<5.00	136
2	Granitic lithologies (G)	<3	8,16	225	<20	677	<1.2	<15	2,1	<2.5	11,3	78	24	2,76	17,7	<10	1,69	<10.5	0,54	346	10,6	3,12	7,28	22
3	Diorites (D)	<3	6,79	299	<20	154	<1.2	<15	5,8	<2.5	40	133	125	11,2	15,3	<10	1,37	22	2,79	1101	18,5	1,31	<5.00	101
4	Weighted mean		7,561	364,8		486			4,1		23,8	138	52,6	6,2	16,4		1,53		1,7	664	11,3	2,1		80,2
5	Master Composite (MC)	<3	7,683	388	<20	522	<1,2	<15	4,2	<2,5	20,2	131	52,3	6,3	16,2	<10	1,54	12,5	1,8	690	10,2	2,1	<5,00	79,7
6	*Variance (%)*		*1,6%*	*6,0%*		*7,0%*			*0,0*		*17,9%*	*5,3%*	*0,4%*	*2,2%*	*1,0%*		*0,6%*		*3,5%*	*3,7%*	*10,9%*	*1,8%*		*0,7%*

Item	Sample description	P	Pb	Rb	S^{2-}	Sb	Sc	Se	Sn	Sr	Ta	Te	Th	Tl	Ti	U	V	W	Y	Zn	Zr	Hg	C	S
		ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%
1	Volcaniclastics (V)	595	26	53	0,8	5,92	20	<10	<10	365	<5	<10	<4	<1	1990	0,77	148	25	10	89	70	0,35	2,4	0,9
2	Granitic lithologies (G)	569	17,8	60	0,8	8,51	<6	<10	<10	439	<5	<10	4,24	<1	954	2,3	37	45	4,93	52	71	1,2	1,0	0,9
3	Diorites (D)	236	24	75	2,1	4,03	23	<10	<10	176	<5	<10	<4	<1	1990	<0.3	173	<10	7,45	111	31	<0.2	2,8	2,3
4	Weighted mean	526	22,07	59,6	1	6,76	12			367					1534	1,32	103		7,4	76,2	64,2		1,9	1,2
5	Master Composite (MC)	543	37,33	58,33	1,1	6,77	13	<10	<10	372	<5	<10	<4	<1	1558	1,27	105	21,7	7,26	81,3	65,33	0,43	1,8	1,2
6	*Variance (%)*	*3%*	*41%*	*2%*	*4%*	*0%*	*10%*			*1%*					*2%*	*4%*	*1%*		*1%*	*6%*	*2%*		*2%*	*0%*

None of the 3 samples contained Ag but had substantial amounts of As and C with average assay values of ~365ppm and 1.9% respectively. The samples also contained Cu assaying on average ~52.6ppm; one needs to establish the quantity of E_{Cu} (Cyanide easily soluble copper) so as to assess the impact thereof on cyanide consumption during cyanidation. All 3 samples contained reasonable concentrations of S with an average S^T of ~1.2%. The bulk of the sulphur found in these samples was S^{2-} which analysed ~1.1%.

3.4.1.3. Distribution of ore particles, Au and S^{2-} by size (PSD)

The distribution of ore and Au particles by size was analysed through screening tests undertaken on representative portions of the Master composite, V, G and D samples. Approximately 1kg of each of the samples was wet screened through a 53µm aperture screen and both screening products filtered, dried and weighed. The +53µm fractions were then dry screened through a stack of root 2 series of screens starting from the

1mm screen generating 11 products per each sample. Each of the screen products coarser than 53µm were pulverised to 90%-75µm and rotary split to remove ~10g and ~1g aliquots for Au and S^{2-} analyses. Gold analyses was done at Suntech by fire assaying, lead collection with ICP finish while S^{2-} analysis was done at Mintek laboratories by the Leco "Total combustion" method.

3.4.3.1 Distribution of ore, Au and S^{2-} particles by size – Master composite

The results obtained for the Master composite sample are presented in Table 8 while Figure 6 illustrates the bar graph developed from these results.

Table 8: Distribution of ore, Au and S^{2-} particles by size – Master composite

Particle size range	Mass distribution				Fraction assays		Distribution	
	(g)	(%)	Cum (%+)	Cum (%-)	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-}(%)
+850µm	524,28	52,8	52,8	47,2	1,45	0,83	51,3	38,5
-850µm+600µm	80,40	8,1	60,9	39,1	1,2	1,75	6,5	12,4
-600µm+425µm	51,57	5,2	66,1	33,9	1,05	1,3	3,7	5,9
-425µm+300µm	51,82	5,2	71,3	28,7	1,00	2,32	3,5	10,6
-300µm+212µm	39,04	3,9	75,2	24,8	1,89	1,54	5,0	5,3
-212µm+150µm	31,52	3,2	78,4	21,6	1,29	1,63	2,7	4,5
-150µm+106µm	25,97	2,6	81,0	19,0	2,32	3,95	4,1	9,1
-106µm+75µm	19,26	1,9	83,0	17,0	3,64	2,22	4,7	3,8
-75µm+53µm	18,06	1,8	84,8	15,2	2,48	2,03	3,0	3,2
-53µm	151,00	15,2	100	0,0	1,52	0,49	15,5	6,5
Head calc	992,9	100			1,49	1,14	100	100
Head assays					1,42	0,99		
Variance (%)					4,8%	13,1%		

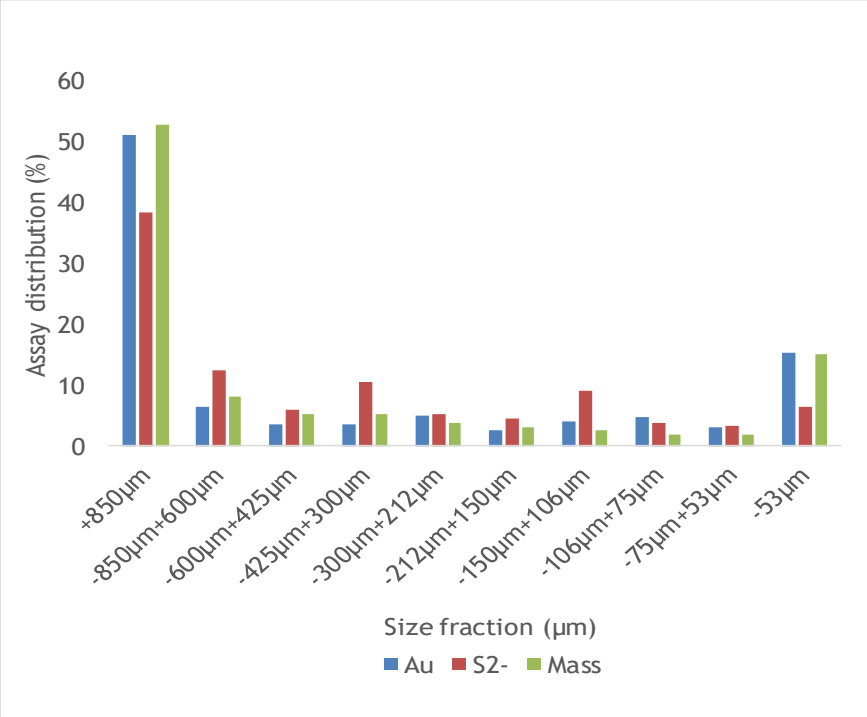


Figure 6: Distribution of ore, Au and S^{2-} particles by size – Master composite

The bulk of the Au found in the Master composite sample was contained within the coarse fractions (+850µm) which accounted for ~51.3% and ~52.8% of the total Au and mass respectively while the finest fraction (i.e. -53µm) accounted for ~15.6% and 15.2% of the total Au and mass respectively showing that the distribution of gold was influenced by mass and not grade. A similar trend was noticed for all the other size fractions. Distribution of sulphur also exhibited a similar trend effectively confirming that the Au found in these samples was largely associated with and/or hosted by sulphur.

3.4.3.2 Distribution of ore, Au and S^{2-} particles by size – V samples

The distributions of Au and S2- within the Volcanoclastics samples are presented in Table 9; Figure 7 illustrates the bar graph developed from these results.

Table 9: Distribution of ore, Au and S^{2-} particles by size (Volcanoclastics)

Particle size range	Mass distribution				Fraction assays		Distribution	
	(g)	(%)	Cum (%+)	Cum (%-)	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-}(%)
+850µm	548,33	55,9	55,9	44,1	1,35	0,69	41,4	53,5
-850µm+600µm	74,39	7,6	63,5	36,5	3,02	0,69	12,6	7,3
-600µm+425µm	49,86	5,1	68,5	31,5	2,41	0,79	6,7	5,6
-425µm+300µm	46,86	4,8	73,3	26,7	1,45	0,94	3,8	6,2
-300µm+212µm	36,43	3,7	77,0	23,0	2,02	0,94	4,1	4,8
-212µm+150µm	28,43	2,9	79,9	20,1	2,58	1,03	4,1	4,1
-150µm+106µm	22,27	2,3	82,2	17,8	2,98	1,38	3,7	4,3
-106µm+75µm	17,86	1,8	84,0	16,0	4,69	1,65	4,7	4,2
-75µm+53µm	11,25	1,1	85,2	14,8	5,41	1,24	3,4	2,0
-53µm	145,51	14,8	100	0,0	1,91	0,39	15,5	8,0
Head calc	981,2	100			1,82	0,72	100	100
Head assays					1,88	0,80		
Variance (%)					3,1%	10,9%		

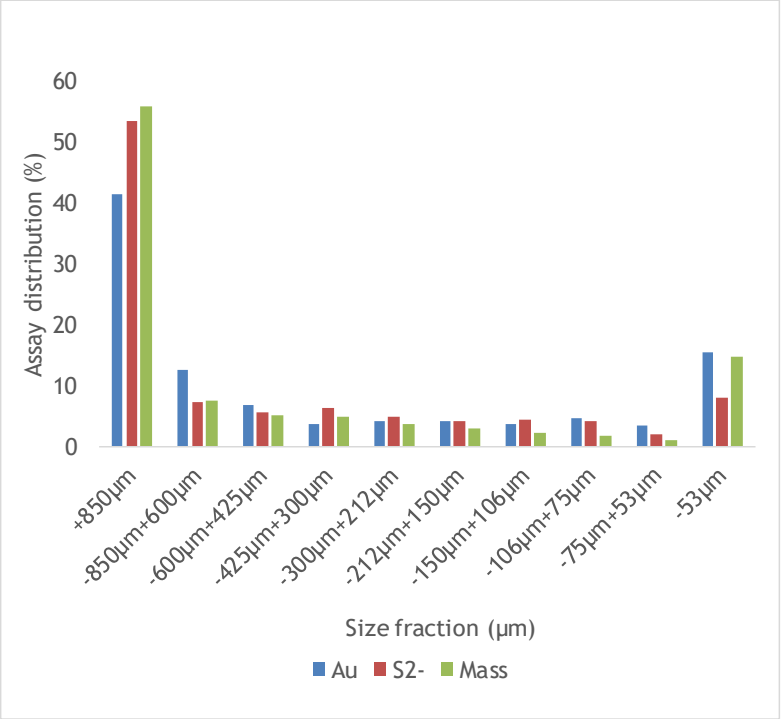


Figure 7: Distribution of ore, Au and S^{2-} particles by size – Volcanoclastics

Again the distribution of Au and S^{2-} was mainly influenced by mass and not grade with the largest proportion contained within the +850µm fraction which accounted for ~41.4% and ~55.9% of the total Au and feed mass respectively. The amount of Au and

S^{2-} progressively decreased with decrease in mass again effectively showing that Au and S^{2-} coexisted and their distributions were both influenced by mass.

3.4.3.3 Distribution of ore, Au and S^{2-} particles by size – Granitic sample

The distribution of ore, Au and S^{2-} particles by size within the Granitic samples are presented in Table 10; Figure 8 illustrates the bar graph developed from these results.

Table 10: Distribution of ore, Au and S^{2-} particles by size – Granitic sample

Particle size range	Mass distribution				Fraction assays		Distribution	
	(g)	(%)	Cum (%+)	Cum (%-)	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-}(%)
+850µm	566,07	57,6	57,6	42,4	0,80	0,69	44,3	47,9
-850µm+600µm	78,23	8,0	65,6	34,4	0,76	0,59	5,8	5,7
-600µm+425µm	49,74	5,1	70,7	29,3	0,87	0,79	4,2	4,8
-425µm+300µm	49,22	5,0	75,7	24,3	1,12	1,14	5,4	6,9
-300µm+212µm	40,15	4,1	79,8	20,2	2,04	1,15	8,0	5,7
-212µm+150µm	33,02	3,4	83,1	16,9	1,46	1,69	4,7	6,8
-150µm+106µm	27,90	2,8	86,0	14,0	2,00	1,50	5,5	5,1
-106µm+75µm	23,37	2,4	88,3	11,7	1,68	1,54	3,8	4,4
-75µm+53µm	15,93	1,6	90,0	10,0	1,75	1,53	2,7	3,0
-53µm	98,57	10,0	100	0,0	1,60	0,80	15,4	9,7
Head calc	982,2	100			1,04	0,83	100	100
Head assays					0,88	0,80		
Variance (%)					15,1%	3,6%		

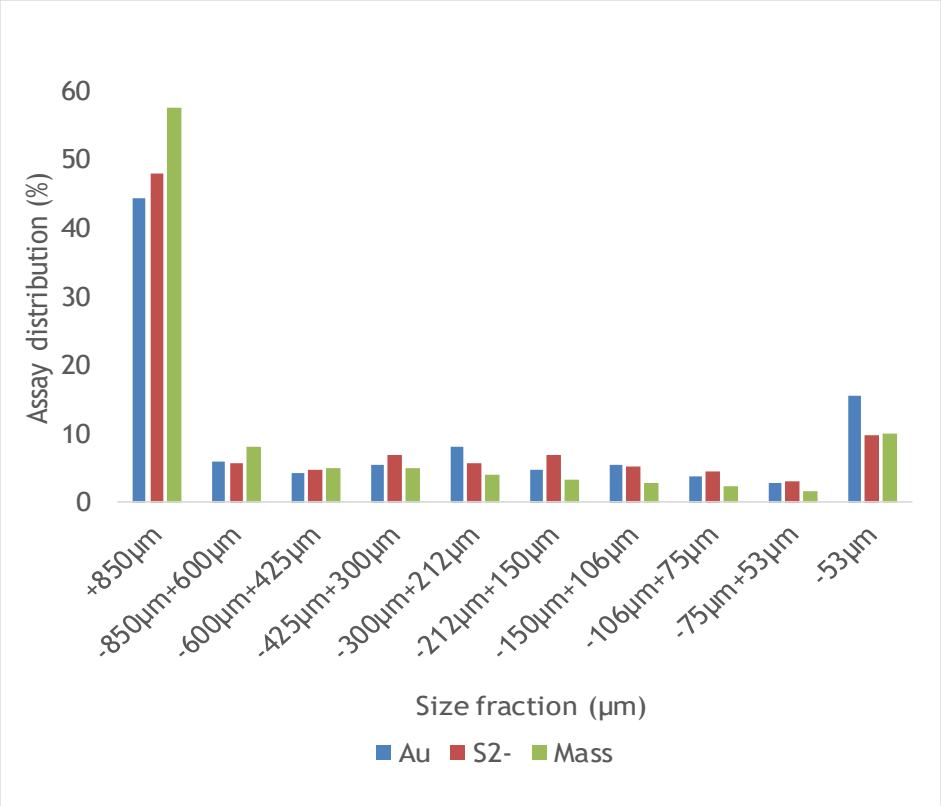


Figure 8: Distribution of ore, Au and S^{2-} particles by size – Granitic sample

The largest proportion of Au found within the Granitic sample was contained within the +850µm size fraction which accounted for ~44.3% and ~57.6% of the total Au and feed mass respectively. The coarsest fraction (+850µm fraction) contained the highest quantity of Au because it accounted for the highest mass fraction. The finest size fraction -53µm contained 9.7% and 10.0% of the total Au and feed mass respectively.

3.4.3.4 Distribution of ore, Au and S^{2-} particles by size - Diorites sample

The distribution of ore, Au and S^{2-} particles by size for the Diorites samples are presented in Table 11; Figure 9 illustrates the bar graph developed from these results.

Table 11: Distribution of ore, Au and S^{2-} particles by size – Diorites sample

Particle size range	Mass distribution				Fraction assays		Distribution	
	(g)	(%)	Cum (%+)	Cum (%-)	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-}(%)
+850µm	523,40	53,1	53,1	46,9	1,14	0,98	38,2	41,5
-850µm+600µm	80,36	8,2	61,3	38,7	1,23	0,66	6,3	4,3
-600µm+425µm	53,88	5,5	66,8	33,2	2,40	0,86	8,3	3,7
-425µm+300µm	50,54	5,1	71,9	28,1	3,40	1,22	11,0	5,0
-300µm+212µm	38,23	3,9	75,8	24,2	2,35	2,96	5,7	9,2
-212µm+150µm	30,28	3,1	78,9	21,1	3,18	3,84	6,2	9,4
-150µm+106µm	23,19	2,4	81,2	18,8	3,24	1,88	4,8	3,5
-106µm+75µm	17,93	1,8	83,0	17,0	3,61	4,41	4,1	6,4
-75µm+53µm	14,17	1,4	84,5	15,5	3,61	3,28	3,3	3,8
-53µm	152,94	15,5	100	0,0	1,24	1,07	12,1	13,2
Head calc	984,9	100			1,59	1,26	100	100
Head assays					1,59	2,10		
Variance (%)					0,2%	67,3%		

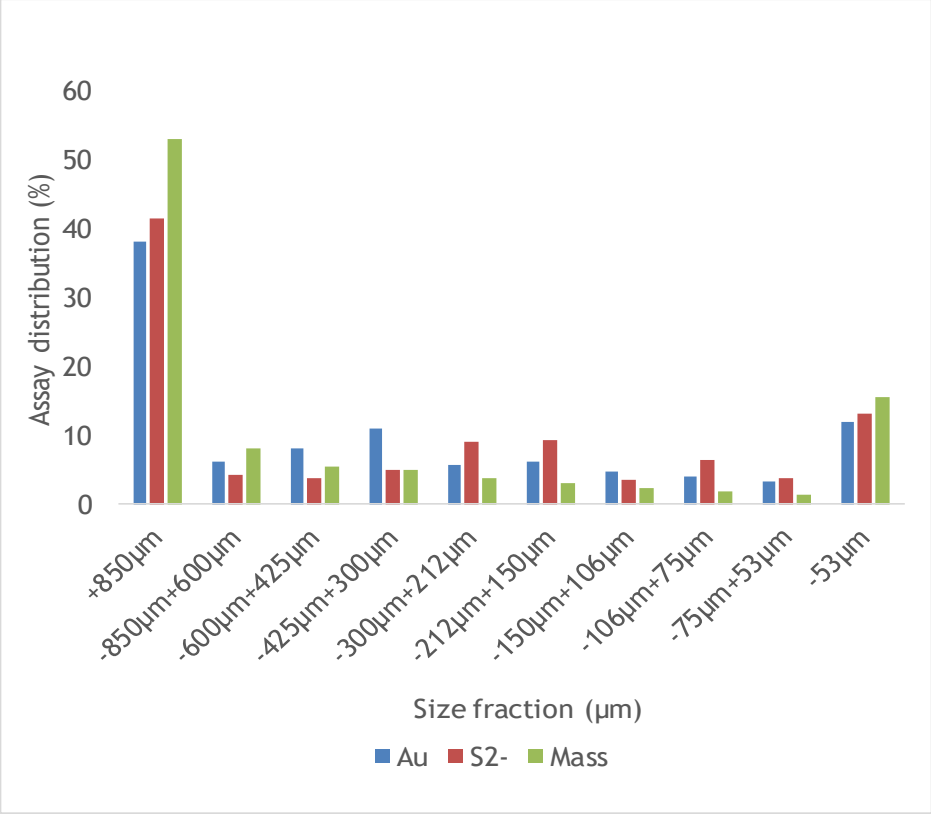


Figure 9: Distribution of ore, Au and S^{2-} particles by size – Diorites sample

Again, the distribution of ore, Au and S^{2-} was mainly influenced by mass rather than grade with the largest proportion of Au and S found within the +850µm size fraction

which contained ~38.2% and 41.5% of the total Au and S respectively recovered into corresponding the feed mass of ~53.1%.

3.4.1.4. Ore S.G of the Namdini Master composite (MC)

The ore S.G. of the Namdini composite was established via the Suntech standard S.G. determination procedure. A representative 1kg aliquot of the MC sample was accurately weighed after drying for 24 hours at 105°C. This sample was thoroughly mixed with 500ml of tap water at ambient temperature (24°C). The pulp was transferred into a graduated measuring glass cylinder and an additional 500ml of tap water added cleaning all material sticking onto the sides of the cylinder. The contents were allowed to stand for 24 hours in a cool dry place to ensure that no evaporation took place. The final water level after 24 hours was noted and used to calculate the volume of the solids and to determine the ore SG as follows:

Dry ore mass (g)	1000g,
Volume of water	1000ml,
Final volume of mixture	1365ml
Volume or ore	Total mixture volume - Volume of water (*x)*
	1365ml-1000ml=365ml
Ore S.G	Mass of ore/Volume of ore
	= 1000g/365ml
	= <u>2.739g/ml > 2.734tons/m^3</u>

3.4.1.5. Ore Moisture content

A known mass of the MC sample was placed in a laboratory oven at 105°C over a period of 24 hours and the final mass recorded. The moisture content was determined by measuring the sample mass before and after drying; it was assumed that all loose water (moisture) was removed by evaporation. The moisture content was then calculated using the formula:

$\%\ \text{Moisture} = (\text{Mass}_{\text{initial}} - \text{Mass}_{\text{dried}}) * 100 / \text{Mass}_{\text{initial}}$

= (100-99.9)/100=0.1%

It was established that the Namdini MC sample contained approximately 0.1% moisture.

3.4.1.6. Mineralogical characterisation results

Mineralogical characterisation was carried out by XRD and QEMScan analyses on representative aliquots of the master composite (ROM), sulphide concentrate and the cyanidation tails of the Ultrafine ground cleaner concentrate. The investigations were done to establish the mode of occurrence of gold-bearing minerals, mineral associations, liberation and grain size distributions; results of these analyses would inform the optimisation of the flowsheet. It was envisioned to:

- ✓ Identify the minerals present and to obtain an estimate of the modal mineral assemblage by X-Ray diffraction (XRD) of the samples,
- ✓ Determine the modal abundance of minerals, mineral association and the liberation characteristics of the gold and sulphides by QEMScan analyses on polished sections of the bulk rock and the flotation concentrate and
- ✓ To obtain data on Au occurrences in order to determine the Au speciation, Au size distribution and Au association as well as liberation by a QEMScan Au search on polished sections of each sample.

3.4.6.1 Mineralogy results on ROM and concentrate

Representative aliquots of each sample were rotary split (ROM sample was milled to 80%-425µm) and submitted to Mintek Laboratories for mineralogical characterisation. A detailed report of the procedures and results is appended to this report while the summarised results are presented in the following sections.

3.4.6.1.1 Relative gold mineral abundance & deportments

Table 12 presents the Au modal mineralogy established for the Namdini Master composite feed sample and the flotation concentrates.

Table 12: Abundance of Au species & their deportment in feed and concentrates

Sample	Mineral	Formula	Area (%)	Grain count	Au deportment (Wt %)	Theoretical Au content (Wt %)
Feed	Gold	Au	18.9%	13	39.6%	100%
	Electrum	AuAg	81.1%	70	60.4%	79%
	Total		**100%**	**83**	**100%**	
Concentrate	Gold	Au	14.2%	101	17.4%	100%
	Electrum	AuAg	85.8%	816	82.6%	79%
	Total			917	100%	

The QEMScan search picked up in total about 83 and 917 Au grains from the polished sections processed for the feed and concentrate samples respectively. The major contributor of gold by weight in both samples was electrum with a contribution of ~60.5% and ~82.6% respectively for the feed and concentrates while the corresponding balances of ~39.6% and ~17.4% existed as native gold.

3.4.6.1.2 Liberation of Au species

Table 13 presents the Au liberation data collected for the feed and sulphide concentrates.

Table 13: Au liberation data for the Namdini feed and sulphide concentrate

Sample	Mineral	Locked	Partially exposed	Liberated	Total
ROM	Gold	96,0	0,0	4,0	100
Concentrate	Gold	31,8	40,8	27,5	100

Approximately 96.0% and ~31.8% of the Au contained within the feed and concentrate samples was locked (completely surrounded by another mineral) while none of the Au identified in the feed was partially exposed. Milling the feed sample to 80%-75µm resulted in significant liberation of Au from the sulphides as evidenced by ~40.8% of

the Au in the concentrate partially exposed. Both samples contained significantly low quantities of liberated Au with only ~4.0% and ~27.5% of the Au contained within the feed and concentrates totally liberated. These results confirmed why it was not possible to achieve Au leach dissolutions exceeding 69%.

3.4.6.1.3 Distribution of gold mineral by size

NB – Mineral grain sizes are reported in terms of an Equivalent Circle Diameter (ECD), which is the diameter of a circle of equivalent area to that of the grain; the size distribution is reported in area percent of gold in each size class. The distribution of Au grains by size on the Namdini ROM and concentrate samples are presented in Table 14 while detailed results are presented in the appendix in Table 41 and Table 42 respectively.

Table 14: Gold mineral size distribution in the feed & concentrate samples

Size class (µm)	Percent area (%)	
	Feed	Concentrate
0µm to 2µm	11.6%	25.9%
2µm to 5µm	28.4%	24.2%
5µm to 10µm	48.7%	27.9%
10µm to 16µm	11.3%	22.0%
Total	100%	100%

***NB** – Because of the low number of grains of the lesser species, the influence of statistics on data is large and one large grain may skew the data and should be interpreted with caution.*

The bulk of the Au grains contained within the ROM and concentrate sample were generally fine, with approximately ~88.7% and ~78.0% of the total Au contained within the feed and concentrates respectively being less than 10µm. These results explain why there was still significant quantities of Au locked in sulphides when the sample was milled to 80%-75µm.

3.4.6.1.4 Distribution of pyrite mineral by size

The distributions of pyrite particles by size in the Namdini feed and concentrate samples are presented in Table 15.

Table 15: Pyrite mineral distribution in the feed and concentrate

Size class (µm)	Percent area (%)	
	Feed	Concentrate
0µm to 20µm	4.08%	23.99%
20µm to 40µm	10.57%	29.27%
40µm to 60µm	13.54%	18.17%
60µm to 80µm	19.43%	17.55%
80µm to 100µm	17.56%	6.22%
100µm to 120µm	17.13%	2.95%
120µm to 140µm	10.40%	1.34%
140µm to 160µm	4.18%	0.51%
160µm to 180µm	1.51%	0.00%
180µm to 200µm	1.60%	0.00%
Total	100%	100%

The majority of the pyrite particles (~88.63%) occurred within the 20 ECD(µm) to 140 ECD(µm) size class for the feed sample while those found within the concentrate sample occurred mostly between 60ECD(µm) to 80ECD(µm). By extrapolation, one can see that the distribution of pyrite within the flotation concentrate almost confirms that pyrite was ground to 80%-75µm as the rest of the ore particles. With only 23.99% of the pyrite being less than 20ECD(µm), it is confirmed that a significant proportion of the Au was still locked within the sulphide particles and not available solubilisation by cyanidation.

3.4.6.1.5 Gold mineral association

NB- Minerals are considered as associated with Au when they share grain boundaries with the Au minerals (*the greater the shared grain boundary, the higher the degree of association*) while free surface indicates that the minerals have exposed surfaces, i.e. no other minerals are attached at those surfaces.Table 16 presents the results of the association of Au with the minerals making up the ROM and sulphide concentrates.

Table 16: Association of the Au with host minerals

		Area (%)	
Item	**Mineral**	**ROM**	**Conc**
1	Gold	0,0	0,0
2	Uraninite	0,1	0,0
3	Pyrite	75,4	73,8
4	Arsenopyrite	8,1	1,5
5	Chalcopyrite	2,3	0,0
6	Galena	0,6	0,0
7	Other BMS	0,2	0,7
8	Feldspar	0,0	0,0
9	Quartz	5,8	1,0
10	Oxides	0,0	0,0
11	Clay	0,0	0,0
12	Mica	0,3	0,0
13	Silicates	0,0	1,9
14	Fe-Oxides	0,0	0,3
15	Dolomite	1,4	0,3
16	Other	0,2	0,1
17	Free Surface	5,6	20,3
Total		**100**	**100**

The bulk of the Au found within the Namdini samples was mostly associated with pyrite; this mineral contained ~75.4% and 73.8% of the total Au found within the feed and concentrate samples respectively. For the feed sample, arsenopyrite, quartz and free surface Au accounted for ~8.1%, 5.8% and 5.6% respectively; the corresponding quantities of Au found in these minerals recovered into the sulphide concentrates were 1.5%, 1.0% and 20.3%. The increase in relative quantities of free surface Au from ~5.6% in feed to ~20.3% in the flotation concentrate confirms that milling the ore sample to 80%-75µm resulted in liberation of Au. From the results presented in Table 16, it will be reasonable to extrapolate that the bulk of the free surface Au found within the concentrate originated from mostly from the base metal sulphides (arsenopyrite, chalcopyrite and galena), quartz, dolomite as well as the free surface Au that originally existed within the feed sample.

3.4.1.7. Comminution test work results

The comminution test work carried out during this campaign was of limited scope and restricted to SAG Power Index (SPi)/Mod Bond tests. This work was done in order to gain an initial understanding of the comminution characteristics of the orebody. A total of 3 SPi/paired Mod Bond tests were completed, one each on representative material of 3 key mineralised lithology (V, G & D) and a single test on the composite sample.

Samples were split as representatively as possible with intact quarter core lengths separately crushed to 100%-12mm, thoroughly blended and ~10kg of each lithology split out for the SPi tests. The lithology samples were then further stage crushed to 100%-3.35mm and composited in the appropriate ratios to make up the master composite. The master composite sample was thoroughly blended and ~10kg split from out and submitted for full Bond BM Work Index (BBWI) test as well.

3.4.1.8. SPi test results

The materials for the SPi tests were crushed systematically to 100%-12mm and handed over to SGS South Africa for testing. Results from these tests can be used to predict throughputs for SAG/AG mills and in the determination of the power requirements. Results obtained at SGS indicated that the power draw for the V, G, D and master composite samples ranged from 8.83 to 9.57 KWh/t; these results were within the expected range for typical gold ores (see SGS report attached for details). The G sample (Power draw ~9.57 kWh/t) was the hardest ore amongst those tested.

3.4.1.9. Bond Ball Work Index (BBWI) test results

A full Bond BM Work Index (BBWI) test was completed on the master composite at SGS; this test determines the net power requirements for the sizing of Ball mills. Results obtained indicated that the Bond ball work index for the master composite was 14.9 kWh/t; at this index the Namdini ore was classified as being hard (See SGS reported appended for detailed procedures and results).

3.4.1.10. Milling curve calibration

In order to determine the time required to mill representative 1kg sample portions of the Namdini Master composite to the target grinds of 80% passing;

- ✓ 212µm,
- ✓ 150µm,
- ✓ 106µm,
- ✓ 75µm and
- ✓ 53µm,

representative aliquots of the of the MC sample were milled at 50% solids in a laboratory scale rod mill for 4 different time periods and the milled samples screened through the requisite screens. Both the oversize and undersize screening products were filtered, dried and further dry screened through the same screens; the data generated was recorded and used to plot graphs of % passing certain size vs. time. The milling times required to mill a 1kg sample to various grinds are presented in Table 17; individual milling curves are illustrated in Figure 20 to Figure 24 in the appendix.

Table 17: Milling times required to achieve requisite grinds on Namdini composite

Item	Grind	Milling Time
1	80%-53µm	16min 12 sec
2	80%-75µm	11min 24 sec
3	80%-106µm	9min 12 sec
4	80%-150µm	7min 24 sec
5	80%-212µm	5min 30 sec
6	80%-425µm	3min 36 sec

Based on the Suntech data base of previously milled ores, the milling times required to mill the Namdini master composite to different grinds indicated that this sample was generally hard to mill.

3.4.1.11. Diagnostic leach test results

A single diagnostic leach test was completed on each of the master composite feed and final float concentrate in order to infer the deportment of gold within the mineral phases making up these samples and the amenability thereof to beneficiation by various known unit processes. Each test was completed using a 2kg /1kg sample portion by sequential solubilisation of the least stable mineral phases followed by conventional cyanidation; the summarised diagnostic leach procedure is presented in the Appendix while the detailed procedure is available on request. These tests were completed on samples milled to 80%-75µm. All leach products were pulverised and analysed for Au only by fire assay, lead collection with ICP finish.

3.4.11.1 Diagnostic leach results for feed composite sample

The diagnostic leach results obtained for the master composite are presented in Table 18.

Table 18: Diagnostic leach results for the feed master composite

Item	Au Association/Solubilisation process	Mass Dist (g)	Au tails (g/t)		Gold Dist (%)	
			Act	Normalised	Rec %	Dissolved Au g/t
1	Free milling Au (leached by direct cyanidation (CIP): No carbon)	250	0,55	0,55	61,3%	0,87
2	Total free milling + Preg-robbed Au (leached in presence of C (CIL)	1960,1	0,51	0,51	64,1%	0,91
3	Au extrcated via mild oxidative pre-leach i.e. Au associated with pyrrhotite, calcite, dolomite & haematite)	1339,86	0,58	0,40	8,0%	0,11
4	Au exctracted via severe oxidative pre-leach i.e. Au associated with pyrite, arsenopyrite etc	848,8	0,02	0,01	27,3%	0,39
5	Au extracted via complete oxidation i.e. Au associated with kerogen	823,3	0,02	0,01	0,0%	0,00
6	Undissolved gold (Au assumed to be associated with Quartz and cannot be dissolved)	823,3	0,02	0,01	0,6%	0,01
7	Total		1,42		100%	1,42

Approximately ~61.3% of the total gold was leached by direct cyanidation without carbon while an additional 2.8% was further leached in the presence of carbon indicating that this Au was preg-robbed. The total Au leached by direct cyanidation exists as free gold. An additional 8.0% of the total Au became soluble after treatment of the direct cyanidation tails via mild oxidative pre-leach; this Au was assumed to be associated with pyrrhotite, calcite, dolomite and hematite. An additional 27.3% of the Au became soluble after severe oxidative pre-leach (achieved by treatment with HNO_3) indicated that a substantial portion of the gold was locked up in pyrites and/or arsenopyrites. None of the Au was extracted via complete oxidation by roasting (this Au is normally assumed to be associated with fine carbon and/or kerogen). The final residue containing 0.6% of the Au was presumed to be associated with quartz and analysed ~0.02g/t. HF leach was not carried out to solubilise this portion of the Au.

3.4.11.2 Diagnostic leach results on flotation concentrates

The Diagnostic leach results obtained on the flotation concentrates are presented in Table **19**.

Table 19: Diagnostic leach results – Flotation concentrates

Item	Au Association/Solubilisation process	Mass Dist (g)	Au tails (g/t)		Gold Dist (%)	
			Act	Normalised	Rec %	Dissolved Au g/t
1	Free milling Au (leached by direct cyanidation (CIP): No carbon)	250	15,80	15,80	66,7%	31,70
2	Total free milling + Preg-robbed Au (leached in presence of C (CIL)	987,2	15,30	15,30	67,8%	32,20
3	Au extrcated via mild oxidative pre-leach i.e. Au associated with pyrrhotite, calcite, dolomite & haematite)	866,2	15,80	13,86	3,0%	1,44
4	Au exctracted via severe oxidative pre-leach i.e. Au associated with pyrite, arsenopyrite etc	632,9	7,53	4,83	19,0%	9,04
5	Au extracted via complete oxidation i.e. Au associated with kerogen	602,7	6,79	4,15	1,4%	0,68
6	Undissolved gold (Au assumed to be associated with Quartz and cannot be dissolved)	602,7	4,15	4,15	8,7%	4,15
7	Total		47,50		100%	47,5

Approximately 66.7% of the total gold was leached by direct cyanidation without carbon while ~67.8% was leached by direct cyanidation with carbon; by calculation, ~1.1% of the total Au was preg-robbed. The Au leached by direct cyanidation was assumed to exist as liberated gold. An extra 3.0% of the Au became soluble after treatment via mild oxidative pre-leach; the additional leached Au was assumed to be associated with pyrrhotite, calcite, dolomite and hematite. An additional 19.0% of the Au became soluble after pre-treatment with HNO_3 which indicated that this portion of gold was locked up in pyrites and/or arsenopyrites. A further 1.4% of the Au was extracted via complete oxidation by roasting; this Au was assumed to be associated with fine carbon. The final residue containing ~8.7% of the Au was presumed to be associated with quartz. A HF leach was not carried out to solubilise this portion of the Au.

3.5. GOLD DISSOLUTION TEST RESULTS

Gold dissolution tests were carried out on representative portions of the master composite, individual lithology and flotation concentrate samples by direct cyanidation. The effects of different sets of conditions on the overall dissolution of Au were investigated; all tests were done in duplicate. The conditions used for gold dissolution were;

- ✓ Carbon concentration – 20g/l,
- ✓ Solids density – 50% solids (w/w),
- ✓ Dissolution period – 24 hours,
- ✓ pH of between 10.5 and 11 and adjusted using lime and
- ✓ NaCN – 2kg/t (At start of test) & 5kg/t for intensive cyanidation

All leach products were analysed for Au only; Au analysis was done by fire assay, lead collection with ICP finish.

3.5.1.1. Grind optimisation bottle roll cyanidation results – Master composite

In order to establish the effect of grind on Au dissolution on the master composite, 4 Au dissolution tests were completed at the different grinds of 50%, 60%, 70% and 80%-75µm while an additional 5 Au dissolution tests were carried out on samples milled to 80%-212µm, 150µm, 106µm, 75µm and 53µm. The results obtained are presented in Table 20.

Table 20: Grind optimisation leach results – master composite

Item	Grind	Leach head, Au g/t	Leach tails, Au g/t	Au Recovery, %
1	80%-53µm	1,42	0,52	63,4
2	50%-75µm	1,42	0,69	51,4
3	60%-75µm	1,42	0,68	52,1
4	70%-75µm	1,42	0,65	54,2
5	80%-75µm	1,42	0,51	64,1
6	80%-106µm	1,42	0,62	56,3
7	80%-150µm	1,42	0,66	53,5
8	80%-150µm	1,42	0,66	53,5
9	80%-212µm	1,42	0,85	40,1
	QAQC results			

Gold dissolution steadily rose with increasing grind reaching a maximum of ~64.1% at the grind of 80%-75µm. Grinding finer than 80%-75µm did not prove beneficial as a similar amount of Au was solubilised (~63.4%) at the finer grind of 80%-53µm. With the coarsest Au grain identified by QEMScan analysis being less than 16µm ECD and with ~32.1% of the Au particles identified being less than 3.4µm, significant liberation

of Au may only be possible after milling finer than say 3µm where significant gains on Au dissolutions are expected. However, it is not economically viable to mill the ore sample to this level of fineness using conventional milling processes thus milling the ore finer than 80%-75µm is not recommended.

3.5.1.2. Effect of leach enhancers on Au dissolution

The effect of different leach enhancers on Au dissolution was tested on the master composite; the leach enhancers tested comprised of;

- ✓ $PbNO_3$,
- ✓ H_2O_2,
- ✓ Air injection and
- ✓ Pure oxygen

Results obtained are presented in Table 21.

Table 21: Effect of leach enhancers

Test/ leach enhencer	Baseline	$PbNO_3$, 0,4kg/t	$PbNO_3$, 1,2kg/t	$PbNO_3$, 1,6kg/t	H_2O_2	Air injection	Pure O_2
Head assay value, g/t	1,42	1,42	1,42	1,42	1,42	1,42	1,42
Leach residue, g/t	0,51	0,52	0,50	0,48	0,46	0,51	0,43
Au Rec, %	64,1%	63,4%	64,8%	66,2%	67,6%	64,1%	69,7%

Inclusion of leach enhancers generally resulted in increased Au dissolution with pure oxygen achieving the highest leach recovery of ~69.7%; this amounted to an increase in Au leach recovery of ~5.6%. H_2O_2 and $PbNO_3$ also enhanced Au dissolution with additional 3.5% and 2.1% Au leached respectively. Air injection did not have an impact on Au dissolution when compared to the baseline conditions.

3.5.1.3. Preg-robbing test results

A single pre-robbing test was carried out on a representative aliquot of the master composite in order to establish if the Namdini sample exhibited preg-robbing

characteristics. Results obtained indicated that preg-robbing was not severe with ~2.8% of the total Au was preg-robbed.

3.5.1.4. Kinetic Au dissolution test results

A single gold dissolution kinetic test was carried out on a representative portion of the master composite sample in order to establish the baseline Au leaching kinetics for comparison with a test incorporating a leach accelerant ($PbNO_3$). Leaching was carried out for 48 hours with pulp samples drawn at intervals of 0, 1, 2, 4, 8, 16, 24, 36 and 48 hours of cumulative leaching time. The results obtained are presented in Table 22 while Figure 10 illustrates the curves obtained from the data.

Table 22: Kinetic Au dissolution test results

Leach time, hrs		0	1	2	4	8	16	24	36	48
Head assay value, g/t		1,42	1,42	1,42	1,42	1,42	1,42	1,42	1,42	1,42
Baseline	Leach residue, g/t	1,42	0,84	0,76	0,61	0,59	0,53	0,51	0,50	0,49
	Au Rec, %	0,0	40,8	46,5	57,0	58,5	62,7	64,1	64,8	65,5
$PbNO_3$	Leach residue, g/t	1,42	0,62	0,60	0,58	0,55	0,5	0,48	0,45	0,44
	Au Rec, %	0,0	56,3	57,7	59,2	61,3	64,8	66,2	68,3	69,0

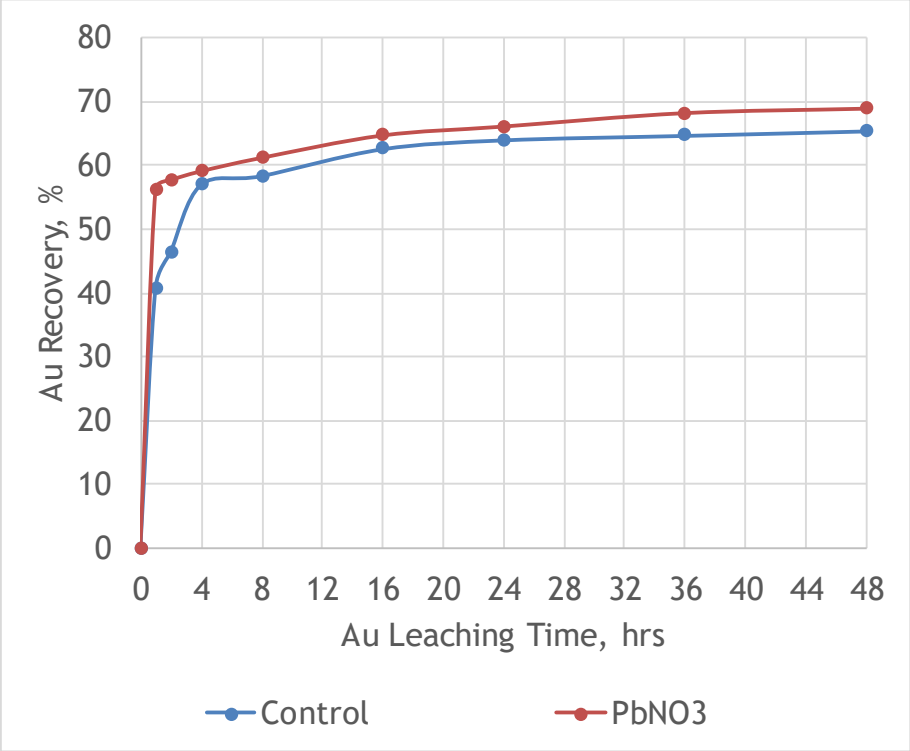


Figure 10: Effect of leach time/ kinetic dissolution test results

Use of a leach accelerant ($PbNO_3$) resulted in improved Au dissolution kinetics and overall Au recovery. The impact of the accelerant was noticed within the first hour of cyanidation wherein ~56.3% and 40.8% of the total Au was solubilised with and without the accelerant respectively. Overall, the accelerant resulted in an additional 3.5% of gold leached. These results indicated that use of the accelerant was beneficial both in terms of recovery kinetics and overall Au recovery.

3.5.1.5. Variability Au dissolution test results

The extent of variability in Au dissolution exhibited across the 3 lithology/variability Namdini samples was tested through Au dissolution tests on the individual 3 samples. Cyanidation was done using conventional cyanidation conditions on samples milled to 80%-75µm. The results obtained are presented in Table 23.

Table 23: Variability cyanidation results on Namdini lithology samples

	Volcanoclastics		Granitic		Diorites	
Grind	89,1%-75µm		66,3%-75µm		87,5%-75µm	
Leach time, hours	24	48	24	48	24	48
Head assay value, g/t	1,88	1,88	0,88	0,88	1,59	1,59
Leach residue, g/t	0,69	0,68	0,47	0,45	0,53	0,47
Au Rec, %	63,3%	63,8%	46,8%	49,1%	66,7%	70,4%

There was wide variability in leach recoveries achieved across the 3 lithology samples with Au dissolutions ranging from 46.8% to 66.7% for a 24 hour leach period; leaching for an additional 24 hours resulted in additional leach recoveries of ~0.5%, ~2.3% and ~3.7% for the V, G and D samples respectively. The G sample performed worst when compared to the other 2 lithology samples with Au dissolutions of ~46,8% and 49.1% achieved after 24 and 48 hours of cumulative leaching time. It was envisioned that this was caused by poor grind of ~66.3%-75µm compared to the grinds of ~89.1% and ~87.5%-75µm achieved for the V & D samples respectively.

3.5.1.6. Cyanide and lime consumptions

The consumption of cyanide and lime per ton of ore samples processed was determined through the standard titration methods. Lime addition was determined by running a control experiment wherein lime was incrementally added into an agitated leach tank containing pulp and pH monitored until it remained above 10 consistently. From these tests, it was established that ~0.15g of lime was required per 250g of ore samples pulped at 50% solids in order to increase and maintain pH at the required levels for optimal Au cyanidation. The lime and cyanide consumptions obtained during testing of various scenarios are presented in Table 24

Table 24: Reagent consumption results for the cyanidation tests on Namdini samples

		Reagent dosages	
Item	**Sample description**	**NaCN (kg/t)**	**Lime (kg/t)**
1	Master composite	1,55	0,53
2	Volcanoclastics	1,60	0,26
3	Granitic	1,35	0,29
4	Diorites	1,60	0,42
Average		**1,53**	**0,38**

Lime consumption across the 3 lithology samples and the Master composite ranged between 0.26kg/t and 0.53kg/t for all conditions tested. Cyanide consumptions were high ranging between 1.35kg/t and 1.60kg/t. It was established that these samples may have contained cyanicides that consumed cyanide during cyanidation. It was also projected that a substantial amount of the Cu found within the feed sample was easily cyanide soluble. Further investigations are recommended to establish the source of cyanicides.

3.6. FLOTATION TEST RESULTS: MASTER COMPOSITE

Flotation test work was carried out on representative aliquots of the Namdini MC samples in order to test suitability of flotation in pre-concentrating the Au found in the samples prior to cyanidation and to select the best reagent suite for optimum recovery and upgrading of gold into a concentrate suitable for further processing. In total, 15 reagent scouting flotation tests were carried out on representative 1kg

sample portions in a 2.5L Denver flotation cell agitated by a D12 Denver flotation machine operating at a rotational impeller speed of 1200 rpm. All samples were milled to a grind of 80%-75µm. The robustness of combination of reagent suites including collectors, frother and activators were tested using a rougher cleaner recleaner circuit configuration illustrated in Figure 11.

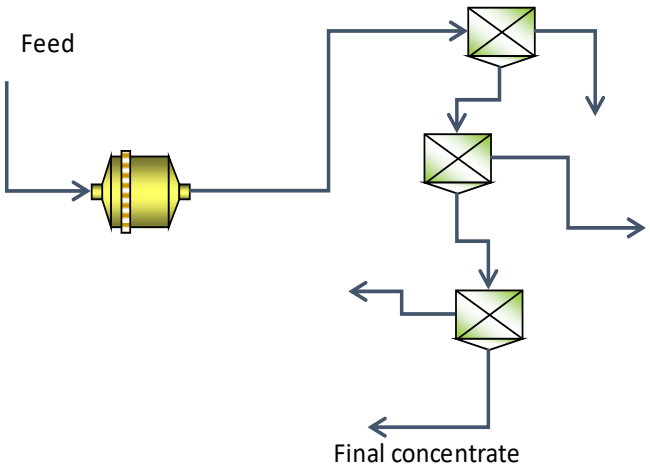


Figure 11: Schematic flotation diagram – Reagent scouting

The flotation conditions tested are summarised in Table 25.

Table 25: Summarised flotation conditions tested during reagent scouting phase

Test No	Reagent dosages (g/t)										
	$CuSO_4$	PAX	SEX	SIBX	Betacol 381	Betacol 316A	Maxgold 900	Aero 3418A	Aero 8045	Betafroth 436	XP200
1			200								40
2					150						40
3		150									40
4	40		100							20	
5						15	80			20	
6							80				40
7		50						10			40
8									80		40
9	40			150							40
10						15		80		20	
11	40	50						10			40
12	40	30						10			40
13	40	50							40		40
14	40			50				10			40
15	40			50					40		40

The flotation products generated were filtered, dried, pulverised and analysed for Au and S^{2-}. The results obtained are presented in Table 26 and further illustrated in Table 43 to Table 57 in the appendix.

Table 26: Reagent scouting test results – Namdini Master composite

Test	Rougher Conc			Cleaner Conc			ReCleaner Conc		
No.	Mass Pull, %	Grade, g/t	Rec, %	Mass Pull, %	Grade, g/t	Rec, %	Mass Pull, %	Grade, g/t	Rec, %
1	8,5	14,92	95,9	2,8	42,00	87,5			
2	4,4	28,87	89,3	2,4	48,00	81,5			
3	11,5	11,47	89,8	4,3	29,00	84,5			
4	7,5	14,92	87,1	3,0	35,54	83,6	2,2	44,00	76,6
5	22,8	5,84	93,5	9,3	13,93	91,0	3,3	36,00	84,2
6	11,8	11,61	94,0	4,1	32,50	90,2	2,5	49,00	82,6
7	5,4	22,72	94,2	2,2	53,56	89,2	1,7	61,00	77,2
8	5,5	24,80	91,8	2,2	24,80	82,4			
9	9,1	14,95	93,8	3,5	36,89	89,8	2,2	58,00	85,9
10	20,7	7,09	96,3	5,4	23,60	83,8	3,1	39,00	80,1
11	6,4	20,2	99,3	2,6	45,7	90,2	1,8	53,1	74,6
12	8,2	14,9	94,3	3,2	36,1	89,9	2,1	48,5	80,0
13	5,9	21,0	92,3	2,6	44,5	84,6	2,1	50,7	79,0
14	7,3	15,3	92,3	2,5	40,1	84,2	1,8	46,1	69,3
15	7,1	17,7	91,6	2,8	41,9	91,6	2,1	52,3	77,1

Figure 12 illustrates the Grade/Recovery curves developed from the results obtained.

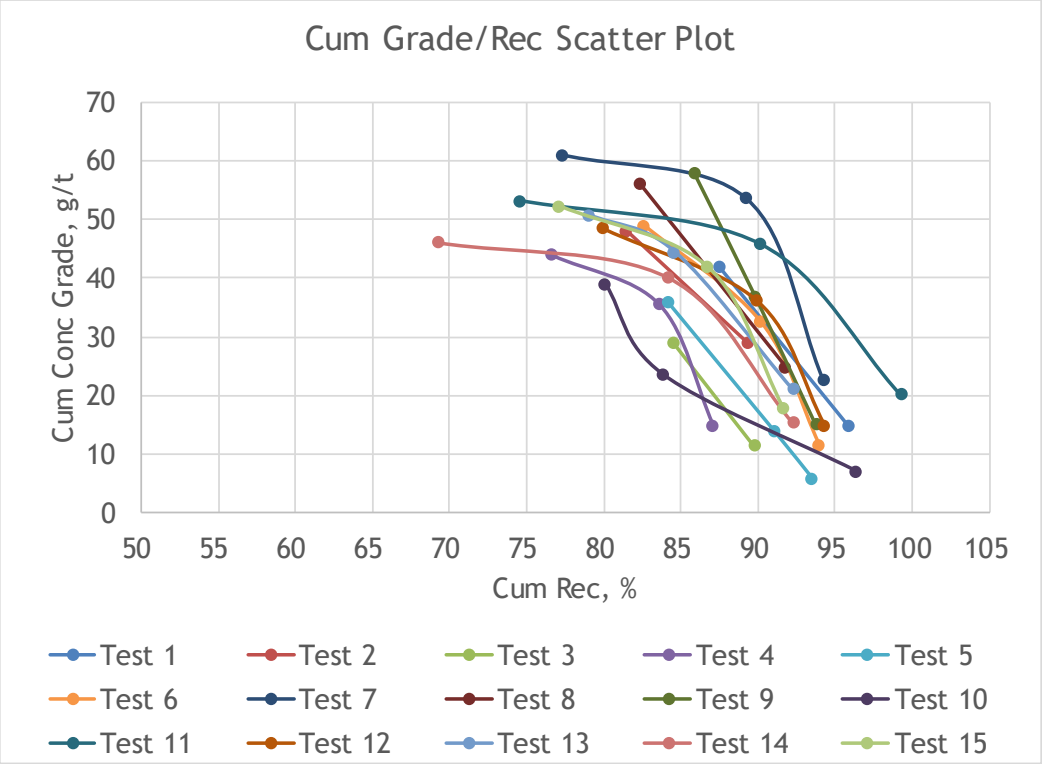


Figure 12: Reagent scouting results for Namdini MC

The Namdini composite sample generally responded well to flotation with high rougher Au recoveries exceeding 90% achieved with most of the flotation conditions tested. The best results were obtained using Test 7 conditions which comprised of 50g/t PAX (Potassium Amyl Xanthate), 10g/t Aero 3148A (promoter), and 40g/t XP200 frother. Using these conditions, ~89.2% of the total Au was recovered into a cleaner concentrate of mass pull 2.2% at a cleaner concentrate grade of 53.56%. Table 27 presents the results obtained using test 7 conditions.

Table 27: Flotation results obtained with Test 7 conditions

Product	Mass Distribution			Fraction assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	16,53	1,7	1,7	61,0	77,2
Cleaner conc	5,22	0,5	2,2	30,0	12,0
Clnr Tails	32,39	3,3	5,4	2,01	5,0
Ro Tail	941,8	94,6	100	0,08	5,8
Head calc	996,0	100		1,31	100
Head Meas	1000			1,42	
Variance				8,3%	

Product	Mass Distribution			Cum fractions assays (g/t)	Cum distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	16,53	1,7	1,7	61,0	77,2
RClnr Tails	5,22	1	2,2	53,6	89,2
Clnr Tails	32,39	3,3	5,4	22,7	94,2
Ro Tail	941,8	94,6	100	1,31	100
Head calc	996,0	100			
Head Meas	1000				
Variance					

It was also noticed that the Au found in the master composite also generally upgraded well with significantly high upgrading ratios achieved even during the rougher stage of flotation. The best conditions produced good frothing characteristics with small and stable bubbles noticed. Further test work was therefore carried out using Test 7 reagent suite.

3.6.1.1. Grind optimisation flotation test results

In order to determine the effect of the level of fineness on the recovery and upgrading of Au by froth flotation, a grind optimisation campaign was carried out on the master composite sample using the best reagent suite obtained during the reagent scouting phase. In total 5 different grinds were tested, and these comprised of;

i. 80%-53µm,
ii. 80%-75µm,
iii. 80%-106µm,
iv. 80%-150µm and

v. 80%-212µm

All flotation tests were carried out on a rougher cleaner recleaner configuration while the flotation products were oven dried, quantified, pulverised and analysed for Au. The results obtained are summarised in Table 28 and further illustrated in Figure 13.

Table 28: Summarised flotation results – Grind Optimisation

		Cleaner concentrate		ReCleaner concentrate	
Item	Grind	Grade, g/t	Rec, %	Grade, g/t	Rec, %
1	80%-53µm	46,0	88,7	57,5	79,0
2	80%-75µm	53,6	89,2	61,0	77,2
3	80%-106µm	43,3	85,3	51,6	72,4
4	80%-150µm	32,4	85,7	40,2	75,7
5	80%-212µm	31,9	77,4	39,7	68,7

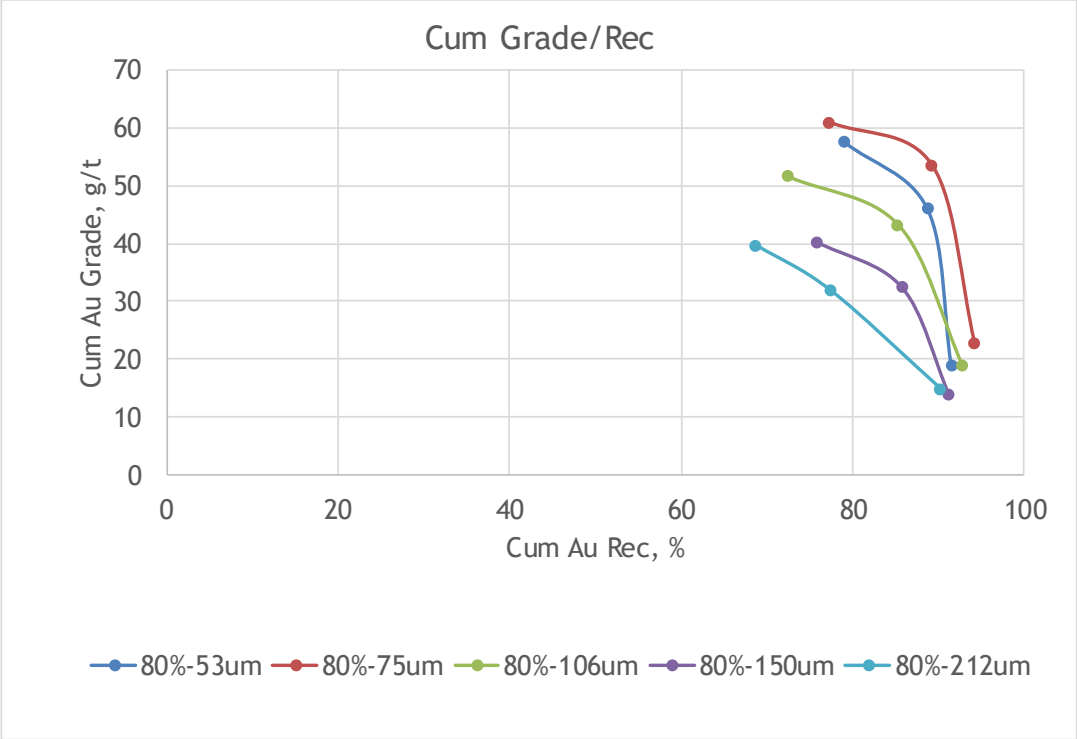


Figure 13: Flotation results – Grind optimisation

The grind of 80%-75µm achieved the best flotation results in terms of Au upgrading and recovery wherein ~89.2% of the total Au was recovered at a cleaner concentrate grade of 53.6g/t. Milling the sample finer to 80%-53µm resulted in decreased flotation recovery and limited Au upgrading. This could have been caused by overgrinding of the sulphide Au carrying particles leading to compromised flotation response. Milling

the sample courser resulted in systematic decrease in flotation response with the coarsest grind producing the worst flotation response both in terms of Au recovery and upgrading.

3.6.1.2. Flash flotation results

A single flash rougher flotation test was completed on a representative 1kg sample portion milled at p80 of 425µm. Rougher flotation was carried out for only 7 minutes (flash float) and a single rougher concentrate collected. Flotation products were pulverised and analysed for Au, S^{2-}, and ICP suite. The results obtained are presented in Table 29.

Table 29: Flash flotation results – Namdini MC

Product	Mass Distribution			Fraction Assays		Distribution	
	(g)	indiv (%)	Cum %	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-} (%)
Ro Conc	45,82	4,6	4,6	24,46	18,40	72,4	86,4
Ro Tail	949,7	95,4	100	0,45	0,14	27,6	13,6
Head calc	995,5	100		1,56	0,98	100	100
Head Meas	1000			1,42	0,99		
Variance				8,7%	1,0%		

Product	Mass Distribution			Cum F.Assays		Cum Distribution	
	(g)	indiv (%)	Cum %	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-} (%)
Ro Conc	45,82	4,6	4,6	24,46	18,40	72,4	86,4
Ro Tail	949,7	95,4	100	1,56	0,98	100	100
Head calc	995,5	100					
Head Meas	2000						
Variance							

The results indicated that it was possible to achieve a rougher concentrate grade of 24.46g/t into a 4.6% mass at a final recovery of 72.4% Au. Approximately 86.4% of the S^{2-} was recovered into the final concentrate at a grade of ~18.40%.

3.6.1.3. Rougher rate flotation results – Master composite

A single rougher rate flotation test was carried out on the master composite sample using the optimum flotation conditions (Test 7). Approximately 1kg sample portion was floated at 80%-75µm in a 2.5L Denver flotation cell agitated by a D12 Denver flotation machine operating at a rotational impeller speed of 1200 rpm. Timed concentrates were collected after 1, 3, 7, 15 and 31 minutes of cumulative flotation time. The flotation products generated were filtered, dried, quantified, pulverised and analysed for Au only using fire assay lead collection with ICP finish. The results obtained are presented in Table 30 and further Illustrated in Figure 14.

Table 30: Rougher rate results – Master composite

Product	Mass distribution			Fraction Assays	Distribution
	(g)	indiv (%)	Cum %	Au (g/t)	Au (%)
Ro Conc 1	22,05	2,2	2,2	46,00	74,0
Ro Conc 2	9,60	1,0	3,2	20,41	7,9
Ro Conc 3	7,83	0,8	4,0	7,63	2,5
Ro Conc 4	11,16	1,1	5,1	2,28	0,9
Ro Conc 5	24,28	2,4	7,5	0,96	0,5
Ro Tail	920,3	92,5	100	0,20	14,4
Head calc	995,2	100		1,33	100
Head Meas	1000			1,42	
Variance				6,5%	

Product	Mass			Cum Fraction Assays	Cum Distribution
	(g)	indiv (%)	Cum %	Au (g/t)	Au (%)
Ro Conc 1	22,1	2,2	2,2	46,00	74,0
Ro Conc 2	9,6	1,0	3,2	41,06	81,8
Ro Conc 3	7,8	0,8	4,0	36,41	84,3
Ro Conc 4	11,2	1,1	5,1	31,54	85,2
Ro Conc 5	24,3	2,4	7,5	26,92	85,6
Ro Tail	920,3	92,5	100	1,33	100
Head calc	995,2	100			
Head Meas	1000				
Variance					

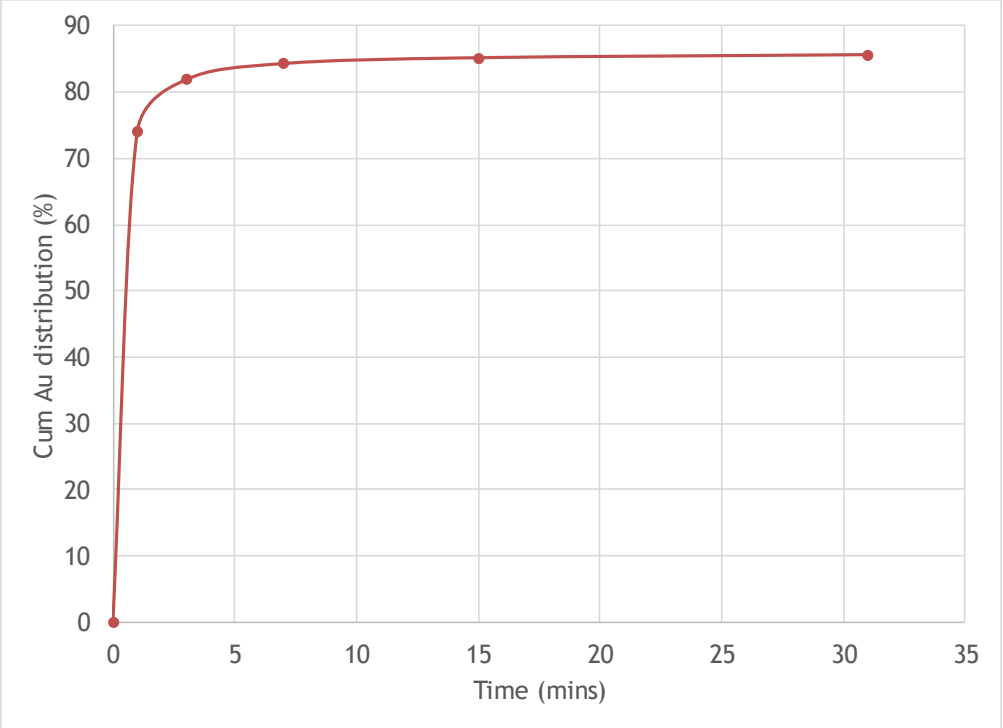


Figure 14: Rougher rate results – Master composite

In general, the Au found within the Namdini master composite sample was fast floating with ~81.8% of the total Au recovered into a 3.2% mass at a concentrate grade of ~41.06g/t Au within the first 3 minutes of cumulative flotation time. Floating for an additional 4 minutes resulted in an average of ~84.3% of the total Au being recovered into a ~4.0% cumulative mass at an average concentrate grade of 36.41g/t Au. There was limited flotation recovery achieved after 7 minutes of cumulative flotation time showing that there were insignificant quantities of slow floating Au within the samples.

3.6.1.4. Rougher flotation test results on lithology samples

In order to ascertain the extent of variability across the different lithology samples via flotation, single rougher flotation tests were carried out using the optimum conditions established. The results obtained for the V, G and D samples are presented in Table 31, Table 32 and Table 33 respectively while Figure 15 illustrates the scatter plot obtained from the results.

Table 31: Rougher flotation results – Volcanoclastics

Product	Mass Distribution			Fraction Assays		Distribution	
	(g)	indiv (%)	Cum %	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-} (%)
Ro Conc	42,70	4,3	4,3	31,75	13,20	83,5	95,0
Ro Tail	953,7	95,7	100	0,28	0,03	16,5	5,0
Head calc	996,4	100		1,63	0,60	100	100
Head Meas	1000			1,88			
Variance				15,4%			

Table 32: Rougher flotation results – Granitic

Product	Mass Distribution			Fraction Assays		Distribution	
	(g)	indiv (%)	Cum %	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-} (%)
Ro Conc	31,02	3,2	3,2	26,02	16,30	81,7	93,8
Ro Tail	953,7	96,8	100	0,19	0,04	18,3	6,2
Head calc	984,8	100		1,00	0,55	100	100
Head Meas	1000			0,88			
Variance				12,0%			

Table 33: Rougher flotation results – Diorites

Product	Mass Distribution			Fraction Assays		Distribution	
	(g)	indiv (%)	Cum (%)	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-} (%)
Ro Conc	74,63	7,5	7,5	18,64	15,20	90,4	98,9
Ro Tail	922,2	92,5	100	0,16	0,01	9,6	1,1
Head calc	996,9	100		1,54	1,15	100	100
Head Meas	1000			1,59			
Variance				3,0%			

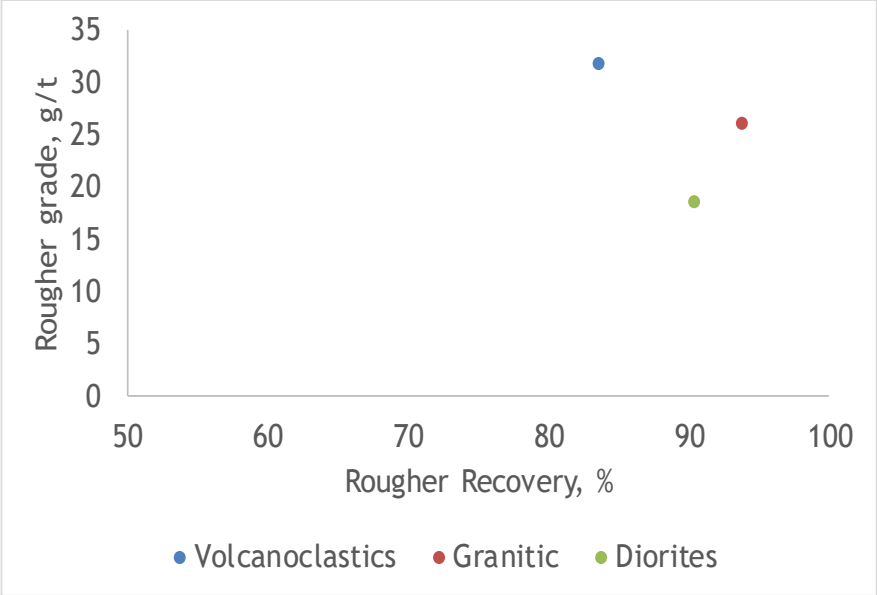


Figure 15: Rougher flotation results – V, G and D

All 3 lithology generally exhibited good flotation response with ~83.5%, 81.7% and 90.4% of the total Au recovered from the V, G and D samples respectively. The corresponding rougher concentrate grades were ~31.75g/t, ~26,02g/t and ~18.64g/t. Sulphur (S^{2-}) recovery was higher being ~95.0%, ~93.8% and ~98.9% for the V, G and D samples respectively.

3.6.1.5. Bulk concentrate production campaign results

A flotation campaign was carried out to produce bulk flotation concentrate for further metallurgical test work. The campaign was carried out using the optimum flotation conditions established under section 3.5. A representative portion of the concentrate was submitted for mineralogical analyses (QEMSCAN) evaluation and the results are reported in section 3.4.1.6. while another portion was analysed for Au and S using fire assay lead collection with ICP finish. The results obtained are presented in Table 34.

Table 34: Bulk concentrate production results on Master composite

Product	Mass Distribution			Fraction Assays		Distribution (%)	
	(kg)	indiv (%)	Cum %	Au, g/t	S^{2-}, %	Au	S^{2-}
Clnr Conc	3,10	2,6	2,6	47,50	31,25	81,9	71,0
Clnr Tails	6,15	5,2	7,8	2,63	3,06	9,0	13,8
Ro Tail	109,6	92,2	100	0,15	0,19	9,1	15,3
Head calc	118,9	100		1,51	1,15	100	100
Head Meas	120			1,42	1,15		
Variance				6,2%	0,1%		

Product	Mass Distribution			Cum F.Assays		Cum Distribution (%)	
	(kg)	indiv (%)	Cum %	Au, g/t	S^{2-}, %	Au	S^{2-}
Clnr Conc	3,10	2,6	2,6	47,50	31,25	81,9	71,0
Clnr Tails	6,15	5,2	7,8	17,67	12,51	90,9	84,7
Ro Tail	109,6	92,2	100	1,51	1,15	100	100
Head calc	118,9	100					
Head Meas	120						
Variance							

From the results ~81.9% of the total Au was recovered into a cleaner concentrate of mass pull 2.6% at a cleaner concentrate grade of 47.5g/t Au; the S^{2-} grade was ~31.25%. It was however noticed that the S^{2-} recovery was significantly lower when compared to the other tests being ~71.0% and this negatively affected Au recovery into the final concentrate. This could have been caused by low energy input during bulk flotation since a bigger flotation cell was used. The concentrate was stored in a contamination free environment prior to further test work.

3.7. GOLD DISSOLUTION TEST RESULTS ON FLOTATION PRODUCTS

In order to determine the effect of gold dissolution on the flotation products, a set of different tests were carried out on the flotation concentrates using intensive cyanidation conditions which comprised of;

- ✓ 2kg/t NaCN for conventional cyanidation and 5kg/t Intensive cyanidation
- ✓ 20g/L Carbon,
- ✓ pH of between 10.5 – 11.0 controlled by addition of lime and

- ✓ dissolution time of 24 hours.

All leach products were analysed for Au only using fire assay, lead collection with ICP finish.

3.7.1.1. Au dissolution tests on grind optimisation products results

Au dissolution tests were carried out on the grind optimisation flotation products in order to ascertain the extent of Au recovery achieved at different grinds. This work was carried out on the rougher tails. The results obtained are presented in Table 35.

Table 35: CIL test results on grind optimisation rougher tails – Master composite

Item	Float Grind	Leach feed, g/t	Leach tail, g/t	Au Recovery, %
1	80%-53µm	0,12	0,11	8%
2	80%-75µm	0,10	0,06	40%
3	80%-106µm	0,10	0,10	0%
4	80%-150µm	0,09	0,08	11%
5	80%-212µm	0,11	0,08	27%

In all cases, Au solubilised by conventional cyanidation was proportionately small reaching a maximum of ~40% at the grind of 80%-75µm. Because of magnitudes of dissolutions involved it may not be recommended to leach the flotation tails.

3.7.1.2. Effect of leach enhancers on float concentrates

The effect of different leach enhancers on Au dissolution from the flotation concentrate was tested using 4 different products comprising of;

- ✓ $PbNO_3$ (1.6kg/t)
- ✓ H_2O_2,
- ✓ Air injection and
- ✓ Pure oxygen

All leach products were analysed for Au only using fire assay, lead collection with ICP finish. The results obtained are presented in Table 36.

Table 36: Effect of leach enhancers of Au dissolution – Flotation concentrate

Bottle roll	Control expt	$PbNO_3$, 1,6kg/t	**H_2O_2**	**Air injection**	**Pure O_2**	Roasting
Head assay value, g/t	47,5	47,5	47,5	47,5	47,5	47,5
Leach residue, g/t	15,3	14,7	16,3	16,9	16,2	6,4
Au Rec, %	67,8%	69,1%	65,7%	64,4%	65,9%	86,5%

Incorporating $PbNO_3$ @1.6kg/t resulted in slight improvement in Au dissolution with ~69.1% of the total Au leached; there was ~1.3% increase in Au dissolution when compared to ~67.8% leached without the accelerant. Using H_2O_2, Air injection and Pure O_2 did not prove beneficial with Au solubilisations of ~65.7%, 64.4% and 65.9% attained. Roasting the concentrate however resulted in the best performance with ~86.5% of the gold solubilised during cyanidation of roasted concentrate. This was attributed to improved liberation of gold from the sulphide matrix which was destroyed by roasting leaving even the fine Au exposed to chemical attach during cyanidation.

3.7.1.3. Kinetic dissolution test results – Flotation concentrate

A single gold dissolution kinetic test was carried out on a representative sample portions of the flotation concentrate as a control experiment while the effect of Pb nitrate on a similar sample was carried out to establish any change in rate of gold dissolution achieved using $PbNO_3$ as an enhancer. Leaching was carried out for 48 hours with pulp sample drawn at intervals of 0, 1, 2, 4, 8, 16, 24, 36 and 48 hours. The results obtained are presented in Table 37 while Figure 16 illustrates the curves obtained from the data.

Table 37: Kinetic dissolution test results – Namdini MC

Leaching time, hrs		1	2	4	8	16	24
Head assay value: g/t, Au		47,5	47,5	47,5	47,5	47,5	47,5
Baseline	Leach residue, g/t	18,9	16,2	16,1	16,1	15,9	15,3
	Au leach recovery (%)	60,2	65,9	66,1	66,1	66,5	67,8
PbNO3 @ 1,6kg/t	Leach residue: g/t, Au	18,2	15,9	15,2	15,0	14,9	14,7
	Au leach recovery (%)	61,7	66,5	68,0	68,4	68,6	69,1

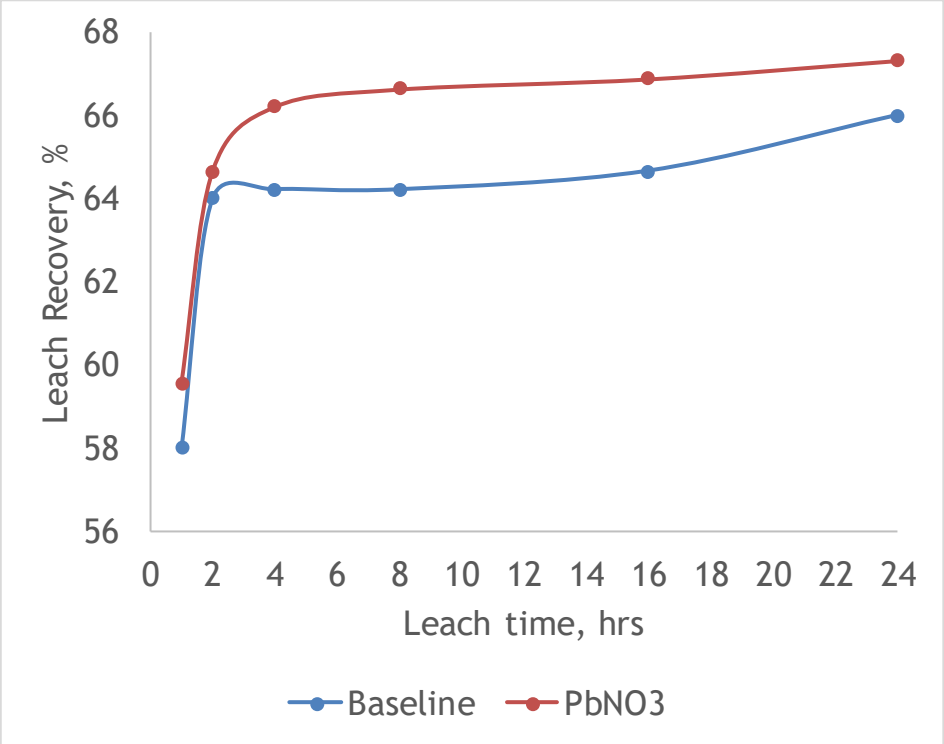


Figure 16: Kinetic dissolution test results – Namdini MC

Use of a leach accelerant ($PbNO_3$) resulted in improved Au dissolution kinetics and overall Au recovery. The impact of the accelerant was noticed within the first 4 hours of cumulative cyanidation hours wherein ~66.1 and 68.0 of the total Au was solubilised with and without the accelerant respectively. Overall, the accelerant resulted in an additional 1.3% of gold leached; however the project economics will determine whether the cost of the accelerant can be justified by the additional Au recovery achieved.

3.7.1.4. Cyanidation test work on flash flotation products

The extent of Au dissolution on flash flotation products was tested on representative portions of products generated under section 3.6.1.2.The flash flotation tails were subjected to conventional cyanidation conditions while the flash flotation concentrates were cyanided under intensive cyanidation conditions. The results obtained are presented in Table 38.

Table 38: Cyanidation test results on flash flotation products

Item	Sample	Head, g/t Au	Tail, g/t Au	Au rec, %
1	Flash Float concentrate	24,46	15,9	35,0%
2	Flash Float tails	0,45	0,26	42,2%

Intensive and conventional cyanidation leaching on the flash flotation concentrates and tails resulted in ~35.0% and 42.2% of the total Au solubilised respectively. The results obtained showed there was no significant benefit in leaching the flash flotation products.

3.7.1.5. Cyanidation with total roasting

A representative 500g aliquot of the flotation concentrate was sub-sampled by rotary splitting and further split into 2 equal portions weighing ~250g each. One portion was roasted at Suntech by placing it inside an oven at ~750^{o}C for about 4 hours to destroy the sulphide matrix. The other portion was roasted by SGS South Africa for comparative purposes. Both roasted portions were cooled down to ambient temperatures and cyanided using the conventional cyanidation method for ~24 hours of cumulative leaching time. In both cases, the cyanidation residues were pulverised to 90%-75µm and analysed for Au by fire assaying, lead collection with ICP finish. Table 39 presents the results obtained.

Table 39: Cyanidation results on roasted flotation concentrates

Item	Description	Concentrate roasted at	
		Suntech	SGS South Africa
1	Leach head: g/t, Au	47,5	47,5
2	Leach tails: g/t, Au (Assayed)	9,8	9,8
3	Leach tails: g/t, Au (normalised)	6,4	6,4
4	Mass loss during roasting (%)	35	34,9
5	Leach recovery (%)	86,6%	86,6%

Approximately, 86.6% of the total Au contained within the roasted flotation concentrate was leached out leaving behind a residue assaying ~9.8g/t, Au. The results obtained at SGS and Suntech compared well with exactly the same leach tails obtained in both cases.

3.7.1.6. Cyanidation of concentrates after ultra-fine milling

Representative sample portions of the flotation concentrates were submitted for ultra-fine milling using a stirred media mill (SMD) in order to further liberate any locked Au within the sample. The SMD test was run at 3 energy levels of 5 kWt/h, 10 kWt/h and 15kWt/h. Representative portions of the milled products were submitted for Malvern sizing in order to determine the grind. The Malvern sizing results are illustrated in Figure 17.

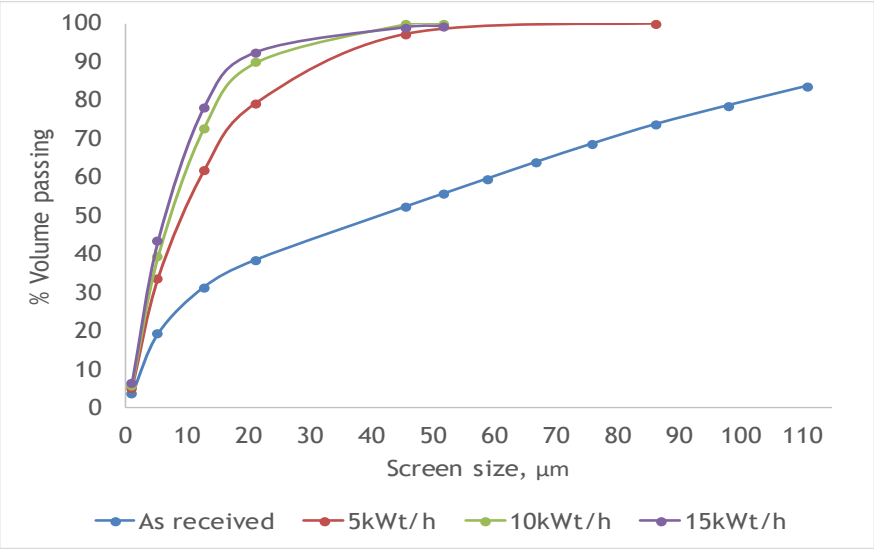


Figure 17: Particle size distribution by a Malvern sizer

The Malvern sizer results indicated that milling the concentrates resulted in increased breakage (level of fineness) when compared to the concentrates as received. Ultra-fine milling at 5kWt/h, 10kWt/h and 15kWt/h resulted in grinds of ~80%-22μm, ~80%-15μm and ~80%-14μm respectively from 80%-109μm as received. The cyanidation results obtained at the different grinds are presented in Table 40.

Table 40: Cyanidation on ultra-fine milled concentrates

Energy input, kWh/t	**As received**	**5**	**10**	**15**
Grind	~80%-109μm	~80%-22μm	~80%-15μm	~80%-14μm
Head assay value, g/t	47,5	47,5	47,5	47,5
Leach residue, g/t	15,2	10,5	9,4	9,6
Au Rec, %	**68,0%**	**77,9%**	**80,2%**	**79,8%**

Ultra-fine milling the concentrates at different energy inputs generally resulted in increased Au dissolution with 10kWh/t achieving the highest leach recovery of ~80.2%; this amounted to an increase in Au leach recovery of ~12.2%. Ultra-fine milling to 15kWh/t did not result in improved Au solubilisation with ~79.8% of the total Au leached. These results made sense since there was no real benefit accrued additional energy input to 15kWt/h as the level of fineness remained the same with that achieved at 10kWt/h.

4.0. CONCLUSIONS

The results of the flotation and cyanidation test work as well as findings of the mineralogical characterisation work on the master composite, V, G and D ore samples confirmed that the Au found in the samples was highly amenable to beneficiation by froth flotation but partially refractory during cyanidation. The bulk of the Au in the feed was associated with or hosted by pyrite (~75.4%) while an additional 10.6% was hosted by other sulphides.

The other conclusions drawn from this phase of test work were:

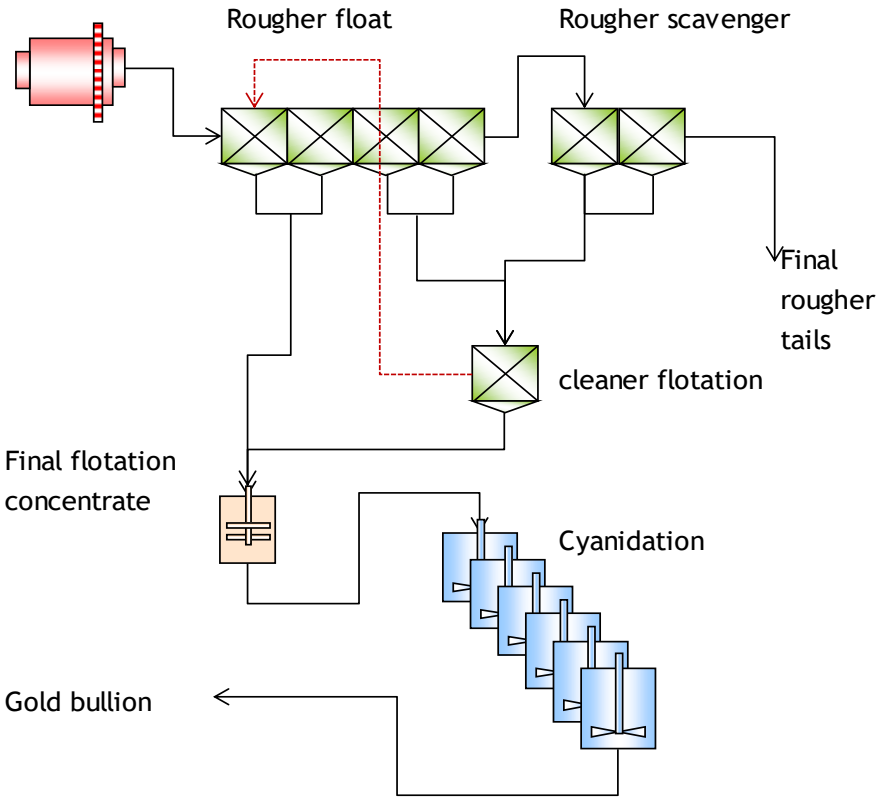
- ✓ The bulk of the Au found in the Namdini master composite was generally fine with a particle size <13.5 ECD(µm) while the grain size of the host sulphide minerals was much coarser with >7% of the pyrite being coarser than 140µm.,
- ✓ The Au recovered into the flotation concentrate was poorly liberated even at the grind of 80-75µm with ~31.8% locked and only 27.5% liberated,
- ✓ All 3 samples contained significant quantities of As and C with average assay values of ~365ppm and 1.9% respectively,
- ✓ The SG of the master composite was approximately 2.74,
- ✓ SPi results indicated that the power draws from the V, G, D and master composite samples ranged from 8.83 and 9.57 kWh/t while the Bond ball work index for the master composite was 14.9 kWh/t thus the Namdini ore was classified as being hard,
- ✓ The Namdini Au sample exhibited insignificant preg-robbing characteristics with ~2.8% of the total Au preg-robbed,
- ✓ The optimum grind for both froth flotation and cyanidation was 80%-75µm,
- ✓ Incorporating leach enhancers during Au cyanidation was beneficial with additional Au leach recoveries exceeding 5% noted,
- ✓ Dosing $PbNO_3$ at 1.6kg/ton significantly improved Au leaching kinetics within the first 4 hours of cumulative cyanidation time,
- ✓ There was wide variability in Au solubilisations achieved amongst the 3 lithology with ~63.3%, ~46.8% and ~66.7% of the Au found in the V, G and D samples solubilised; though it was suspected that the G sample's poor

performance was due to poor liberation of Au as a result of coarse grind, it is recommended to investigate further,

- ✓ Lime consumption across the 3 lithology samples and the master composite ranged between 0.26kg/t and 0.53kg/t for all conditions tested but cyanide consumptions were high ranging between 1.35kg/t and 1.60kg/t,
- ✓ The best reagent scouting results were obtained using Test 7 conditions which comprised of 50g/t PAX (Potassium Amyl Xanthate), 10g/t Aero 3148A (promoter), and 40g/t XP200 frother and a grind of 80%-75µm. Using these conditions, ~89.2% of the total Au was recovered into a cleaner concentrate of mass pull 2.2% at a cleaner concentrate grade of 53.56%,
- ✓ Au found within the Namdini master composite sample was fast floating with ~81.8% of the total Au recovered into a 3.2% mass at a concentrate grade of ~41.06g/t Au within the first 3 minutes of cumulative floatation.
- ✓ Ultra-fine milling the concentrates at different energy inputs generally resulted in increased Au dissolution with 10kWh/t achieving the highest leach recovery of ~80.2%; this amounted to an increase in Au leach recovery of ~12.2%. Ultra-fine milling to 15kWh/t as no beneficial was noted.

5.0. RECOMMENDATIONS

The test work results obtained indicated that the Namdini ore tested was a typical partially refractory gold ore. Based on these results, the author recommends 2 process routes for recovery of gold into saleable bullion. Recommended flowsheet 1 is illustrated in Figure 18.



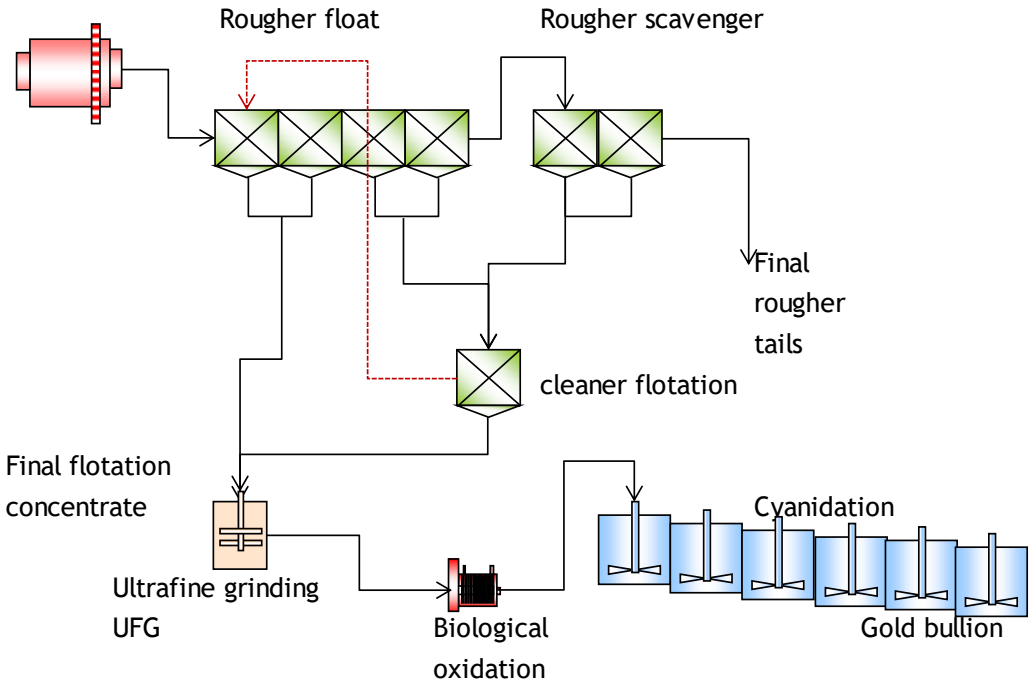
Projected Au recovery (%) = 95% (Float rec) * 80% (UFG + leach rec) = 76%

Figure 18: Recommended process flow sheet (Flotation>UFG>Cyanidation)

This option is projected to achieve ~76% overall Au recovery. The process would require the ore to be milled to 80%-75µm followed by a rougher and rougher scavenger flotation with a single cleaning stage to target an overall flotation recovery of ~95%, mass pull of ~8 % and concentrate grade of between 15g/t, Au and 25g/t, Au. The cleaner concentrate is UFG to 80%-10µm and cyanided to obtain a recovery of ~80% which implies an overall recovery of 80%x95%=76%. This process combination is not expected to exceed 76% as the ore mineralogy indicates that a significant percentage of the gold occurs in sub-microscopic (50% <5 µm and 26% <2 µm) and it is uneconomic to try and liberate the sub-microscopic gold occurring at <3 µm generally using

conventional processes. This gold only requires destruction of the sulphide matrix using oxidation processes such as bacterial, pressure and roasting oxidation.

Figure 19 illustrates the second recommended flow sheet.



Projected Au recovery (%) = 95% (Float rec) * 95% (UFG, BIOX + leach rec) = 90.3%

Figure 19: Recommended process flow sheet 2 (Flotation>UFG>BIOX>Cyanidation)

This process route is projected to achieve an overall Au recovery of up to ~90% but can only be justified on the basis of comparative economics as they have a higher OPEX and CAPEX; it therefore remains to be evaluated if the incremental 14% Au recovery can make up for the additional CAPEX and OPEX.

Based on the results obtained at this stage and in order to make informed decisions regarding the optimal flow sheet for further testing it is further recommended to carry out further work as follows:

i. Since there is sufficient test work information, it is recommended to conduct a Preliminary Economic Evaluation (PEE) of the conventional processes and their variables and a high level evaluation of alternative refractory processes for comparison and decision making on whether it's worthwhile to conduct

additional test work (non-conventional) or optimise the conventional on the basis of an expected overall recovery of 76 %: *Typically operators of partially refractory ores apply the conventional processes to get the recoveries of 70-75 % and stockpile the high grade tails for future treatment depending on the economics)*

ii. The variability cyanidation test work results on the different lithology indicated that the Au found within the G samples was significantly more refractory when compared to the other 2 lithology; it is therefore recommended to exclude this sample from the composite and treat it separately. It is therefore recommended to run separate tests for this sample,

iii. Complete further test work as follows:

a. Optimisation test work to fine tune conditions for optimal recovery of Au by flotation and gangue rejection for production of a high quality. Low mass concentrate at flotation recoveries exceeding 95%,
b. Complete a locked cycle test using optimum conditions in order to project the flotation performance or to simulate a continuous flotation operation of the ore samples when tails streams are recycled back into the process and to assess the impact of recycled effluent on overall flotation response of the ore samples at industrial scale,
c. Complete a comprehensive variability flotation and comminution test work in order to establish the extent of variability exhibited across and deeper into the deposit,
d. Evaluate the effectiveness of Option 2 circuit which comprise of milling > flotation > Ultrafine grinding > BIOX > Cyanidation,
e. Investigate presence of cyanicides within ore sample and to establish the type of these cyanicides

APPENDIX

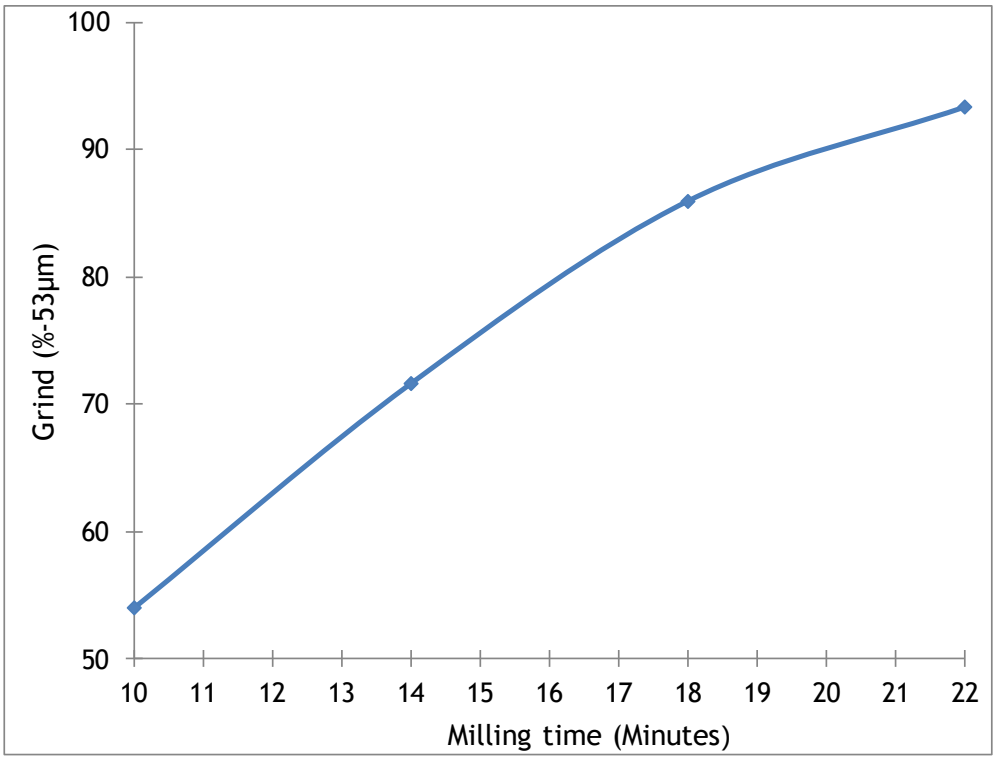


Figure 20: Milling curve – 80%-53µm

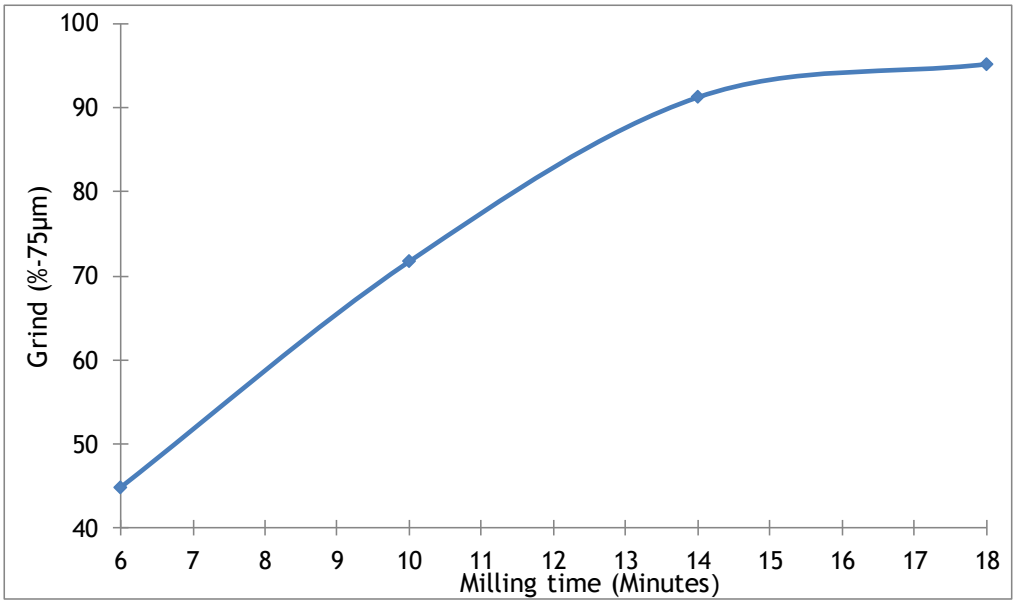


Figure 21: Milling curve – 80%-75µm

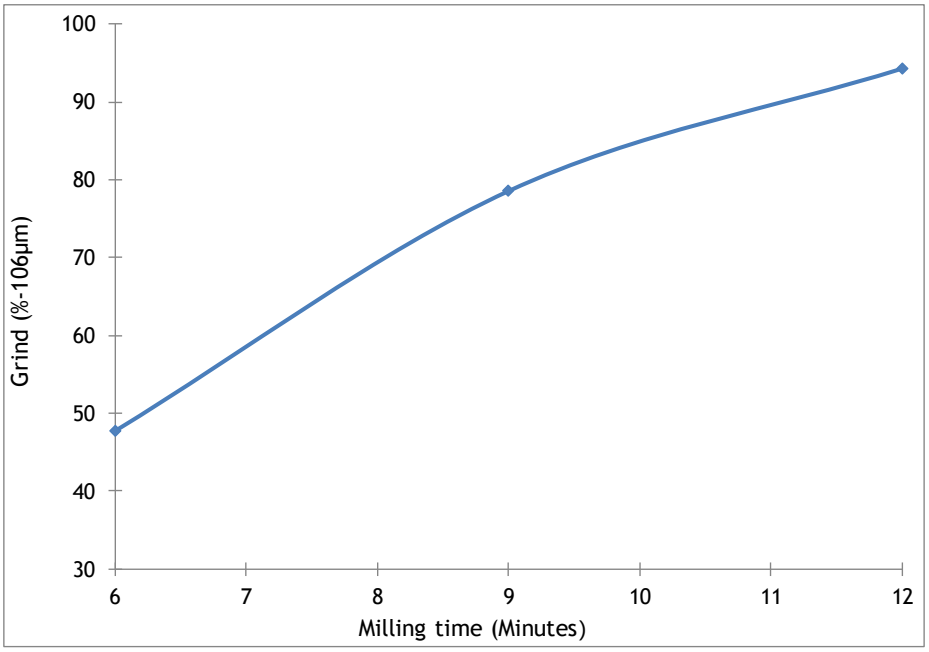


Figure 22: Milling curve – 80%-106µm

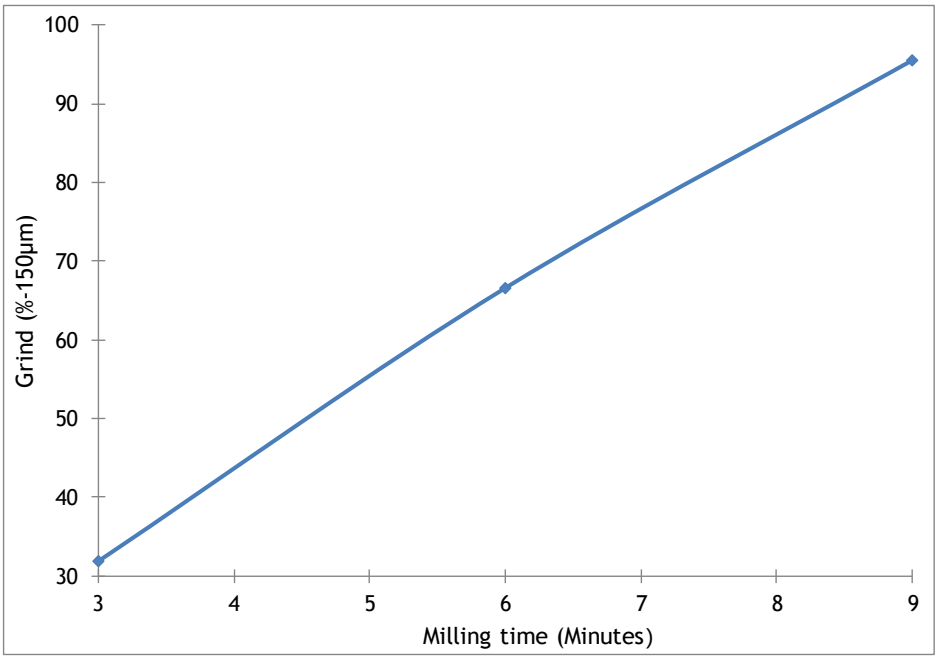


Figure 23: Milling curve – 80%-150µm

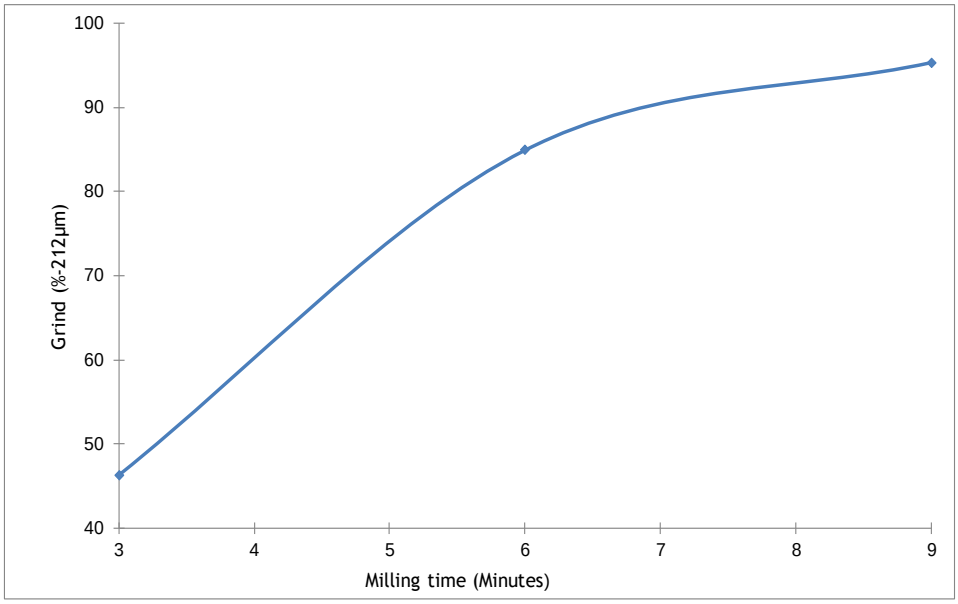


Figure 24:Milling curve – 80%-212µm

Table 41: Distribution of Au grains by size on Namdini ROM sample

Sieve Size (µm)	Mass %	Cumulative %
13.5	0,0	100,0
11.4	11,3	100,0
9.6	23,9	88,7
8.1	6,5	64,8
6.8	11,0	58,3
5.7	7,3	47,3
4.8	7,9	40,0
4.1	0,0	32,1
3.4	13,3	32,1
2.9	4,0	18,9
2.4	3,2	14,9
2	5,0	11,6
1.75	2,3	6,6
1.45	1,2	4,2
1.2	1,6	3,0
1	0,8	1,4
0.87	0,6	0,6

Table 42: Distribution of Au grains by size on Namdini sulphide concentrate

Sieve Size (µm)	Mass %	Cumulative mass %
19	0,0	100,0
16	9,5	90,5
13.5	3,3	87,2
11.4	9,3	78,0
9.6	9,7	68,3
8.1	6,3	62,0
6.8	5,9	56,1
5.7	6,0	50,1
4.8	4,7	45,4
4.1	2,6	42,8
3.4	6,9	35,9
2.9	5,7	30,3
2.4	4,4	25,9
2	5,7	20,2
1.75	3,9	16,3
1.45	5,9	10,4
1.2	6,0	4,4
1	4,4	0,0
0.87	0,0	0,0

Table 43: Test 1 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	27,57	2,8	2,8	42,00	87,5
Clnr Tails	57,48	5,8	8,5	1,93	8,4
Ro Tail	912,6	91,5	100	0,06	4,1
Head calc	997,7	100		1,33	100
Head Meas	1000			1,42	
Variance				7,0%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	27,57	2,8	2,8	42,00	87,5
Clnr Tails	57,48	5,8	8,5	14,92	95,9
Ro Tail	912,6	91,5	100	1,33	100
Head calc	997,7	100			
Head Meas	1000				
Variance					

Table 44: Test 2 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	24,25	2,4	2,4	48,00	81,5
Clnr Tails	19,97	2,0	4,4	5,64	7,9
Ro Tail	951,6	95,6	100	0,16	10,7
Head calc	995,8	100		1,43	100
Head Meas	1000			1,42	
Variance				1,0%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	24,25	2,4	2,4	48,00	81,5
Clnr Tails	19,97	2,0	4,4	28,87	89,3
Ro Tail	951,6	95,6	100	1,43	100
Head calc	995,8	100			
Head Meas	1000				
Variance					

Table 45: Test 3 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	42,78	4,3	4,3	29,00	84,5
Clnr Tails	72,16	7,2	11,5	1,08	5,3
Ro Tail	882,0	88,5	100	0,17	10,2
Head calc	997,0	100		1,47	100
Head Meas	1000			1,42	
Variance				3,6%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	42,78	4,3	4,3	29,00	84,5
Clnr Tails	72,16	7,2	11,5	11,47	89,8
Ro Tail	882,0	88,5	100	1,47	100
Head calc	997,0	100			
Head Meas	1000				
Variance					

Table 46: Test 4 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	22,38	2,2	2,2	44,00	76,63
RClnr Tails	7,84	0,8	3,0	11,40	7,0
Clnr Tails	44,79	4,5	7,5	1,00	3,5
Ro Tail	922,7	92,5	100	0,18	12,9
Head calc	997,7	100		1,29	100
Head Meas	1000			1,42	
Variance				10,3%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	22,38	2,2	2,2	44,00	76,63
RClnr Tails	7,84	1	3,0	35,54	83,6
Clnr Tails	44,79	4,5	7,5	14,92	87,1
Ro Tail	922,7	92,5	100	1,29	100
Head calc	997,7	100			
Head Meas	1000				
Variance					

Table 47: Test 5 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	33,26	3,3	3,3	36,00	84,16
RClnr Tails	59,76	6,0	9,3	1,64	6,9
Clnr Tails	134,71	13,5	22,8	0,26	2,5
Ro Tail	770,1	77,2	100	0,12	6,5
Head calc	997,8	100		1,43	100
Head Meas	1000			1,42	
Variance				0,4%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	33,26	3,3	3,3	36,00	84,16
RClnr Tails	59,76	6,0	9,3	13,93	91,0
Clnr Tails	134,71	13,5	22,8	5,84	93,5
Ro Tail	770,1	77,2	100	1,43	100
Head calc	997,8	100			
Head Meas	1000				
Variance					

Table 48: Test 6 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	24,63	2,5	2,5	49,00	82,6
RClnr Tails	15,92	1,6	4,1	6,96	7,6
Clnr Tails	77,66	7,8	11,8	0,71	3,8
Ro Tail	880,8	88,2	100	0,10	6,0
Head calc	999,0	100		1,46	100
Head Meas	1000			1,42	
Variance				2,9%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	24,63	2,5	2,5	49,00	82,6
RClnr Tails	15,92	2	4,1	32,50	90,2
Clnr Tails	77,66	7,8	11,8	11,61	94,0
Ro Tail	880,8	88,2	100	1,46	100
Head calc	999,0	100			
Head Meas	1000				
Variance					

Table 49: Test 7 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	16,53	1,7	1,7	61,0	77,2
Cleaner conc	5,22	0,5	2,2	30,0	12,0
Clnr Tails	32,39	3,3	5,4	2,01	5,0
Ro Tail	941,8	94,6	100	0,08	5,8
Head calc	996,0	100		1,31	100
Head Meas	1000			1,42	
Variance				8,3%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	16,53	1,7	1,7	61,0	77,2
RClnr Tails	5,22	1	2,2	53,6	89,2
Clnr Tails	32,39	3,3	5,4	22,7	94,2
Ro Tail	941,8	94,6	100	1,31	100
Head calc	996,0	100			
Head Meas	1000				
Variance					

Table 50: Test 8 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	21,95	2,2	2,2	56,00	82,4
Clnr Tails	33,28	3,3	5,5	4,22	9,4
Ro Tail	943,0	94,5	100	0,13	8,2
Head calc	998,3	100		1,49	100
Head Meas	1000			1,42	
Variance				5,0%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
Clnr Conc	21,95	2,2	2,2	56,00	82,4
Clnr Tails	33,28	3,3	5,5	24,80	91,8
Ro Tail	943,0	94,5	100	1,49	100
Head calc	998,3	100			
Head Meas	1000				
Variance					

Table 51: Test 9 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	21,24	2,2	2,2	58,00	85,9
RClnr Tails	13,70	1,4	3,5	4,15	4,0
Clnr Tails	55,04	5,6	9,1	1,02	3,9
Ro Tail	895,6	90,9	100	0,10	6,2
Head calc	985,5	100		1,46	100
Head Meas	1000			1,42	
Variance				2,4%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	21,24	2,2	2,2	58,00	85,9
RClnr Tails	13,70	1	3,5	36,89	89,8
Clnr Tails	55,04	5,6	9,1	14,95	93,8
Ro Tail	895,6	90,9	100	1,46	100
Head calc	985,5	100			
Head Meas	1000				
Variance					

Table 52: Test 10 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	30,53	3,1	3,1	39,00	80,1
RClnr Tails	22,27	2,3	5,4	2,48	3,7
Clnr Tails	149,08	15,3	20,7	1,25	12,5
Ro Tail	775,2	79,3	100	0,07	3,7
Head calc	977,1	100		1,52	100
Head Meas	1000			1,42	
Variance				6,7%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	30,53	3,1	3,1	39,00	80,1
RClnr Tails	22,27	2	5,4	23,60	83,8
Clnr Tails	149,08	15,3	20,7	7,09	96,3
Ro Tail	775,2	79,3	100	1,52	100
Head calc	977,1	100			
Head Meas	1000				
Variance					

Table 53: Test 11 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	18,02	1,8	1,8	53,12	74,6
RClnr Tails	7,29	0,7	2,6	27,49	15,6
Clnr Tails	37,81	3,8	6,4	3,09	9,1
Ro Tail	929,3	93,6	100	0,01	0,7
Head calc	992,4	100		1,29	100
Head Meas	1000			1,42	
Variance				9,8%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	18,02	1,8	1,8	53,12	74,6
RClnr Tails	7,29	1	2,6	45,74	90,2
Clnr Tails	37,81	3,8	6,4	20,19	99,3
Ro Tail	929,3	93,6	100	1,29	100
Head calc	992,4	100			
Head Meas	1000				
Variance					

Table 54: Test 12 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	21,21	2,1	2,1	48,45	80,0
RClnr Tails	10,83	1,1	3,2	11,82	10,0
Clnr Tails	49,40	5,0	8,2	1,13	4,3
Ro Tail	916,4	91,8	100	0,08	5,7
Head calc	997,8	100		1,29	100
Head Meas	1000			1,42	
Variance				10,3%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	21,21	2,1	2,1	48,45	80,0
RClnr Tails	10,83	1	3,2	36,07	89,9
Clnr Tails	49,40	5,0	8,2	14,88	94,3
Ro Tail	916,4	91,8	100	1,29	100
Head calc	997,8	100			
Head Meas	1000				
Variance					

Table 55: Test 13 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	20,82	2,1	2,1	50,68	79,0
RClnr Tails	4,58	0,5	2,6	16,18	5,6
Clnr Tails	33,33	3,4	5,9	3,09	7,7
Ro Tail	935,7	94,1	100	0,11	7,7
Head calc	994,4	100		1,34	100
Head Meas	1000			1,42	
Variance				5,8%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	20,82	2,1	2,1	50,68	79,0
RClnr Tails	4,58	0	2,6	44,46	84,6
Clnr Tails	33,33	3,4	5,9	20,98	92,3
Ro Tail	935,7	94,1	100	1,34	100
Head calc	994,4	100			
Head Meas	1000				
Variance					

Table 56: Test 14 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	18,04	1,8	1,8	46,08	69,3
RClnr Tails	7,16	0,7	2,5	25,00	14,9
Clnr Tails	47,40	4,8	7,3	2,05	8,1
Ro Tail	925,3	92,7	100	0,10	7,7
Head calc	997,9	100		1,20	100
Head Meas	1000			1,42	
Variance				18,1%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	18,04	1,8	1,8	46,08	69,3
RClnr Tails	7,16	1	2,5	40,09	84,2
Clnr Tails	47,40	4,8	7,3	15,25	92,3
Ro Tail	925,3	92,7	100	1,20	100
Head calc	997,9	100			
Head Meas	1000				
Variance					

Table 57: Test 15 – Reagent scouting

Product	Mass Distribution			Fraction Assays (g/t)	Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	18,02	1,8	1,8	52,27	77,1
RClnr Tails	7,29	0,7	2,6	16,17	9,6
Clnr Tails	37,81	3,8	6,4	1,59	4,9
Ro Tail	929,3	93,6	100	0,11	8,4
Head calc	992,4	100		1,23	100
Head Meas	1000			1,42	
Variance				15,3%	

Product	Mass Distribution			Cum F.Assays (g/t)	Cum Distribution (%)
	(g)	indiv (%)	Cum %	Au	Au
RClnr Conc	20,54	2,1	2,1	52,27	77,1
RClnr Tails	7,63	1	2,8	41,87	86,7
Clnr Tails	41,74	4,2	7,1	17,74	91,6
Ro Tail	920,1	92,9	100	1,23	100
Head calc	990,0	100			
Head Meas	1000				
Variance					

Diagnostic leach procedure

A diagnostic leach test is done in order to quantify the extraction efficiency that can be achieved by various unit operations and to infer the deportment of gold in the minerals present. This procedure involves sequential solubilising of the least stable mineral and extracting the associated gold. The diagnostic leach yields an empirical mineralogical analysis with process information on a relatively large and hence representative sample.

The leach procedure involves the following sequential processing steps:

- ✓ To quantify free milling gold (i.e. gold that can be extracted by cyanidation), the sample is subjected to direct cyanidation,
- ✓ To quantify the gold that can be extracted by carbon-in-leach (CIL), that includes free gold and gold that is preg-robbed, a second sample is cyanided in the presence of carbon,
- ✓ To quantify gold that is extracted by a mild oxidative pre-leach or acid pre-leach, that is gold associated with pyrrhotite, calcite, dolomite, haematite, etc., the resulting residue from the preg-robbing test is pre-leached in hot HCl and cyanided in the presence of carbon,
- ✓ To quantify the gold that can be extracted via a severe oxidative pre-leach (pressure oxidation, bacterial oxidation, roasting), that is gold associated with pyrite, arsenopyrite, etc., the resulting residue is pre-leached in hot HNO_3 and cyanided in the presence of carbon,
- ✓ To quantify the gold that can be extracted via complete oxidation (roasting), that is gold associated with carbonaceous material such as kerogen, the resulting residue is roasted and cyanided in the presence of carbon,
- ✓ The undissolved gold appearing in the residue is assumed to be associated with quartz.

All gold dissolution products will be analysed in duplicate for gold only.