



Assessment of Vangorda Pit Backfilling 2004/05 Task 14g

Prepared for

Deloitte and Touche Inc.

On behalf of

Faro Mine Closure Planning Office

Prepared by



*Project Reference Number
SRK 1CD003.48*

March 2006



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2004/05 Task 14g

Deloitte and Touche Inc.

On behalf of

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March 2006

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1 Introduction

The Vangorda waste rock pile contains approximately 9.7 million m³ of waste rock, most of which is acidic or potentially acid generating. One option for the Vangorda dump is to relocate some or all of the material to the Vangorda Pit, which has a capacity to accommodate in excess of 9.0 million m³.

Potential benefits of relocating the waste rock include reduction of the impacted surface area, the isolation of acid generating material below the ultimate groundwater level, and the capability to restore Vangorda Creek to its original alignment. In other projects where acidic waste rock has been used to backfill a pit, lime or limestone have been added in sufficient quantities to neutralize any soluble acidity present in the rock. That approach provides the additional benefit of treating the acidic contaminants before they can be released to the ground or surface water.

As part of the 2004/05 planning meetings, a task to further investigate the option of relocating Vangorda waste rock into the Vangorda Pit was recommended. The task description was:

14g) Assess Vangorda Pit Backfill Requirements

Samples of Vangorda waste rock will be collected and tested for lime addition requirements. Volume calculations will be updated and lime addition, excavation and deposition costs will be estimated, and summarized in a report.

SRK was subsequently asked to provide a proposal to complete task 14g. The proposal was forwarded to Deloitte and Touche in May 2004, and the work was authorized in July 2004.

The steps undertaken to meet the task requirements were:

- i) A field sampling and testing program was undertaken to assess the amount of lime or limestone required to neutralize the soluble acidity in the Vangorda waste rock;
- ii) Laboratory sequential extraction and column tests were undertaken to assess the potential evolution of porewater quality in Vangorda waste rock amended with lime or limestone;
- iii) The spatial distribution of the soluble acidity in the waste rock was reviewed to assess the possibility of selectively placing waste rock at different locations in the pit, and the need for field control of lime or limestone addition rates;
- iv) A conceptual program for waste rock amendment and backfilling was developed, including related activities to re-route Vangorda Creek over the backfill; and,
- v) Overall project costs were estimated.

This report has been prepared as part of the ongoing technical evaluation for the closure planning of the Faro Mine and provides the results of the above steps. Section 2 below presents results of the field and laboratory testing. Section 3 presents and discusses the results from the geochemical investigation. A conceptual approach for the backfilling program is presented in Section 4. Cost estimates for the backfilling program are presented in Section 5. Aspects of this report may have been superseded by subsequent technical studies.

2 Field and Laboratory Methods

2.1 Sampling

Test pits were excavated in the Vangorda waste rock to obtain samples for field and laboratory testing. A backhoe was used for most of the pits. Shallow test pits were excavated in areas that were inaccessible by backhoe. The test pit locations and sample identification numbers are shown in Figure 2.1.

2.2 Field Testing

The samples were screened to less than 10 mm in the field, and paste pH and conductivity measurements were obtained. Each test pit photographed and logged for the rock types encountered and the state of oxidation. Test pit logs are provided in Appendix A.

2.3 Laboratory Testing

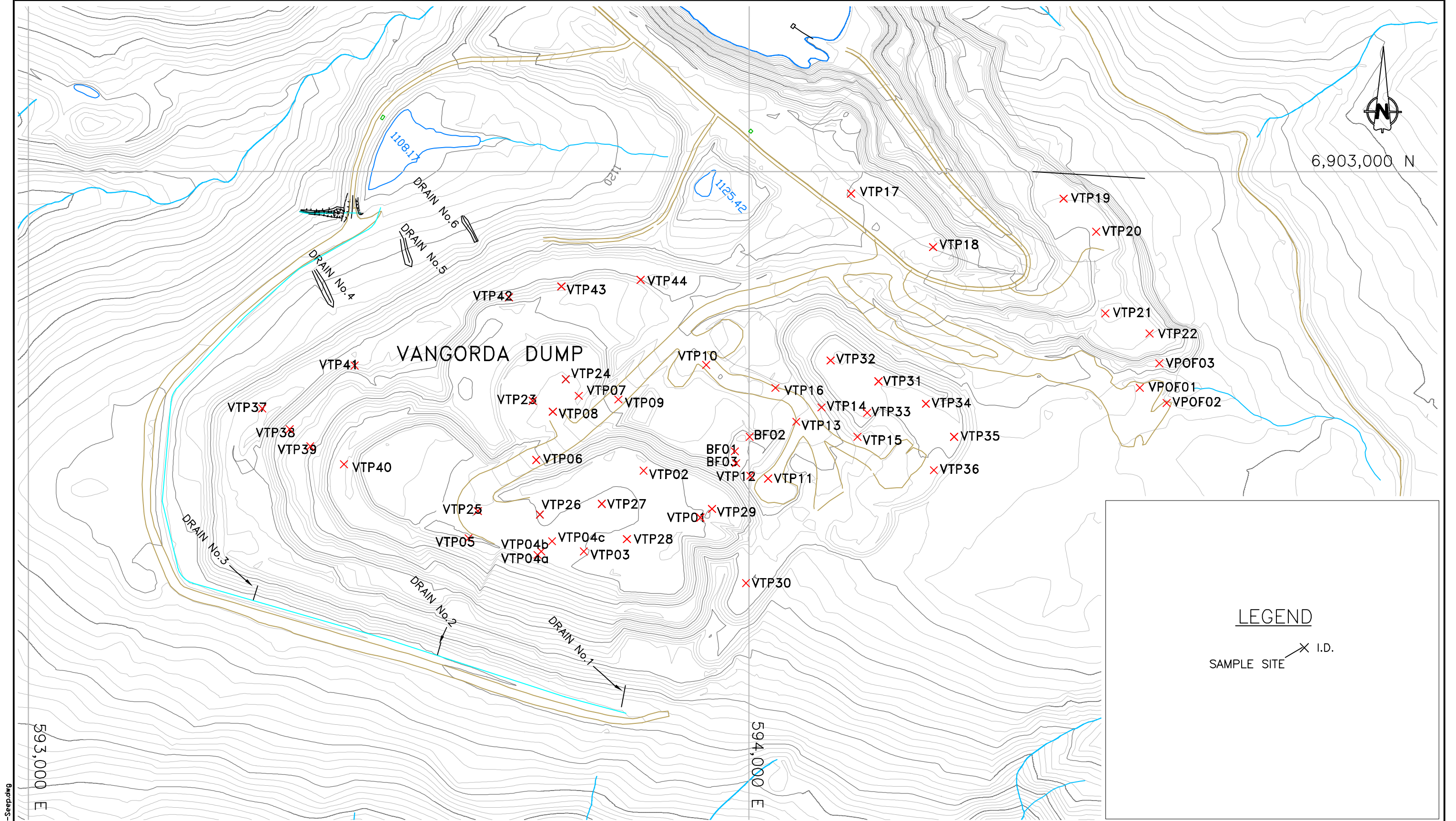
2.3.1 Lime Demand

Lime demand tests were completed at site, in the Anvil Range Laboratory, on all samples obtained during the 2004 sampling program. The tests provide an estimate of the amount of lime needed to neutralize the soluble acidity in a given mass of waste rock. (The estimated “lime demand” can be converted stoichiometrically to an estimate of “limestone demand”).

The tests were completed on the less than 10 mm size fraction of each sample. The procedure was as follows:

- i) 200 grams of the < 10 mm sample was placed in a 1 litre flask.
- ii) 400 mL of distilled water was added to the sample and agitated.
- iii) The slurry was mixed and the pH and Conductivity was measured.
- iv) The slurry was again mixed after 30 minutes and again at 60 minutes after which the pH and conductivity were again measured.
- v) Either a 10 g/L milk of lime or a 100 g/L milk of lime slurry was then used to adjusted the pH of the slurry to in excess of 9.5.
- vi) The slurry was allowed to react for an additional 60 minutes, and the pH and conductivity were measured. If the pH was below 9.5, 10 g/L milk of lime slurry was used to adjust the pH above 9.5.
- vii) The combined milk of lime addition was then used to calculate the lime demand.

Additional verification tests were completed to show that the lime demand after one hour reasonably approximated the lime demand estimated from longer 8-hour and 24-hour tests.



LEGEND

X I.D.

SAMPLE SITE

2.3.2 Basic Geochemical Characterization

The samples remaining after the lime demand tests were completed were shipped to Canadian Environmental and Metallurgical Inc. (CEMI) in Vancouver. All samples were analyzed for metals by ICP, and about half were submitted for acid base account (ABA) testing.

The samples selected for ABA testing were also submitted for net acid generation (NAG) tests, which were completed on the < 10mm size fraction. The NAG test has been used in other projects as a rapid and inexpensive means to identify acid generating material. The method was tested here to determine if it would be suitable for field control of lime or limestone addition rates.

2.3.3 Leach Extraction and Column Tests

Two composite samples were prepared and amended with lime and limestone. Sub-samples of the amended and un-amended material were then used in leach extraction and column tests, both of which were designed to illustrate the range of porewater quality that may result within the backfill after lime and limestone neutralization. The procedures are described in detail in a memorandum from SRK to CEMI, which is provided in Appendix B.

The two composite samples were prepared as shown in Table 2.1. The calculated lime demands for the composite samples were 0.65 and 2.53 kg $\text{Ca}(\text{OH})_2$ per tonne for Composites VG1 and VG2 respectively.

Three leach extraction tests were completed on each composite sample. The first test was on un-amended material, the second was on material amended with lime at the calculated lime demand, and the third was on material amended with limestone at a rate equivalent to the calculated lime demand.

The lime and limestone amended composite samples were also used in the column tests. Water from the base of the columns was recycled through the column to establish equilibrium conditions. After 15 days of recycling, the water was displaced with distilled water and sent for analysis. The water was then recycled for a further 21 days, and again sampled and analyzed.

Table 2.1: Samples Selection for Composites used in Leach Extraction and Column Tests

Composite VG1		Composite VG2	
Sample	Wt (g)	Sample	Wt (g)
VTP24A	280	VTP26A	750
VTP27A	280	VTP37A	750
VTP28B	280	VTP21	750
VTP29B	280	VTP42	750
VTP17A	280		
VTP23B	280		
VTP30B	280		
VTP25A	280		
VTP43	280		
VTP35A	280		
VTP23A	280		
Total Wt. (g)	3080		3000

3 Results

3.1 Basic Geochemical Characterization

Complete results of the metal analyses, acid base accounting and the net acid generation tests are provided in Appendix C.

3.1.1 Metals Analyses

The metals analyses show that samples taken from the Oxide Fines and the Baritic Fines consistently have zinc and lead contents in excess of 1%. Approximately 43 percent of the waste rock samples have more than 1% zinc. About 80 percent of the samples contain more than 0.1% zinc.

3.1.2 ABA Tests

The acid base account (ABA) and net acid generation (NAG) test results are provided in Table 3.1. The results indicate that:

- The Oxide Fines and Baritic Fines have a very high net potential for acid generation;
- All of the waste rock samples are considered net acid generating; and,
- The overburden and till area generally has a low sulphur content and is net acid consuming.

These results are consistent with the results from the 2002 geochemical characterization program. The spatial distributions of zinc content and acid generation potential are discussed further below in Section 3.5.

3.1.3 NAG Tests

The NAG test generally underestimated the net acid generation potential derived from the more rigorous ABA tests, as shown in Figure 3.1. However, Figure 3.2 shows a comparison of the final pH measured in the NAG tests to the net acid generation potential determined from the ABA tests, and indicates that samples yielding a NAG-pH of less than 4.5 are net acid generating. Conversely, samples with a NAG-pH in excess of 5 are likely to be non acid forming. It is concluded that the NAG-pH could be used as a rapid and inexpensive indicator of acid generation potential. However, the NAG-test apparently does not provide an accurate indication of the magnitude of the net acid generation potential.

Table 3.1: Acid Base Account and NAG Test Results

Sample	Paste pH	S(T) %	S(SO ₄) %	AP	NP	Net NP	NP/AP	TIC %	CO ₃ NP	NAG pH	NAG pH 7.0
Oxide Fines											
VPOF 01	-	19.40	1.52	558.8	-	-	-	-	-	-	-
VPOF 02	2.8	14.00	1.74	383.1	-11.5	-394.6	<0.1	<0.01	0.8	1.99	106.8
VPOF 03	-	14.00	1.76	382.5	-	-	-	-	-	-	-
Baritic Fines											
BF 01	-	15.40	0.88	453.8	-	-	-	-	-	-	-
BF 02	2.8	21.30	1.89	606.6	-11.1	-617.6	<0.1	<0.01	0.8	1.90	136.7
BF 03	-	22.30	1.19	659.7	-	-	-	-	-	-	-
Till/Overburden											
VPT 31	8.5	0.07	0.02	1.6	65.9	64.3	42.2	0.75	62.5	6.68	1.2
VPT 32	-	0.01	<0.01	0.3	-	-	-	0.23	19.2	-	-
VPT 33	8.6	0.02	0.01	0.3	59.0	58.7	188.8	0.63	52.5	6.44	2.2
VPT 34	-	0.04	0.01	0.9	-	-	-	1.49	124.2	-	-
VPT 35A	7.5	0.70	0.24	14.4	27.5	13.1	1.9	1.72	143.3	5.37	2.2
VPT 35B	-	0.11	0.04	2.2	-	-	-	0.67	55.8	-	-
VPT 36	8.1	0.11	0.04	2.2	56.9	54.7	26.0	0.66	55.0	6.62	1.2
Waste rock											
VPT 18A	-	2.25	0.16	65.3	-	-	-	-	-	-	-
VPT 18B	7.1	4.27	0.15	128.8	29.8	-98.9	0.2	0.83	69.2	3.04	39.7
VPT 19	-	18.50	0.20	571.9	-	-	-	-	-	-	-
VPT 20	5.5	22.90	0.41	702.8	19.3	-683.6	<0.1	0.48	40.0	2.37	85.3
VPT 21	-	7.55	0.74	212.8	-	-	-	0.20	16.7	-	-
VPT 22	5.3	4.81	0.37	138.8	9.7	-129.1	0.1	0.27	22.5	2.38	53.7
VPT 23A	-	8.80	0.49	259.7	-	-	-	0.54	45.0	-	-
VPT 23B	6.2	21.90	0.37	672.8	45.5	-627.3	0.1	1.04	86.7	2.94	66.2
VPT 24A	-	10.90	0.39	328.4	-	-	-	1.27	105.8	-	-
VPT 24B	5.8	25.10	0.40	771.9	45.9	-726.0	0.1	1.20	100.0	2.73	82.7
VPT 25A	-	9.37	0.46	278.4	-	-	-	0.02	1.7	-	-
VPT 25B	5.7	4.54	0.39	129.7	1.8	-127.9	<0.1	0.07	5.8	2.54	38.9
VPT 26A	-	2.05	0.37	52.5	-	-	-	0.05	4.2	-	-
VPT 27A	6.3	3.67	0.11	111.3	20.5	-90.8	0.2	0.54	45.0	3.52	40.7
VPT 27B	-	1.89	0.13	55.0	-	-	-	1.23	102.5	-	-
VPT 27C	6.7	2.00	0.15	57.8	19.8	-38.0	0.3	1.00	83.3	3.12	26.1
VPT 28A	-	2.75	0.08	83.4	-	-	-	0.82	68.3	-	-
VPT 28B	6.3	1.48	0.24	38.8	14.8	-23.9	0.4	0.66	55.0	4.04	10.6
VPT 29A	-	5.40	0.15	164.1	-	-	-	0.91	75.8	-	-
VPT 29B	6.6	6.26	0.21	189.1	29.1	-160.0	0.2	0.96	80.0	3.12	41.6
VPT 30A	-	0.18	0.04	4.4	-	-	-	0.78	65.0	-	-
VPT 30B	6.1	2.48	0.23	70.3	4.4	-65.9	0.1	0.07	5.8	2.47	37.7
VPT 30C	-	1.26	0.07	37.2	-	-	-	0.50	41.7	-	-
VPT 37A	3.6	1.98	0.52	45.6	-4.1	-49.7	<0.1	0.17	14.2	2.60	32.4
VPT 37B	-	2.57	0.18	74.7	-	-	-	1.12	93.3	-	-
VPT 38	-	1.15	0.06	34.1	-	-	-	0.20	16.7	-	-
VPT 39	6.8	2.67	0.13	79.4	19.9	-59.5	0.3	0.52	43.3	3.03	30.3
VPT 40	-	1.04	0.10	29.4	-	-	-	0.03	2.5	-	-
VPT 41	4.8	1.04	0.29	23.4	2.3	-21.1	0.1	0.03	2.5	2.94	12.1
VPT 42	-	11.60	1.01	330.9	-	-	-	0.01	0.8	-	-
VPT 43	4.1	2.16	0.23	60.3	1.8	-58.5	<0.1	0.06	5.0	2.56	32.9
VPT 44	-	1.06	0.06	31.3	-	-	-	0.52	43.3	-	-

AP = Acid potential in kg CaCO₃ equivalent per tonne of material. AP is determined from calculated sulphide sulphur content: S(T) - (SO₄), assuming total conversion of sulphide to sulphate

AP = When calculated AP is 0, it is reported as <0.01.

NP/AP= When AP is 0, then 0.01 value is used for AP to calculate NP/AP.

NP = Neutralization potential in tonnes CaCO₃ equivalent per 1000 tonnes of material.

NET NP = Net neutralization potential = kg CaCO₃ equivalent per tonne of material.

TIC = Total Inorganic Carbon as %C.

Carbonate NP calculated from total inorganic carbon (TIC) assay. TIC value of 0.01 is used in calculation if TIC <0.01%.

NAG = Net Acid Generation (kg H₂SO₄/tonne)

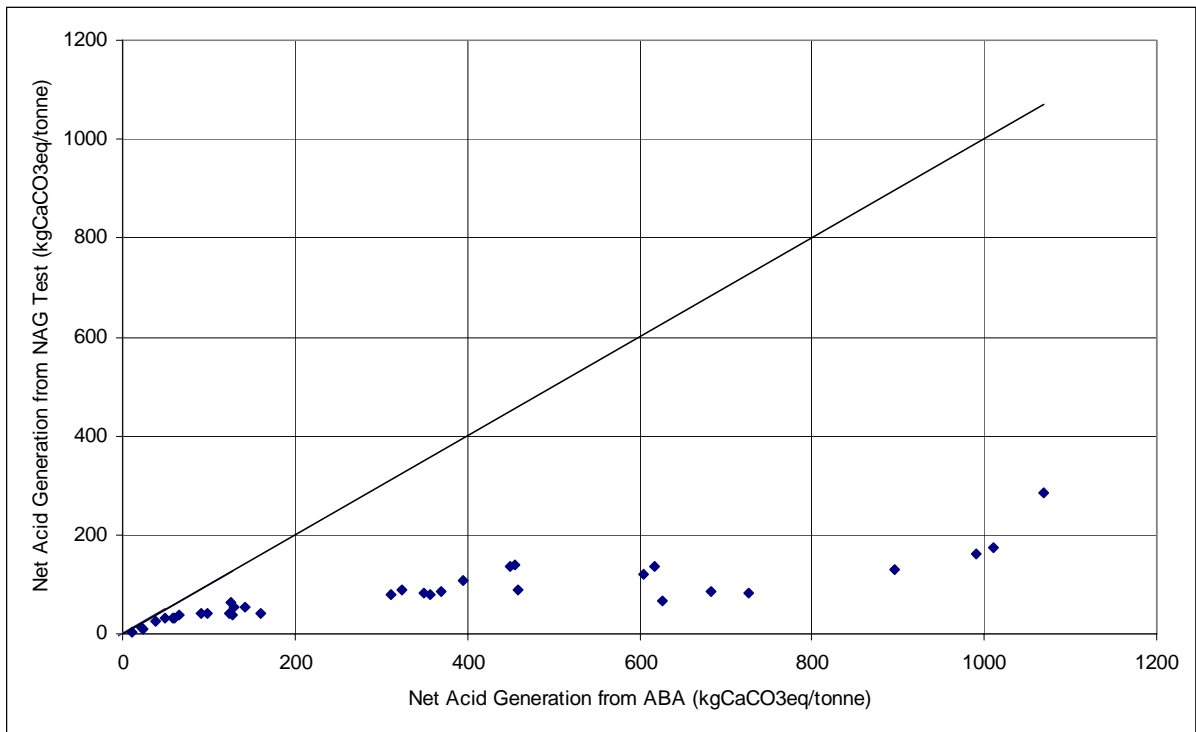


Figure 3.1: Comparison of Net Acid Generation Potential Estimated from NAG Test and Modified Sobek Method

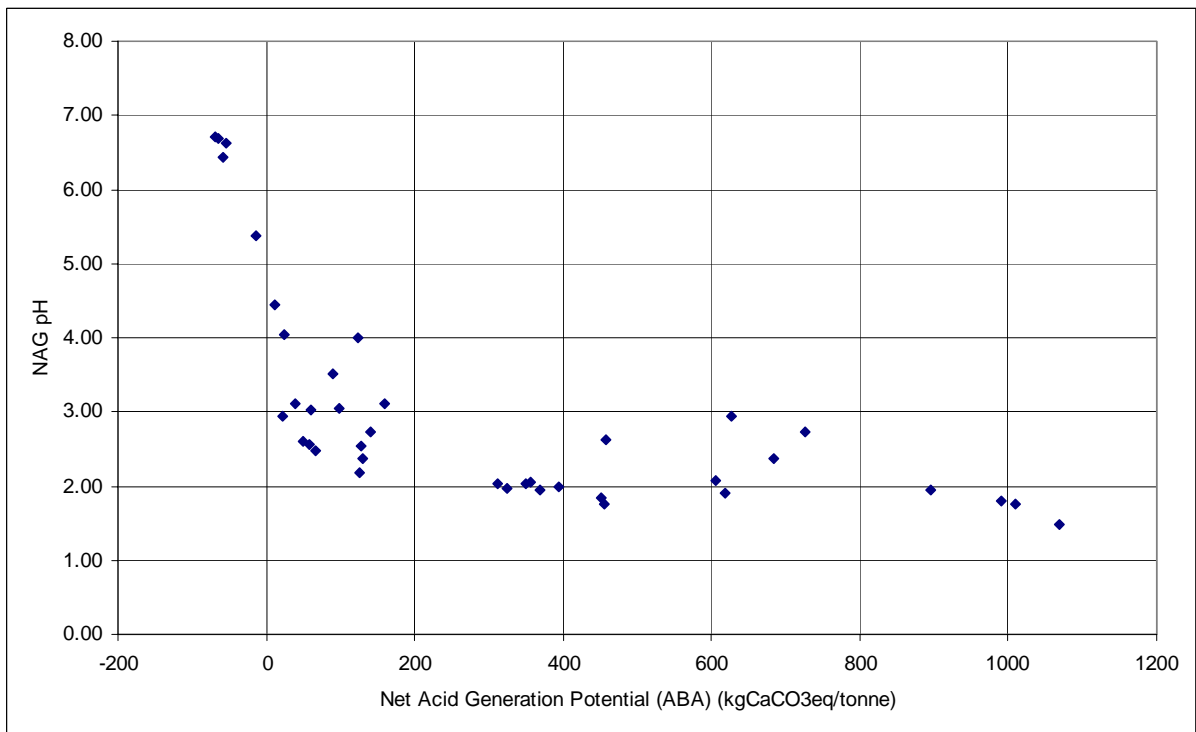


Figure 3.2: Comparison of NAG pH and Net Acid Generation Potential from Modified Sobek ABA Method

3.2 Lime Demand

As noted before, the lime demand test measures the lime required to neutralize acidity contained in the rock sample to a pH in excess of 9.5. The results from the lime demand tests are summarised in Table 3.2. Lime addition and pH values recorded during the tests are provided in Appendix D.

The lime demand of the Oxide and Baritic Fines ranged from 4.9 to 10 kg Ca(OH)₂ per tonne, with an average of 8.4 kg Ca(OH)₂ per tonne. The waste rock lime demand ranged from 0.14 to 3.1 kg Ca(OH)₂ per tonne, with an average of 0.84 kg Ca(OH)₂ per tonne. The till/overburden samples had a very low lime demand, with the exception of Sample 35A. That sample was an oxidized material that was encountered in the overburden dump, and may have originated from over-excavation below the overburden.

Table 3.2: Summary of Lime Demand Test Results

Sample	Paste pH	Conductivity (mS/cm)	S(SO ₄) (%)	Lime Demand (kg Ca(OH) ₂ /tonne)
Till / Overburden				
VTP31	7.65	0.333	0.02	0.20
VTP32	7.80	0.24	0.01	0.13
VTP33	7.39	0.228	0.01	0.25
VTP35A	6.14	1.38	0.24	0.80
VTP35B	7.78	0.64	0.04	0.25
VTP36	7.22	0.741	0.04	0.35
Average				0.36
Oxide / Baritic Fines				
BF01	3.50	5.25	0.88	8.50
BF02	2.40	6.21	1.89	9.80
BF03	3.20	4.69	1.19	4.85
VPOF01	2.60	5.80	1.52	8.95
VPOF02	2.40	5.76	1.74	8.35
VPOF03	2.30	5.37	1.76	10.0
Average				8.41
Waste Rock				
VTP18A	6.56	1.499	0.16	0.35
VTP18B	6.57	1.49	0.15	0.40
VTP19	6.06	1.151	0.20	0.30
VTP20	5.98	2.51	0.41	1.5
VTP21	3.86	2.10	0.74	3.0
VTP22	4.80	1.864	0.37	1.53
VTP23A	5.71	2.28	0.49	0.90
VTP23B	6.22	1.875	0.37	0.65
VTP24A	6.08	0.958	0.39	0.40
VTP24A	6.32	1.935	0.40	0.525
VTP25A	6.02	0.809	0.46	0.70
VTP25B	5.42	1.27	0.39	1.4
VTP25B	6.69	1.72	0.10	0.25
VTP26A	3.48	2.40	0.37	2.0
VTP27A	6.02	0.98	0.11	0.55
VTP27B	6.82	1.77	0.13	0.25
VTP27C	6.68	1.59	0.15	0.40
VTP28A	6.12	0.65	0.08	0.25
VTP28B	6.04	1.677	0.24	0.55
VTP29A	6.62	1.028	0.15	0.35
VTP29B	6.00	1.33	0.21	0.55
VTP30	6.98	0.446	-	0.20
VTP30A	7.19	0.582	0.04	0.30
VTP30B	6.08	1.715	0.23	0.65
VTP30C	6.70	0.685	0.07	0.35
VTP37A	3.15	1.884	0.52	2.0
VTP37B	5.43	1.973	0.18	1.05
VTP39	6.50	1.077	0.13	0.35
VTP40	4.25	0.20	0.10	0.40
VTP41	4.30	1.78	0.29	1.68
VTP42	2.50	2.77	1.01	3.1
VTP43	4.23	0.61	0.23	0.70
VTP44	6.94	0.59	0.06	0.15
Average				0.84

The wide range in lime demand implies that, if an average lime or limestone addition were used for all of the waste rock, a significant proportion would be ‘over-limed’ and another proportion would be ‘under-limed’. The former would be wasteful and the latter could leave large portions of the backfill acidic, and thereby nullify the benefits of the remaining lime or limestone addition. It is concluded that a field control program will be needed to determine the appropriate lime or limestone addition rate for each segment of the waste rock.

The field and NAG parameters were compared to the lime demand to assess the potential for using rapid tests in the field control program. It was found that the lime demand correlated well to a combination of paste conductivity and paste pH, as follows:

$$\text{Lime Demand} = 3.8443 * (\text{Paste Conductivity} / \text{Paste pH})$$

Where: *Lime demand* is given in kg Ca(OH)₂ per tonne
Paste conductivity is in units of mS/cm.

The correlation is illustrated in Figure 3.3. Similar correlations have been used to determine the appropriate rate of lime or limestone addition at other sites.

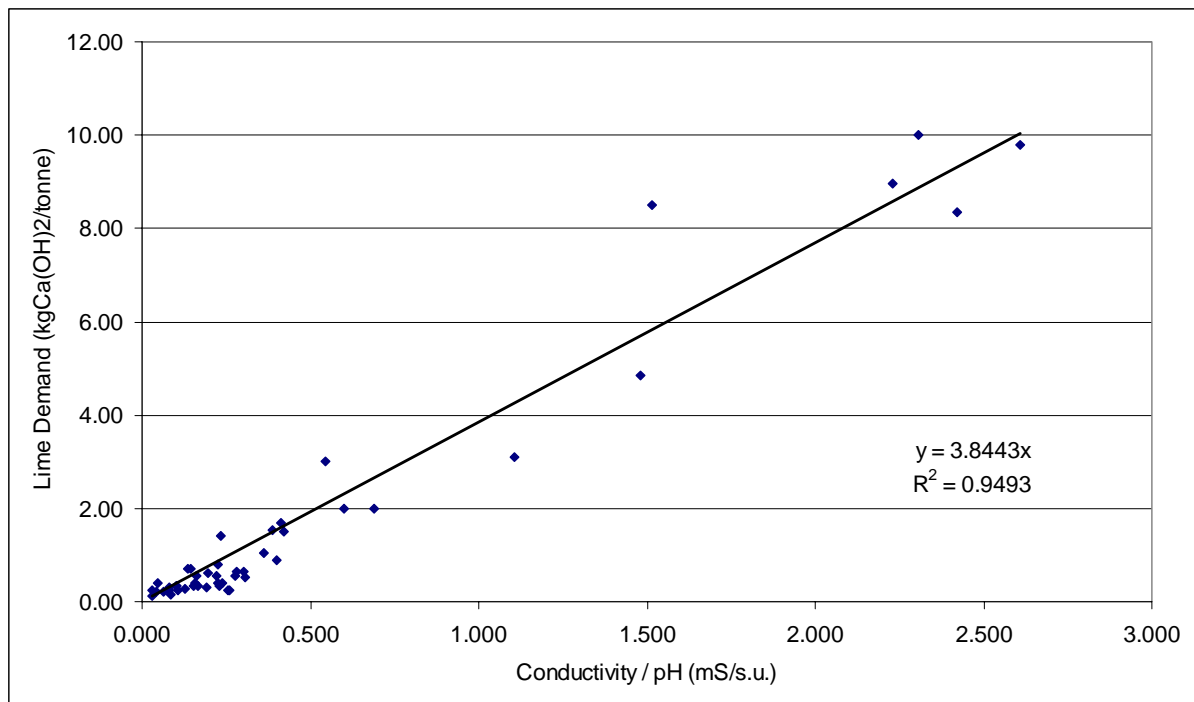


Figure 3.3: Linear Regression of Lime Demand and the Ratio of Conductivity to pH

3.3 Leach Extraction Tests

Complete results from the leach extraction tests are provided in Appendix E, and are summarised in Table 3.3.

The lime amendment to the low lime demand composite (VG1) resulted only in a marginal increase of the leachate pH. However, about 83 percent of the zinc was removed from solution. Similarly, while the pH change affected by the corresponding limestone amended test was minimal, about 71 percent of the dissolved zinc was removed. The lime amendment proved to be more effective at removing manganese.

As shown in Table 3.3, the pH of the VG2 samples amended with lime did not change significantly. Acidity measurements for the test on un-amended VG2 material were used to calculate an actual lime demand of about 14.1 kg Ca(OH)₂ per tonne, which is much higher than the lime demand calculated from the individual samples. Only about 6 percent of the zinc was removed whereas about 40 percent of the iron was removed from solution. The limestone amended test on composite VG2 performed significantly better than the lime amended test, with zinc and iron removals of about 73 and 98 percent respectively. However, it is evident from the three tests that the acidity of the VG2 material was highly variable, so the results favouring limestone amendment are not conclusive.

Table 3.3: Summary of Leach Extraction Test Results

Sample	Units	As Is		With Lime		With Limestone	
		VG1	VG2	VG1	VG2	VG1	VG2
pH		6.72	2.25	6.94	2.28	6.84	4.59
Conductivity	mS/cm	0.92	4.93	1.08	4.25	1.09	1.31
Alkalinity	mg CaCO ₃ /L	9.5	0.0	21.5	0.0	21.0	1.0
Acidity	mg CaCO ₃ /L	53	6340	16	4040	29	435
Sulphate	mg/L	1493	7546	1382	5654	1429	2211
Dissolved Metals (mg/L)							
Aluminum	Al mg/L	<0.20	148	<0.20	110	<0.20	1.29
Copper	Cu mg/L	0.144	39.9	0.030	28.8	0.042	0.195
Iron	Fe mg/L	<0.030	1840	0.053	1080	0.043	45.1
Lead	Pb mg/L	0.843	<0.25	0.247	<0.25	0.317	2.96
Magnesium	Mg mg/L	83.6	61.0	72.2	60.5	78.4	114
Manganese	Mn mg/L	16.2	22.7	8.05	30.5	11.5	63.7
Zinc	Zn mg/L	40.7	585	7.09	549	12.0	158

3.4 Column Tests

Complete results for the column tests are provided in Appendix F and are summarised in Table 3.4. There were no column tests on un-amended material. However, the total mass of solute released in the leach extraction tests can be used to calculate maximum solute concentrations that could develop

in the un-amended material. These estimated concentrations are shown in the “as is” columns in Table 3.4.

Unfortunately, because the samples for the leach extraction and the column tests were prepared on the same basis as for the leach extraction tests, the VG2 samples received insufficient alkali amendment. However, even the low quantities of alkali amendment had a beneficial effect on metal solubility.

Table 3.4: Summary of Column Test Results

Parameter	Units	VG1					VG 2				
Alkali		As Is*	Lime		Limestone		As Is*	Lime		Limestone	
Time	Days	0	15	21	15	21	0	15	21	15	21
Redox	(mV)	-	188	282	176	277	-	274	335	350	264
pH		-	6.43	6.46	6.36	7.00	-	2.32	2.64	4.50	4.79
Conductivity	(uS/cm)	-	1,445	1,424	1,312	1,336	-	6910	8300	1757	1650
Alkalinity	(mg CaCO ₃ /L)	-	40.0	33.0	43.5	41.0	-	0.0	0.0	0.0	1.5
Acidity (pH 8.3)	(mg CaCO ₃ /L)	787	87	73	79	46	95,100	13,995	9,030	407	264
Sulphate	(mg/L)	22,395	2,800	1,845	2,490	1,700	113,190	14,165	10,565	3,065	21,135
Metals											
Aluminum	mg/L	-	<0.20	1.88	<0.20	0.26	2,220	474	302	1.04	0.87
Cadmium	mg/L	3.5	0.307	0.262	0.253	0.180	12	2.58	1.68	0.374	0.292
Calcium	mg/L	-	478	441	431	405	-	293	255	455	450
Cobalt	mg/L	4.6	0.508	0.362	0.479	0.298	16	4.33	2.73	2.94	1.76
Copper	mg/L	2.2	0.678	0.650	0.452	0.364	599	96.5	59.0	0.134	0.087
Iron	mg/L	-	0.464	2.680	0.192	0.416	27,600	4,060	2,650	6.62	5.20
Magnesium	mg/L	1,254	364	194	352	198	915	260	182	415	218
Manganese	mg/L	243	52.4	39.4	50.7	35.4	170	114	77	199	115
Zinc	mg/L	611	52.7	41.4	45.8	29.7	8,775	1,820	1,190	224	147

Note: * Concentrations estimated from results of the leach extraction tests

The limestone amended tests on both composite VG1 and composite VG2 showed consistently lower metal concentrations than the lime amended tests. One explanation is that the lime, which is significantly more reactive than the limestone, was encapsulated by the formation of insoluble precipitates (e.g. iron-oxy-hydroxides). The encapsulated lime might be available over longer time periods than those tested.

The limestone is unlikely to raise the pH above about 7 to 8, which means that the zinc concentrations will be reduced to about 30-50 mg/L. However, because of its buffering capacity the addition of excess limestone has no detrimental effect. Its limited solubility also makes it ideally suited for use in the unsaturated zone, where it is important to retain the alkali at the location where acidity is being generated. Limestone also leads to the formation of carbonate minerals which generally are more stable than corresponding hydroxides. With lime, it would be possible to achieve a higher porewater pH and thereby a better zinc removal. Another advantage offered by lime is that it does not generate carbon dioxide, which can adversely affect contaminant solubilities. On the

other hand, lime requires more stringent addition control since excess amendment will lead to very high pH conditions which may adversely affect the solubility of some contaminants.

3.5 Spatial Variability

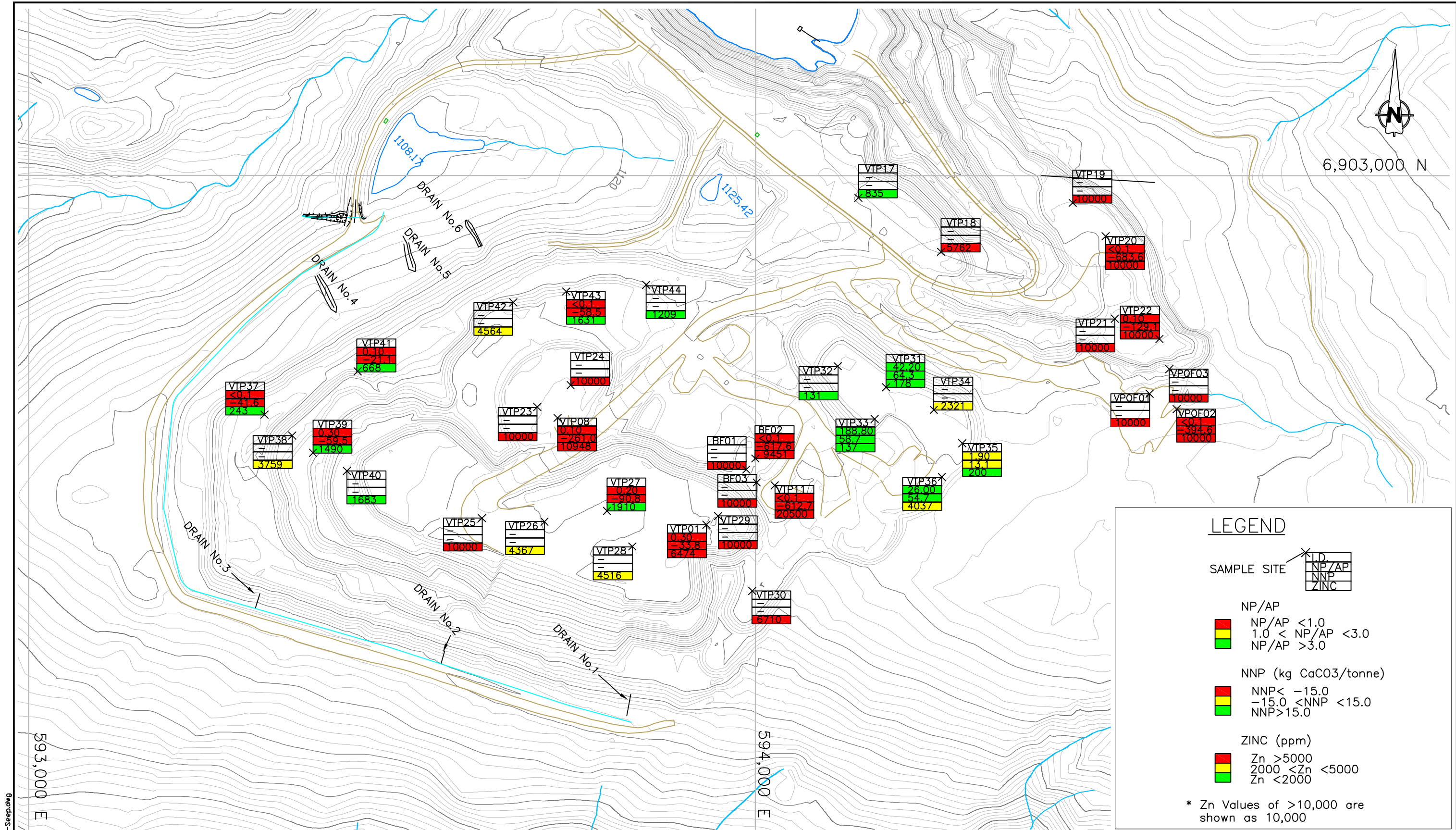
As noted before, one of the objectives of the sampling and testing program was to assess the variability in the Vangorda waste rock dump. The waste rock acid generating properties together with zinc content are shown in plan on Figure 3.4. For ease of interpretation, the results have been colour coded as shown in the legend of the drawing.

The acid generating properties of the till dump are clearly distinguishable from the main waste rock dump. Similarly the high acid generating properties of the Baritic Fines and the Oxide Fines clearly distinguishes these materials from the rest of the waste rock. All of the waste rock is net acid generating and there do not appear to be specific areas that are readily distinguishable from others.

While limited observations are available, the NAG test results shown in Figure 3.5 confirm that the waste rock is net acid generating. The NAG test results also indicate that there may be a zone of higher acid generation potential at an elevation of about 1160 m asl, i.e. the first plateau on the western side of the ramp.

With respect to the zinc content (Figure 3.4), the same conclusion can be drawn. There appears to be some elevated zinc concentrations at two locations on the till dump. However these are likely due to over excavation during segregation of the overburden. In contrast, the zinc content of the main dump is variable. There appears to be a zone on the lower lifts of the eastern portion of the dump that has intermediate to low zinc values. Rock located in the upper plateau generally has higher zinc contents.

The results in Figure 3.4 and Figure 3.5 only show the properties of the surface or outer layer. As an indication of variability with depth, sulphur content results obtained from drill hole samples are shown in Figure 3.6. The results for Drill Hole VG30M-4, installed approximately in the centre of the topmost plateau of the dump (near VTP27 in Figure 3.4), indicate minimal variability within the main part of the dump, with narrow zones where the sulphur content is elevated above the average. The results for Drill Hole VG10M-4, installed on the plateau to the west of the ramp (near VTP24) identified a zone of high sulphide rock, which is consistent with the test pit results.



LEGEND

SAMPLE SITE

X

ID

NP/AP

NNP

ZINC

NP/AP

NP/AP <1.0

1.0 < NP/AP <3.0

NP/AP >3.0

NNP (kg CaCO3/tonne)

NNP < -15.0

-15.0 < NNP <15.0

NNP >15.0

ZINC (ppm)

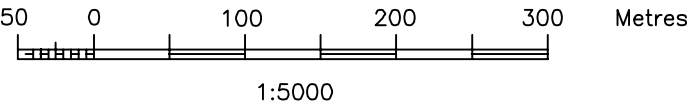
Zn >5000

2000 < Zn <5000

Zn <2000

* Zn Values of >10,000 are shown as 10,000

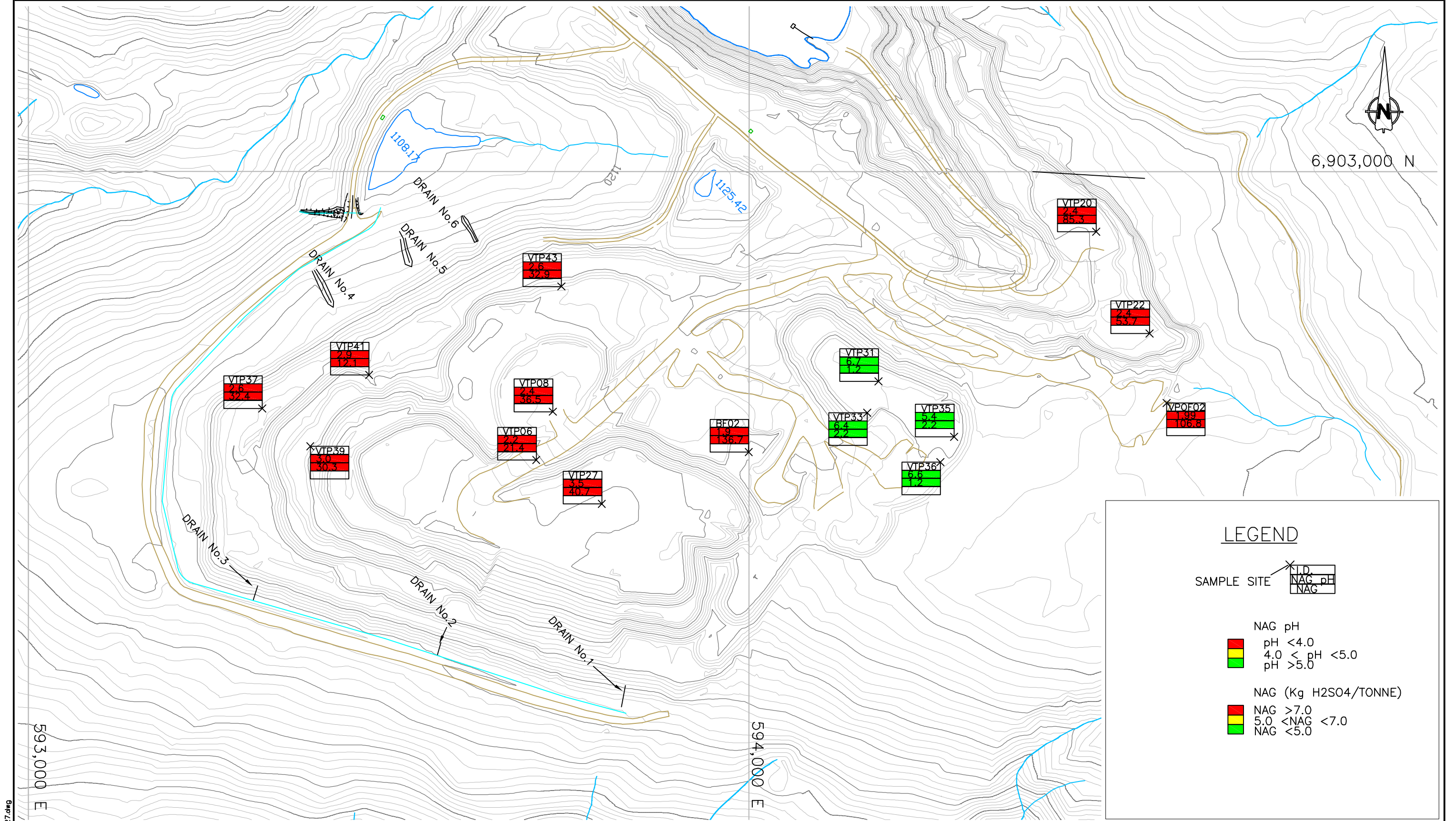
Contour Interval: 2m
Date of Photography: 03/07/25
Scale of Photography: 1:20000
Survey control derived from existing 1:20000 photography
Survey control based on: UTM Projection, NAD27
Compiled by The ORTHOSHOP, Calgary, September 2003
WO 8856



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VANGORDA PLATEAU MINE			
VANGORDA DUMP SURFACE DISTRIBUTION OF ACID GENERATION AND ZINC			
PROJECT NO.	DATE	APPROVED	FIGURE
1CD003.048	Nov. 2004		3.4



LEGEND

SAMPLE SITE

X

NAG

pH

NAG pH

pH <4.0

4.0 < pH <5.0

pH >5.0

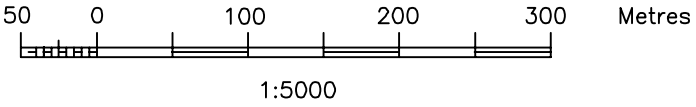
NAG (Kg H2SO4/TONNE)


NAG >7.0

5.0 <NAG <7.0


NAG <5.0

Contour Interval: 2m
Date of Photography: 03/07/25
Scale of Photography: 1:20000
Survey control derived from existing 1:20000 photography
Survey control based on: UTM Projection, NAD27
Compiled by The ORTHOSHOP, Calgary, September 2003
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VANGORDA PLATEAU MINE

VANGORDA DUMP SURFACE DISTRIBUTION
OF NET ACID GENERATION (NAG)

PROJECT NO. 1CD003.048	DATE Nov. 2004	APPROVED	FIGURE 3.5
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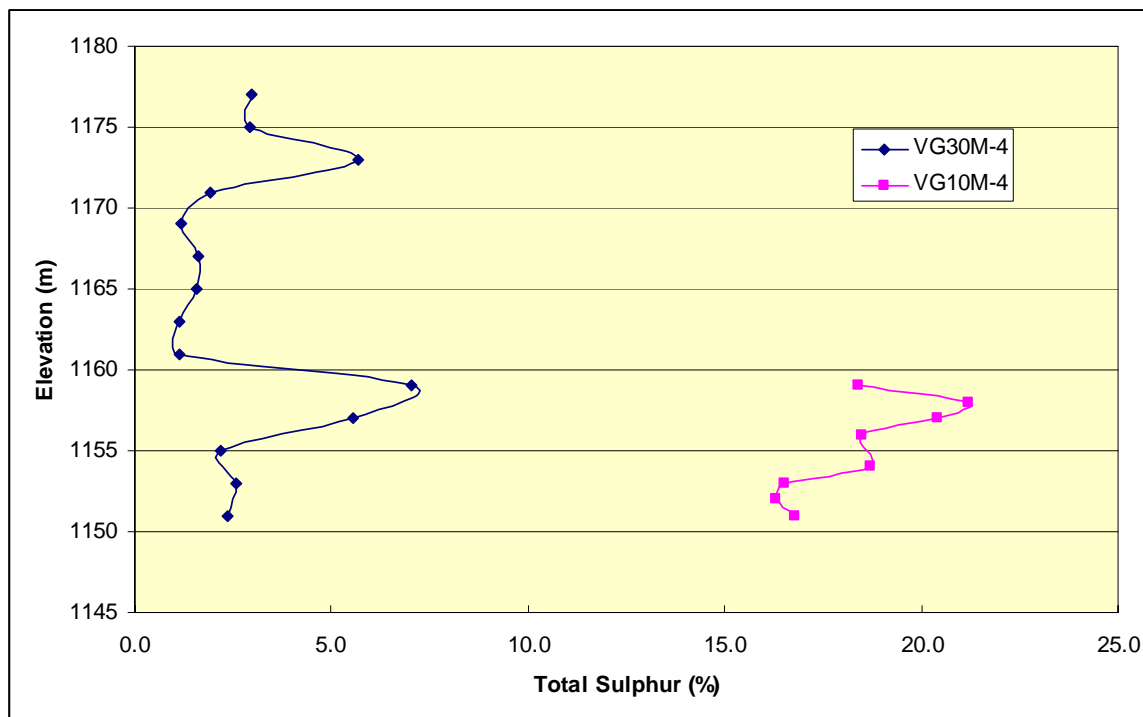
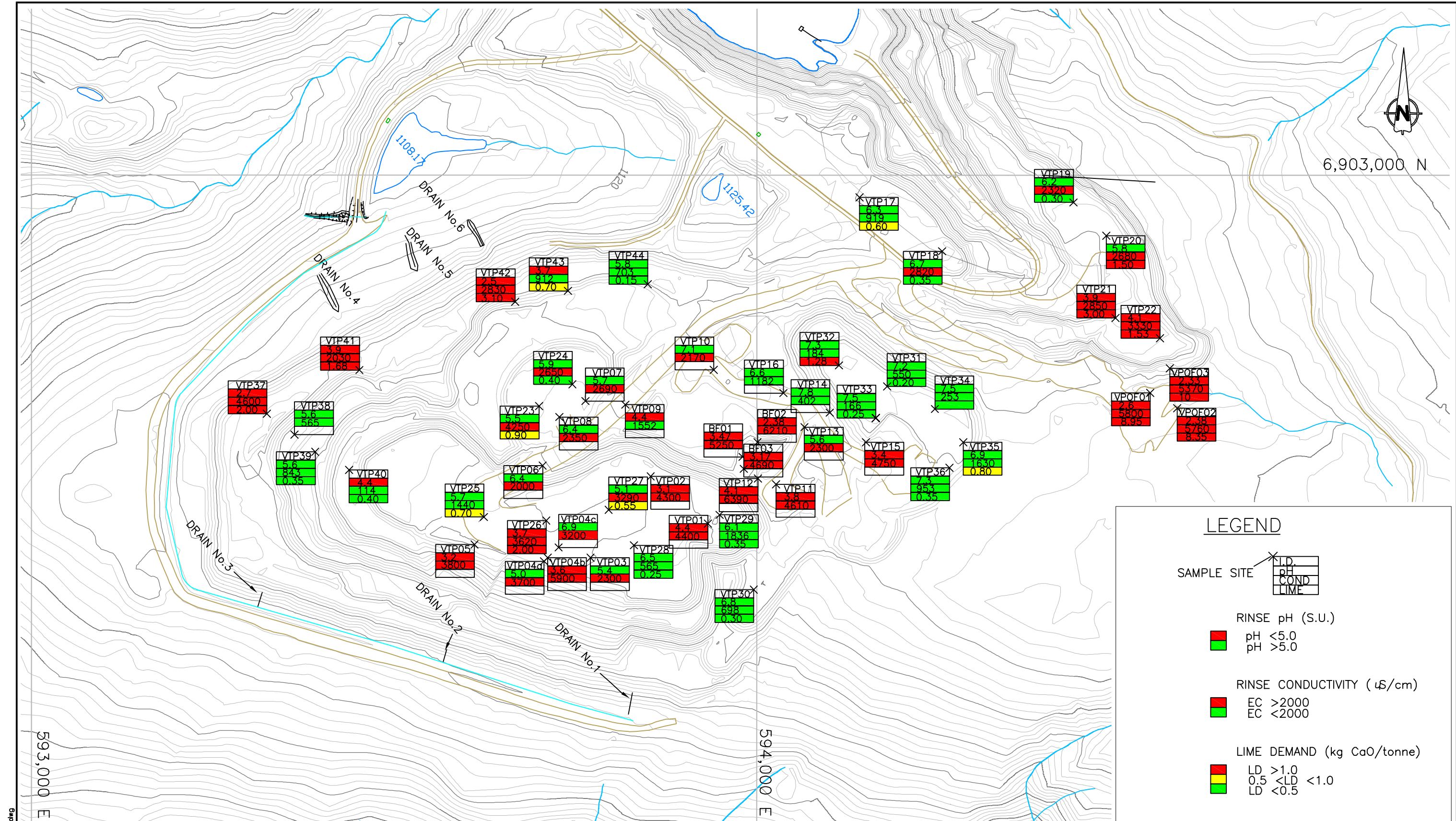


Figure 3.6: Vertical Variability of Sulphur as Indicated by Drill Hole Samples

The distribution of the estimated lime demand for the waste rock is illustrated in Figure 3.7, which also shows the paste parameters. The Till Dump, and the Baritic and Oxide Fines are clearly distinguishable by low and high lime demands, respectively. On the surface of the main waste rock dump, the lime demand and paste properties vary over short distances and it is difficult to distinguish continuous areas of clearly high or low lime demand. The paste parameters obtained from the drill hole samples are shown in Figure 3.8 and Figure 3.9. It should be noted that these parameters represent rock that had been crushed during drilling. Nonetheless, the results indicate that the outer 15 m of the waste rock likely has been influenced by oxidation and hence would require higher lime or limestone amendments.



LEGEND

SAMPLE SITE

I.D.

PH

COND

LIME

RINSE pH (S.U.)

pH <5.0

pH >5.0

RINSE CONDUCTIVITY (µS/cm)

EC >2000

EC <2000

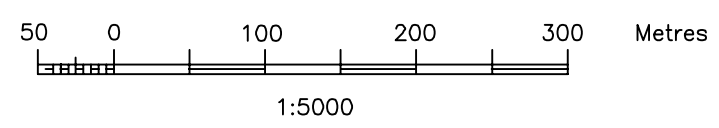
LIME DEMAND (kg CaO/tonne)

LD >1.0

0.5 <LD <1.0

LD <0.5

Contour Interval: 2m
Date of Photography: 03/07/25
Scale of Photography: 1:20000
Survey control derived from existing 1:20000 photography
Survey control based on: UTM Projection, NAD27
Compiled by The ORTHOSHOP, Calgary, September 2003
WO 8856



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VANGORDA PLATEAU MINE			
WASTE ROCK PASTE PARAMETERS AND ESTIMATED LIME DEMAND			
PROJECT NO.	DATE	APPROVED	FIGURE
1CD003.048	Nov. 2004		3.7

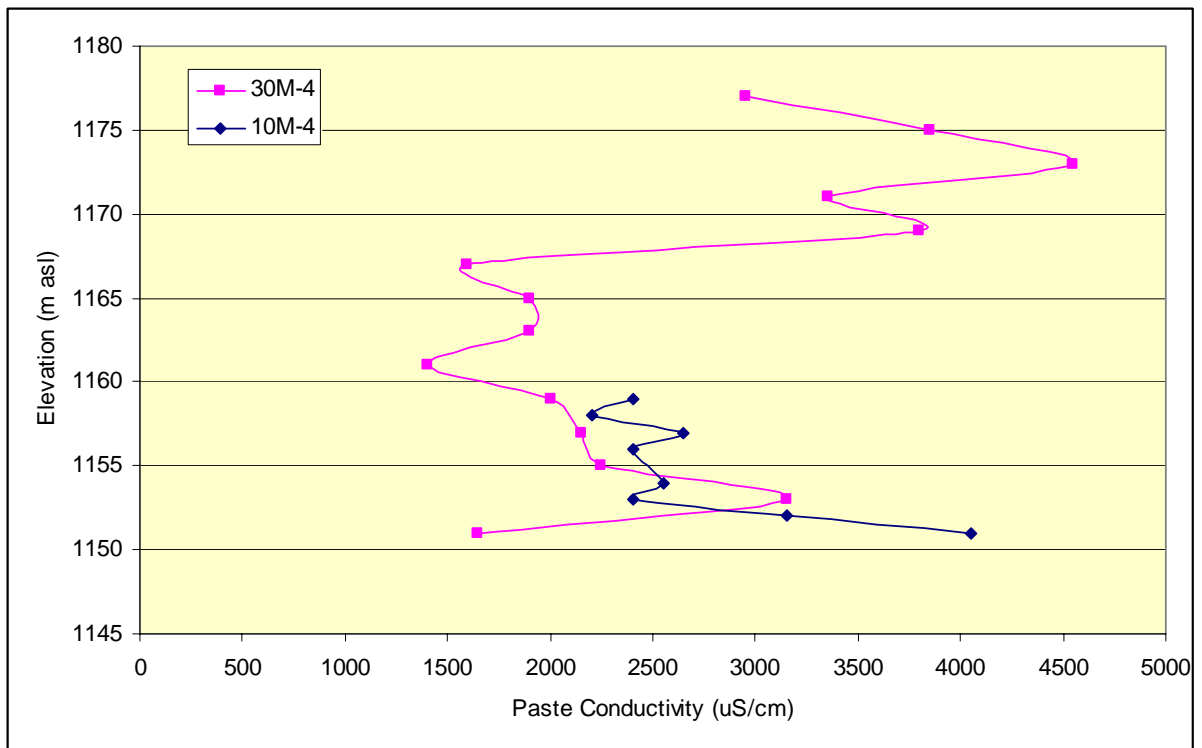


Figure 3.8: Paste Conductivity Measured in Drill Hole Samples

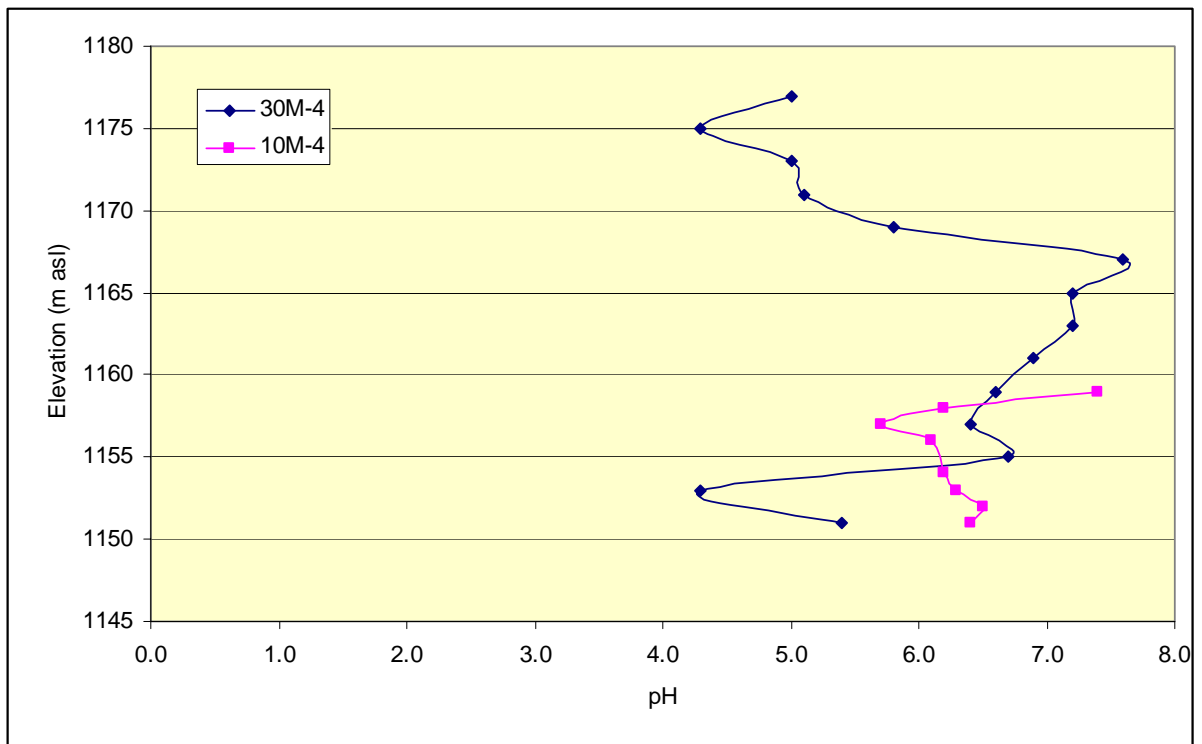


Figure 3.9: Paste pH Measured in Drill Hole Samples

4 Conceptual Backfilling Program

4.1 Water Management

Based on current estimates, the Vangorda Pit currently contains about 2.3 million m³ of water with an average zinc concentration of about 115 mg/L. Two options are available for dealing with this water. The first option would be to dump the backfill directly into the water and treat the contaminated water as the level in the pit rises. The second is to pump and treat the entire pit lake, removing all the water, and then depositing the waste rock into the dry pit.

The advantages and disadvantages of each option are summarised in Table 4.1. The long term benefits that would be gained from removing the water prior to the backfilling appear to outweigh the short term benefits from depositing waste rock directly into the water. It should be noted that previous examples of large scale pit backfilling programs have been conducted using a ‘dry placed’ approach.

Table 4.1: Comparison of Two Options for Vangorda Pit Water

	Option 1 Deposit Into Water	Option 2 Remove Water First
Advantages	<ul style="list-style-type: none"> No delay before backfilling can commence Minimizes water handling and risks associated with water management 	<ul style="list-style-type: none"> Can compact fill as it is placed reducing long term settlement Compacted fill will provide lower permeability Will allow in-pit sampling of backfill to control and monitor lime or limestone amendments
Disadvantages	<ul style="list-style-type: none"> Difficult to remove zinc currently in the pit water, and impacts on long-term groundwater may be significant Lower bulk density of fill will result in higher permeability, lower storage capacity and greater long-term settlement Long-term settlement may be problem for Vangorda Creek 	<ul style="list-style-type: none"> Backfilling will be delayed until all water has been removed from the pit

Managing inflows to the pit during relocation would further require that a sump be installed and maintained within the Vangorda Pit. The sump would be raised to follow the level of the backfill to minimise water accumulation during operations.

Seepage from the waste rock is expected to continue throughout the relocation works until removal is complete. It is further anticipated that the quality of the seepage from the waste rock may worsen as oxidized waste rock is disturbed and exposed to infiltration. Therefore, pumping from Little Creek pond would need to be maintained throughout relocation. The water from Little Creek pond would be pumped directly to the Grum/Vangorda water treatment plant.

4.2 Backfilling Sequence

A three-dimensional and a sectional view of the Vangorda Pit are shown in Figure 4.1 and Figure 4.2 respectively. The latter figure also shows the projected flooding elevation, estimated to be at or above 1130 m asl, which is the elevation of the spill point from the pit. Below this elevation, a flooded volume of about 5.77 million m³ is available.

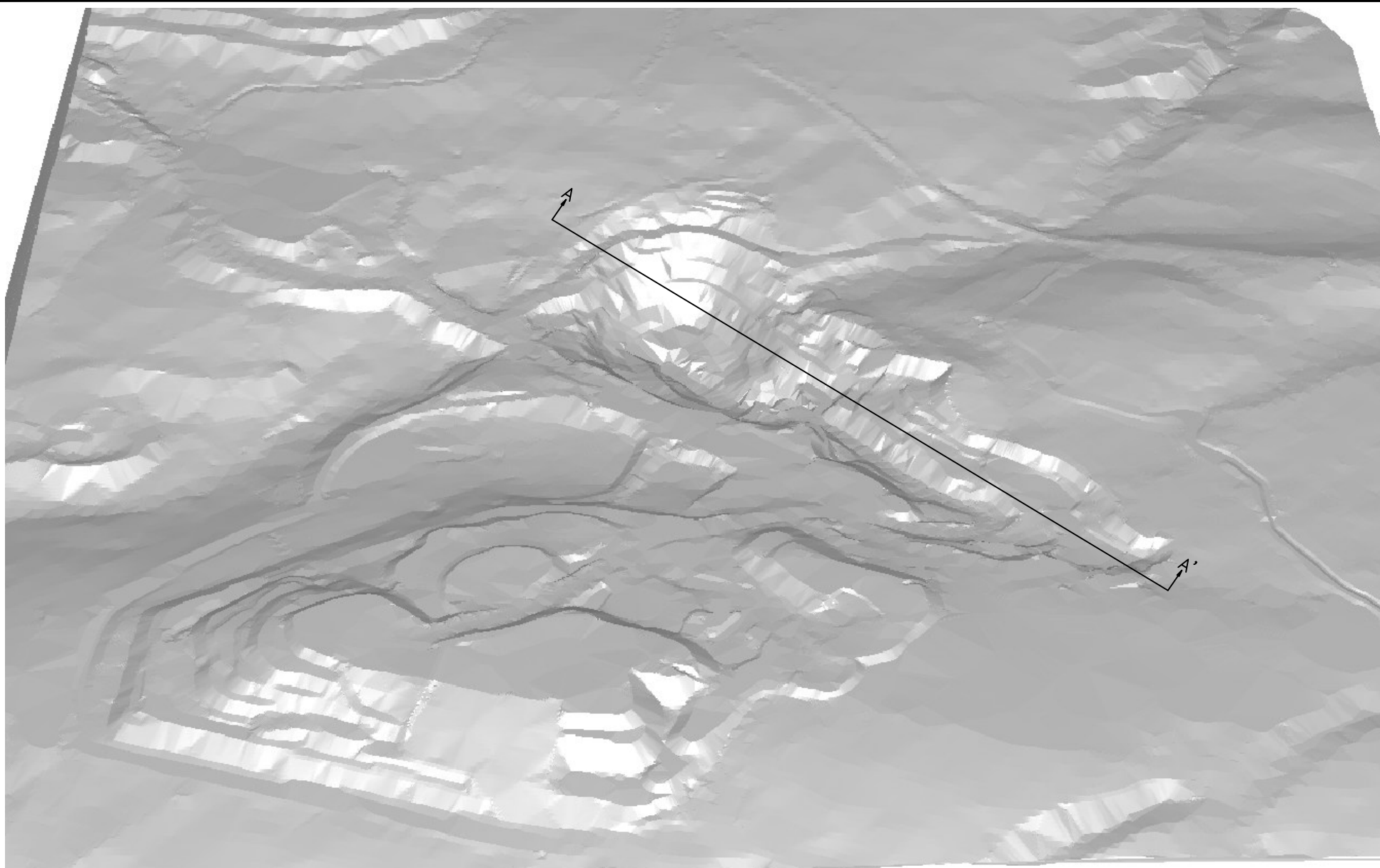
To best utilize the volume available below the pit spill point, backfilling could proceed in the following sequence:

1. The Baritic and Oxide Fines would be placed first at the base of the pit, because of their high potential for acid generation. Combined, these materials represent an estimated total volume of about 286,000 m³. The elevation in the pit of the Baritic and Oxide Fines is shown in Figure 4.3 and Figure 4.4.
2. The high sulphide area adjacent the west of the ramp to the dump, together with the material from the ore transfer pad, would be relocated next. Together, these materials represent between 1.3 and 1.5 million m³.
3. The main waste rock dump would then be mined, from the top down, in lifts of about 5 m.
4. Finally the till dump would be placed on top of the backfill to form a cover. The till would be placed in lifts of about 0.3 to 0.5 m, and compacted. The final lift would not be compacted and would serve as a growth medium for vegetation. The available till is expected to provide a final cover about 2.6 m thick.

The final landform, after all the waste rock and the till have been relocated, is shown as a three-dimensional view in Figure 4.5, and a section through the backfill is provided in Figure 4.6. It is recommended that the waste rock be placed in layers between 0.5 and 1 m thick. Experience at two other sites has shown that this approach allows significant compaction by the construction traffic.

Once the waste rock has been removed to original soil, the soil should be tested for metal contamination and if necessary be stripped and backfilled above the waste rock but below the final till cover. The final landform shown in Figure 4.5 does not account for the relocation of any soil.

The Oxide Fines Management Plan (SRK, 2004) has identified the possibility of incorporating the Faro Oxide Fines and low grade ore stockpiles in the Vangorda Pit backfill. Previous estimates suggest that the total volume of Oxide Fines and low grade ore would be in the order of 0.66 to 1.77 million m³. This means that it would be possible to place the Faro Oxide Fines and low grade materials together with the Vangorda Baritic and Oxide Fines within the flooded regime of the pit. The additional high sulphide material identified in the waste rock dump would also be contained within the flooded zone. The additional volume however would result in an increase of the final surface on average by about 2 to 4 m. This additional volume could be accommodated within the areas adjacent the channel that would carry Vangorda Creek across the backfill, by creating gently sloped knolls which would result in a more natural topography.



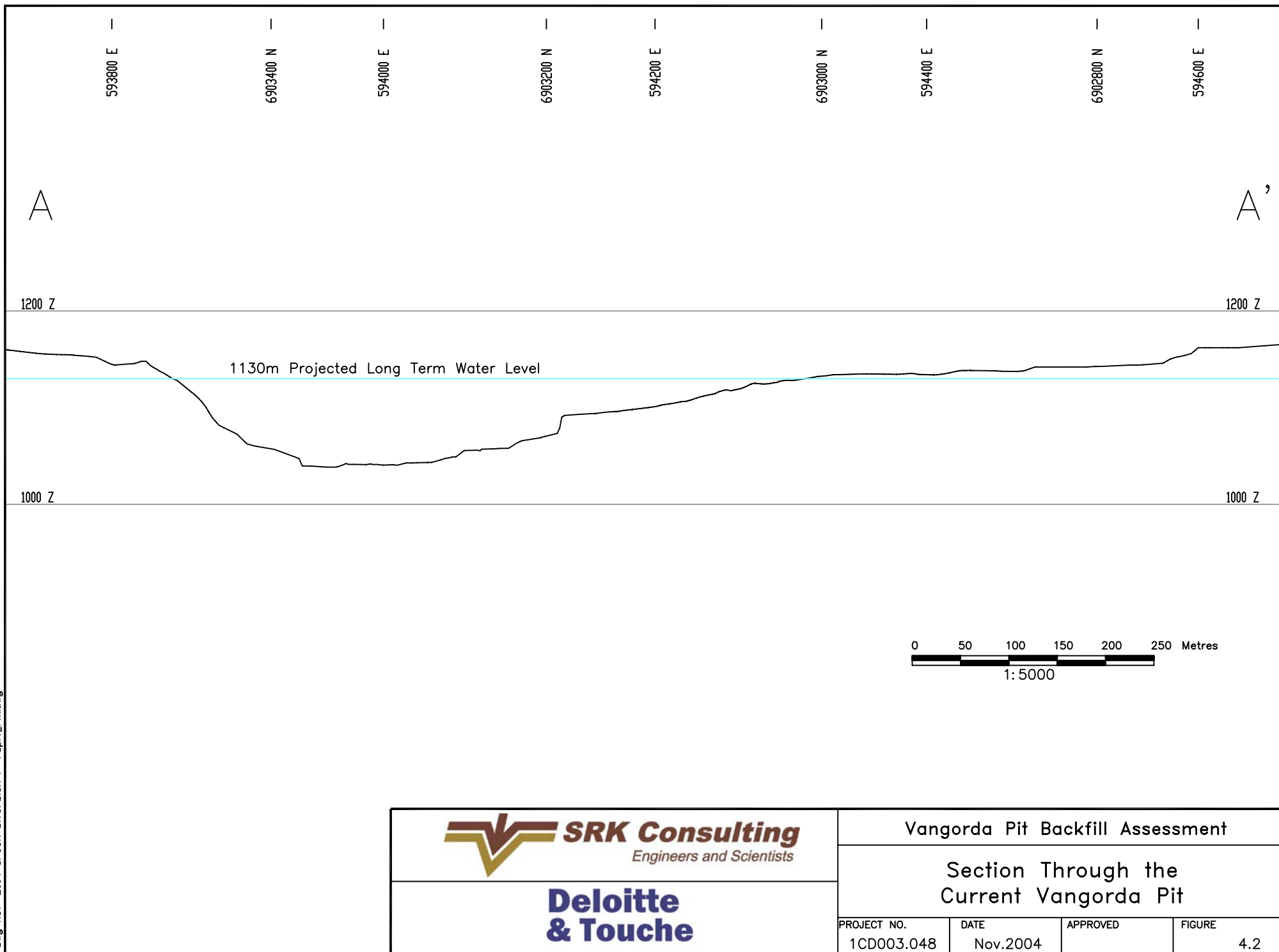
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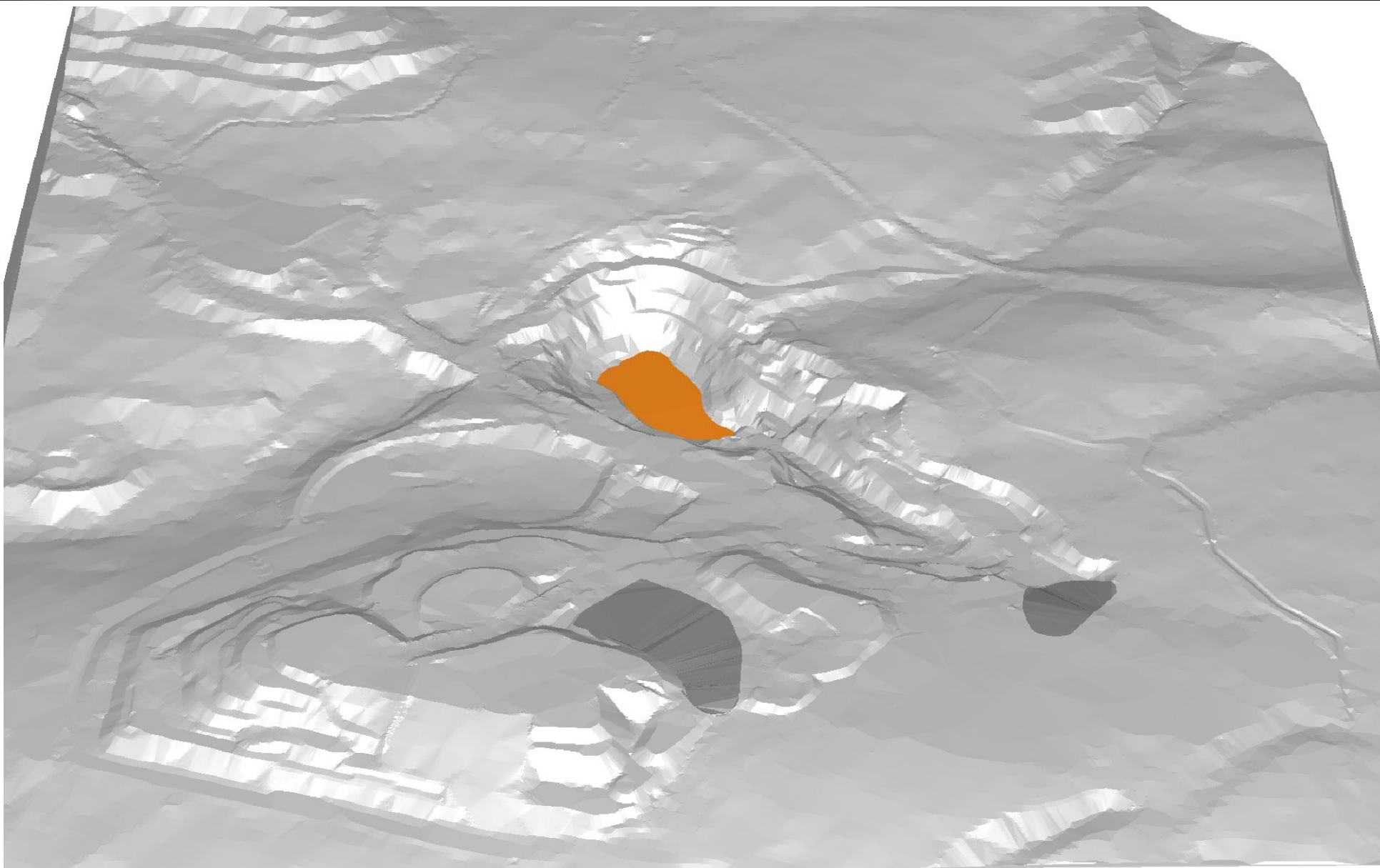
Vangorda Pit Backfill Assessment

Three Dimensional View of Current
Vangorda Pit and Waste Rock Dump

PROJECT NO.	DATE	APPROVED	FIGURE
1CD003.048	Nov.2004		4.1

Dwg Ref: 2004 Creek Diversion / VGpitt_Fill.dwg



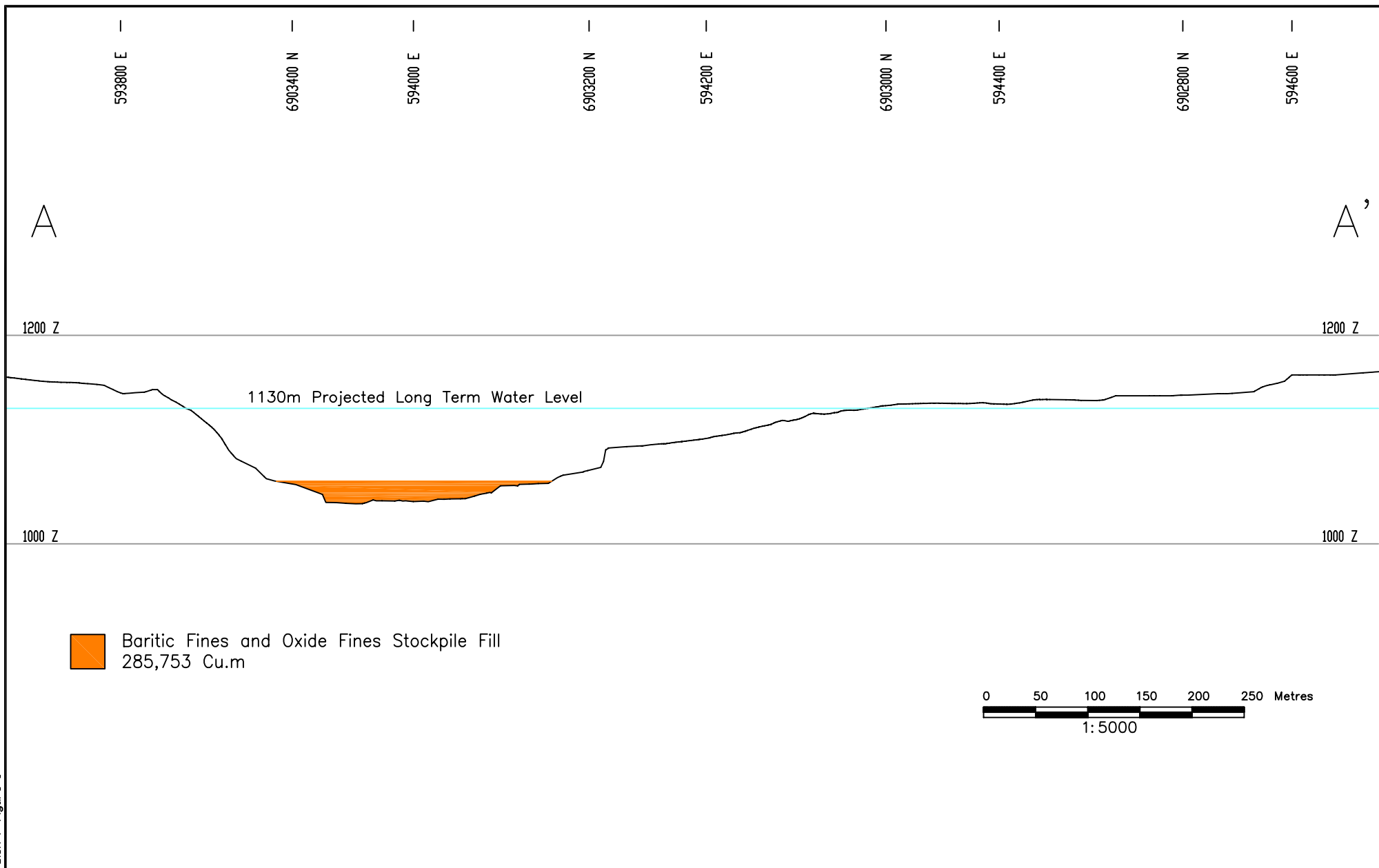




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Vangorda Pit Backfill Assessment

Three Dimensional View After Backfilling
of the Baritic and Oxide Fines

PROJECT NO.	DATE	APPROVED	FIGURE
1CD003.048	Nov.2004		4.3



 SRK Consulting <i>Engineers and Scientists</i>	Vangorda Pit Backfill Assessment		
	Section Through the Vangorda Pit After Backfilling of the Baritic and Oxide Fines		
	PROJECT NO.	DATE	APPROVED
	1CD003.048	Nov.2004	FIGURE 4.4

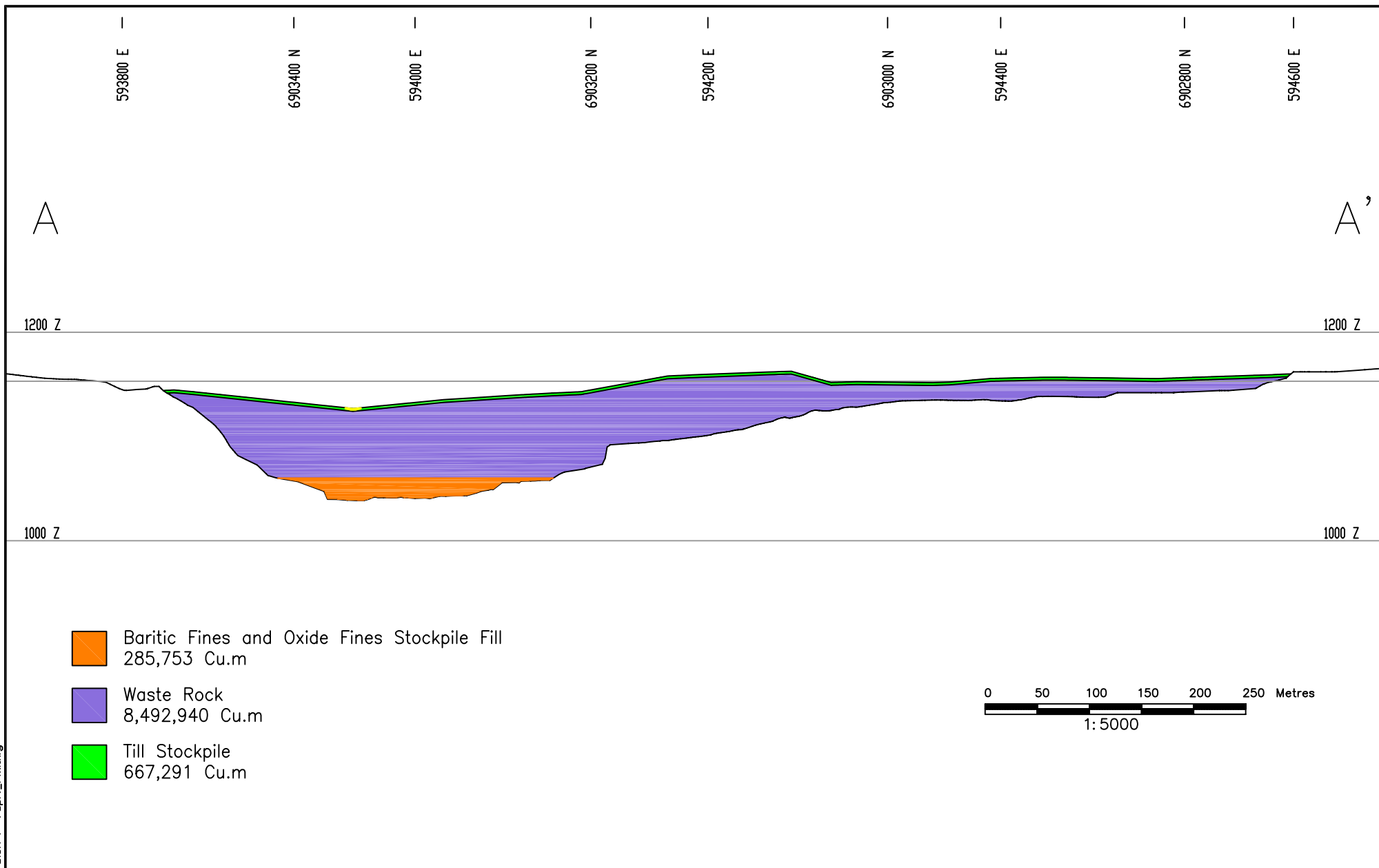




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Vangorda Pit Backfill Assessment

Three Dimensional View of the
Final Landform

PROJECT NO.	DATE	APPROVED	FIGURE
1CD003.048	Nov.2004		4.5



 SRK Consulting <i>Engineers and Scientists</i>	Vangorda Pit Backfill Assessment		
	Section Through the Final Backfilled Vangorda Pit		
	PROJECT NO.	DATE	APPROVED
	1CD003.048	Nov.2004	
			FIGURE
			4.6

4.3 Lime/Limestone Amendment and Control

Experience at other sites has demonstrated the effectiveness of lime or limestone addition programs that follow a careful process of planning and implementation. SRK has developed two such programs by incorporating principles from ore grade control and mine planning. The following steps provide an example of how this process might be implemented for the Vangorda waste rock.

- As noted above, the waste rock would be mined from the dump in horizontal ‘slices’ about 5 m thick. Initially, each ‘slice’ to be relocated would be demarcated in ‘blocks’ of about 50 m x 50 m up to 100 m x 100 m. The block would be labelled and surveyed to provide a record of the source location of the material.
- Waste rock would be sampled on the surface of each block at a 10 m x 10 m grid interval. The paste parameters would be obtained for the less than 10 mm size fraction of each sample. The lime demand would be calculated from the correlation with paste pH and past conductivity. The lime demand for the block would be calculated from the average of all the samples taken in the block. An actual lime demand test would be completed on every fifth sample initially, to verify that the correlation applies, or to modify the correlation as necessary.
- The amount of lime or limestone required for the block as a whole would then be calculated for the 5 m thick slice. The calculated amount of lime or limestone would be spread over the surface of the block.
- The block would then be excavated, loaded, hauled and end-dumped in the pit where it would be spread to a thin layer. The excavation, dumping and spreading would provide the necessary mixing to contact the lime or limestone with the acidic rock.
- The area over which the entire block is spread would be surveyed. Initially, the spread material would be sampled over a regular grid to provide about four samples from each block. Paste parameters and leach extraction tests would be completed on these samples to verify that sufficient lime or limestone was added to the material.

4.4 Vangorda Creek Routing

There would be a significant proportion of acid generating waste rock above the long term water table, and therefore some ongoing oxidation would be expected. For this reason, the channel to carry Vangorda Creek across the fill should be designed to minimize leakage and hence transportation of oxidation products out the waste rock that would remain above the water table.

The channel can be designed using one of two strategies: an engineered chute or a “regime” channel. For the former, the channel would have a uniform slope across its entire reach. A regime channel, on the other hand, would mimic nature. For mildly sloped channels, there are guidelines for estimating appropriate widths, meander lengths, depths, morphology, and so on. There are, however, far fewer guidelines for steep channels, such as would be required for a chute across the fill which would require a uniform slope of about 5.5 %.

A “regime” channel for Vangorda would require a “step-pool” morphology. The bankfull width would be approximately 5 m. The pool lengths would be 15 m, and over a distance of 550 m, about 30 steps (which are effectively rock drop structures) would be required. Given a total elevation drop of 30 m, each step would have to comprise boulders at least 1 m in diameter, which would be keyed into the bed. Since the main channel would not accommodate the full design flow of $31 \text{ m}^3/\text{s}$, a floodplain would be required. It would, however, be difficult to minimize seepage losses from such a configuration, which renders it unattractive.

A third possibility that could be considered is to ‘create’ a waterfall or sequence of waterfalls upstream of the pit within the existing reach of Vangorda Creek. A large elevation drop can be achieved in one step, which would then significantly reduce the downstream slope of the channel. This option would however require that the waterfall be cut into bedrock.

The following conceptual designs were considered:

No waterfall option. The backfill profile suggests that the channel would drop from 1150 m to 1114.4 m over a distance of 705 m. Thus, the channel would have a slope of about 5%. To convey the design flow of $31 \text{ m}^3/\text{s}$, the channel would require riprap with a D_{50} of 480 mm (various other methods of calculation provide D_{50} estimates ranging from 411 mm to 519 mm). The base width of the channel would be 5 m, and it would have sideslopes of 2H:1V. The depth of water at design discharge would be about 1.03 m. Since flow would be supercritical, riprap protection should extend (vertically) to at least 0.5 m above water surface (and preferably 1 m above). The layer of riprap would be about 1 m thick.

Sequence of waterfalls that drop the channel to 1130 m near the upstream pit perimeter. In this case, the channel would have a much more reasonable slope of 3%. The riprap sizing would be reduced to a D_{50} of 320 mm (other methods of calculation provide D_{50} estimates ranging from 272 mm to 394 mm). The base width of the channel would remain at 5 m, with sideslopes of 2H:1V. The depth of water at design discharge would be about 1.14 m. Since the flow would be mildly supercritical, the riprap should be placed to at least 0.5 m above peak water level (i.e., vertical dimension). The riprap layer thickness would be about 0.8 m.

Both these options would allow for lining the channel to minimise seepage losses to the underlying waste rock. The cost estimate was based on a uniform channel cross-section and uniform slope. This was done to provide an estimate of volumes and sizes of rock that will be required to line the channel for erosion protection. At latter stages in the closure planning, an alternative design should also be examined in which the reconstructed channel is given a more natural morphology (step-pool).

4.5 Estimated Water Quality Impacts

Figure 4.7 provides the relationship between the dissolved zinc concentration and the pH for the column tests amended with lime and limestone. The minimum objective of the lime or limestone amendment program will be to increase the porewater pH to above 7. At this pH, the dissolved zinc

concentration in the saturated zone of the backfill is expected to be in the order of about 30 mg/L or less. This will represent a substantial improvement in the zinc concentration from the current Vangorda Pit water zinc concentration of 115 mg/L. Initially, the unsaturated material above the water table will be neutral, and zinc concentrations will also be in the 30 mg/L range. However, the pH in the unsaturated material may decrease and metal concentrations increase if there is significant oxygen penetration through the till cover.

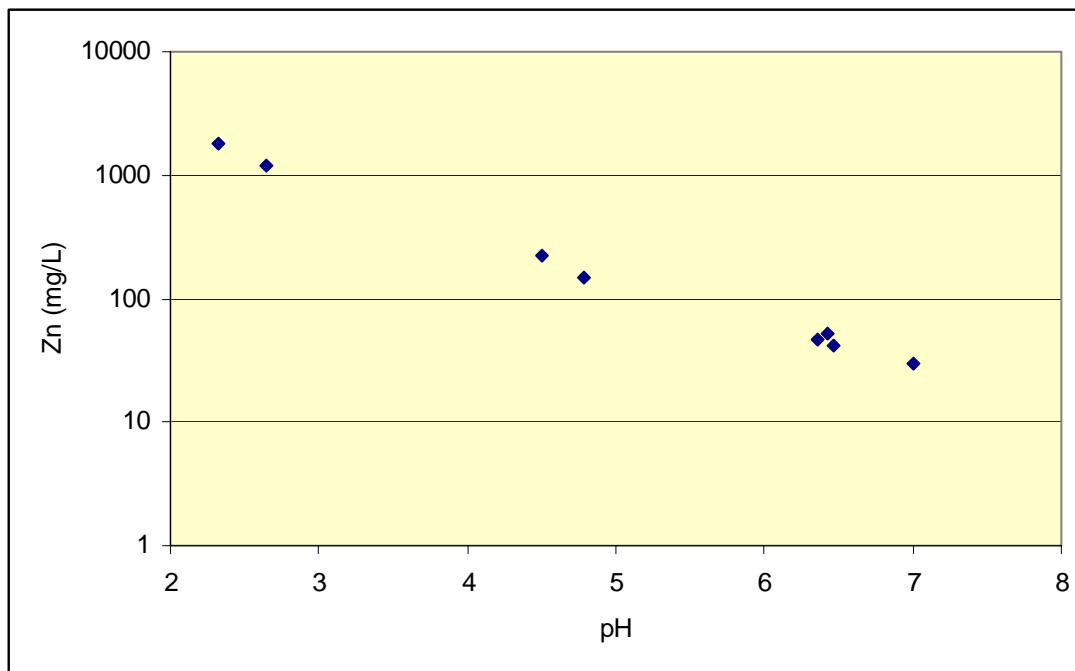


Figure 4.7: Zinc Concentration - pH Relationship Indicated by Saturated Columns

The water table in the backfill will rise until the long term steady state level is reached. Thereafter, any water that infiltrates through the till cover will exit the backfill as seepage. Preliminary cover modelling indicated infiltration through a compacted till cover of up to 20 mm per year. If the porewater zinc concentration remains at or below 30 mg/L, the incremental increase in zinc concentration in Vangorda Creek would be about 0.017 mg/L. (This estimate assumes 20.8 ha of surface area and an annual average flow of about 7.3 million m³ in Vangorda Creek.) However, should the porewater concentrations in the unsaturated zone increase above 50 mg/L, the incremental contribution to Vangorda Creek would increase above 0.03 mg/L. Since the seepage rate will be approximately steady, the incremental increase in zinc concentrations will be higher during periods where Vangorda Creek flows are low.

It is therefore likely that a long-term contingency will be required to manage the contaminant loading from the unsaturated zone of the backfill. Two options are available. The first option would be to estimate the rate of oxygen diffusion into the unsaturated zone, and then increase the lime or limestone addition to neutralize future acidity. The second would be to install one or more wells in

the backfill, creating a slight draw-down cone to capture any contaminated water prior to it escaping the pit. The extracted water would then be treated and discharged.

5 Cost Estimates

5.1 Construction Direct Costs

The estimated costs of the construction activities outlined in the preceding chapter are summarised in Table 5.1. Only direct costs are included. Supporting assumptions and additional information are provided in Appendix G.

Table 5.1: Estimated Relocation and Construction Costs

Item	Cost Estimates	Totals
1	Waste Rock: load, haul 3km and dump	\$36,292,000
2	Riprap: drill, blast and stockpile locally	\$98,000
3	Riprap: screen, load, haul and stockpile (10km)	\$295,000
4	Riprap: load from local stockpile, haul and place (.5km)	\$138,000
5	Filter layer: supply and stockpile locally	\$24,000
6	Filter layer: load , haul, place and compact (3km)	\$52,000
7	Till: Load from stockpile, haul, place and compact	\$4,440,000
8	Contaminated Soil : place and compact	\$0
9	Access Road: clearing and grubbing	NA
10	Access Road: construction	NA
Total Cost		\$41,339,000

Relocation of the Faro Oxide Fines and Low Grade Ore Stockpiles and placing it in the Vangorda Pit could add an additional \$5 to \$14 million to the above costs.

5.2 Lime Amendment Costs

The direct costs associated with lime amendment were estimated as shown in Table 5.2. The calculations are based on the average lime demand (see Table 3.2) and using the current cost of \$373 per tonne of quicklime delivered to site. No allowance was made for the amendment of the overburden, or for any soil that may need to be stripped from the footprint of the dump.

Table 5.2: Estimated Costs Associated with Alkali Amendment

Description	Volume (m ³)	Lime Demand (kg Ca(OH) ₂ /tonne)	Quicklime (tonnes)	Cost (\$)
Waste Rock	8,787,000	0.84	9,496	\$3,542,000
Oxide / Baritic Fines	286,000	8.4	3,094	\$1,154,000
TOTAL			12,590	\$4,696,000

Adding the Faro Oxide Fines and Low-grade Ore Stockpiles could increase the lime amendment costs by about \$3.4 to \$8.9 million.

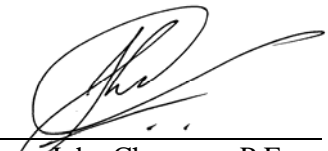
5.3 Total Direct Costs

The direct costs associated with the relocation of the Vangorda Dump and backfilling it to the pit are estimated to be as follows:

Construction and Relocation	\$41,000,000
<u>Alkali Amendment</u>	<u>\$ 5,000,000</u>
Total	\$46,000,000

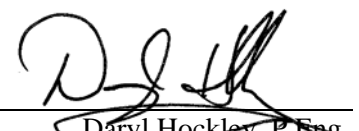
Including the Faro Oxide Fines and Lowgrade Ore Stockpiles in the backfilling program could add between \$8.4 and \$23 million to the projected costs. It is anticipated that that ongoing monitoring and possibly pumping and treating will be required to limit contaminant release from the backfill in the long term.

This report, “**Assessment of Vangorda Pit Backfilling – 2004/05 Task 14g**” has been prepared by SRK Consulting (Canada) Inc.



John Chapman, P.Eng.

Reviewed by



Daryl Hockley, P.Eng.
Principal

Appendix A

Test Pit Logs

Field Notes
June 21-24, 2004
Faro
Vangorda Backfill and Oxide Fines:
Lime Demand Testing

Dylan MacGregor
John Chapman

June 21, 2004

Calibrating pH probe Slope -58.3 mV/pH
Asym 3 mV

Reading Conductivity Std Standard 1413 $\mu S/cm$
Probe 1414 $\mu S/cm$

Reading ORP Std Std 212 mV
Probe 204 mV

Weather: Clear (no clouds), vis. h'd. by
smoke, hot ($\sim 30^\circ C$). Conditions
dry, dusty

Collecting water sample @ top of Green
Dump, where Green Creek exits dump
GID01 sample site

Samples collected

2x 500 mL Bottles
1/25 mL Diss. Metals

Field Params:

pH 6.39 (drifted from 5.75)
T $^{\circ}C$ 25
Cond 2.28 mS/cm
ORP 240

Flow: Vis Est 5 L/s (as 1st May
sample)

V2A

collecting sample @ V2A, 3m/s
v/s culvert opening that samples into
Moose Pond

pH 7.60
T $^{\circ}C$ 13.2
Cond 1746 $\mu S/cm$
ORP 215 mV

Flow Vis Est 5 L/s

Baritic Fines Composite Sample
Collection

BF01 Dark orange br, sl moist
50% > 1 cm, 1.2 kg

Photo 1

Dry on surface, w/ thick surf
coating of white salts

BF02 olive brown, moist, 30% > 1 cm,
local patches of oxidation (orange,
orange br), 1.2 kg

Photo 2

Surface sl moist, minor (<5%)
pale salt coating, high clay/silt content.
- on slope, well drained \rightarrow no depression

BF03

June 21, 2004

Collected from disturbed material fr.
2002 test pit. Material has
continuous white salt coating up to 1mm
thick, beneath coating is olive br,
minor small or clasts (sample size)
30% > 1cm, 2kg sample

Photo 3

Elevated area → no surface ponding.

but grading oxide fines @ SE
end of Vg pit. Fines have been pushed
in to form an impit dump

Commonly surface coated w/ white salt.
coated surfaces are moist

VPOFO1

NW crest of top lift of Vg oxide fines.
Surface is crusted w/ salt, wt to
tan w/ minor pale yellow to orange
Material is mottled 50% ol. br.
50% orange br. 5% moist, 30%
/cm

Photo 4

VPOFO2

June 21, 2004

Appendix A

15m back fr. crest, on small
embankment of fines. Minor
white surface salts (50% coverage).
Material light orange brown, low
moisture content (not completely dry)
50% > 1cm

Photo 4

VPOFO3

3m above base of pit @ SE end,
on slope of impit dump.
Surface is coated w/ thin film of
white salt. Material is mix of
olive br and dark orange brown
More dark orange br material @
surface. Drier @ surf. w/ A in
moisture @ depth.

Photo 5

Photos 7, 8

Salts on overhanging face of
oxide fines
- yellow field book for scale

NORPAC 1-800-480-3542 - 47 Level

NORPAC 1-800-480-3542 - 47 Level

June 22, 2004

Weather: Smoky, high overcast, muggy, hot,
calm

Delivered GDO1, V2A water samples to
Rhonda Haggan (Env.) for
immediate Zn analysis

GDO1 Total Zn 16.9 mg/L

V2A Total Zn 232 mg/L

- V2A is a water license
station. Craig McKinnon says
they have never seen V2A that
high.

Sampling oxide fines and ore
stockpiles at Faro for lime
amendment testing.

LGSP C - Low Grade Stockpile C

LGSP C 01

Pit excavated in embankment. Surface
has discontinuous coating of wht salts.
Abundant sulphides. ~5cm dry surface
layer, orange, underlain by dark orangish brn,
sl. moist material, 50% > 1cm. Sandy fines

Photo 1

NORPAC 1-800-480-3542 - 47 Level

LGSP C 02

June 22, 2004

Appendix A

Traffic Surface of Sulphide Material,
<5% stained. Surface is ~1cm olive
sulphide clasts + fines over ~15cm
yellowish to brownish orange material.
30% > 1cm, abundant weathered
clasts (sulphide + minor intrusives). Bottom
10cm is olive to dark gray sulphide matrix
w/ 70% angular sulphide cobbles.
Sample is a TP wall comparison
over entire pit. All material is moist.
Sampled material is 30% > 1cm

Photo 2

LGSP C 03

Bank of pile 10% ex, some 2'cm thin
white salts forming around ~50% of
clasts. ~10% of surf clasts are stained.
Dark orangish brn. Material has 1cm
olive layer @ surface, underlying material
is uniformly orangish brn, ~15% > 1cm,
moist.

Photo 3.

NORPAC 1-800-480-3542 - 47 Level

LGSPC04

June 22, 2004

Surface 50% fresh, 50% redish brn weath. sulphides
Material below is uniform orange brn.
v. sl. moist, 25% > 1cm. Clasts 100% Sx.
Occasional dk brn weath clasts

Photo 4

// LGSPC05

Surface has 30% coating of wht ppt.
Surface clasts den. brn weath, minor fresh
gray phyll. Material is uniform
orange brn, sl. moist at depth,
30% > 1cm.

Photo 5

// LGSPA01

Sample @ base of embankment. Surface is
olive weath. sulfide fines, sulphide
clasts. 50% of clasts weath brn to
purplish brn, olive surface layer 1cm
thick. Below is orange material, v.
moist to wet, 25% > 1cm. Sulphides
clasts can be intact, unweath, but
disintegrate into individ grains.

Photo 6

LGSPA02

June 22, 2004

Appendix A
Traffic surface of olive sulphide fines, 20%
of sulf. clasts stained brn. Material @
depth is moist, hard to dig (compact + rocks).
Sample 40% > 1cm top 10cm of
material weathered orange, grades
down into olive to gray at base of TP
Clasts subang.

Photo 7

// LGSPA03

Surface fines weath olive. Sulf clasts
25% stained brn. Up to 15% Sx. coated
in wht salt. Surf layer < 1cm. Fines
at depth rusty orange, clasts Sx,
stained dk redish brn. Fines sandy,
sl. moist. 70% > 1cm.

Photo 8

// LGSPA04

Top of dump. Surf. fines olive to
orangish brn, clasts 50% unweath Sx, 50%
brn stained Sx. Material @ depth is v. sl. moist.
Fines orange, clasts ang - subang Sx
stained dark redish brn. 60% > 1cm

Photo 9

LGSP A65

June 22, 2004

From side of embankment, surf faces ^{fine} yish
brn, clasts phyllite, Sr. Sr 30% stained dk brn,
20% fine. Material at depth is compact
partially cemented, moist, lt brn/orange,
clasts 50% stained dk brn, ang. 25% > 1cm

Photo 10.

CHSP 02

13:36

Surface covered w/ disc. coating of wht ppt
Sr cobbles, boulders abundant → down spt
some pygrn. Material is almost
25% > 1cm. Fine dark grey, contain
phyll/graph clasts sph Sr, typically fresh.
Clasts ang to subang

Photo 11

CHSP 01

14:00

Discont. coating of wht crust 40% of surface, surface
clasts incl. quartzose ore (sph) and small
sph. Material in TP is dark grey, moist
fines, fr. ang sph rich clasts. 60% > 1cm

Photo 12

NORPAC 1-800-480-3542 - 47 Level

CHSP 01

June 22, 2004

TP in face of CHSP facing cct. ^{Appendix A} Surface is
coated in discand. by c. of wht ppt - otherwise
fines mod grey. Clasts on surf are
2tz and phyll ore → high spt. Material
in TP is moist, fines dark grey, clasts
angular ore.

Photo 13

FOFO1

Fine oxide fines - green pile. Sample
pit at base of dump face. Abundant
greenish blue, yellow, white sands adj to floor.
Material olive green, w/ ox clasts
(rusty brn) @ depth. Wet. 5% > 1cm.

Photo 14.

FOFO2

Atop pile nr crest. Olive green fines,
< 5% > 1cm. Material as prev

Photo 15

NORPAC 1-800-480-3542 - 47 Level

FOF03

June 22, 2004

Material as prev. Moist, not wet

Photo 16

FOF04

First sample collected from brown/orange oxide fines. Material is reddish brn of surface w/ local coverings of wht salt on local highs. Material in TP is orange w/ abundant clasts which have weathered in situ to individ. min. grains. Moist to wet - will not go through sieve. Est. 5% > 1cm. Not sieved

Photo 17

FOF05

2nd sample on brown FOF.

Material wet, as prev. Not sieved

Photo 18

FOF06

3rd sample of br. FOF. Material as prev, sl^y drier (moist.) sieved.

Photo 19

MGSP01

June 22, 2004

Appendix A

Med. grade stock pile. Abundant grease, lesser phyllitic ore - high sph. little top. staining / ppts. Material in TP has olive brn. Sandy fines with orange mottling around clasts. Clasts are ang. rounded, w/ 70% > 1cm.

Photo 20

MGSP02

More pyritic clasts than @ MGSP01. Fines are sm. ol. brn w/ orange. dk. dis. brn mottling around isolated clasts. Sl. moist. Ang. 60% > 1cm.

Photo 21

MGSP03

17:10

White salts coating surface. Material in TP is mottled v. dk. gry/black and lesser ol. brn. Clasts are ang-subang, and have 70% > 1cm. Fines are sandy, sl. moist, graphitic/phyllitic.

Photo 22

VOF07

June 22, 2004

Oxide fines pile across inner haul road Fr. M GSP.
Surface is light orange, crustal. Material
in TP is orange, sl. moist to moist, 20% > 1cm.
Minor variations in shades of orange w/in
TP. Clasts are dom. Sx, tessar phyllite
chips.

Photo 23

VOF08

Material as prev. Weath. Sx clasts brk into
indiv. min. grains when cut by shovel
Moist to v. moist.

Photo 24.

VOF09

Material brnish orange, moist. Surface
of entire pile is moist - "exception"
are local highs on peaks of boulders.
Weath. Sx clasts as prev

Photo 25.

FOF10

18:37 June 22, 2004

Material as FOF09, moist to wet,
< 5% > 1cm.

Photo 26

FOF11

Material as prev. Moist to wet
< 5% > 1cm

Photo 29

NORPAC 1-800-480-3542 - 47 Level

NORPAC 1-800-480-3542 - 47 Level

June 23, 2004

Weather: Sunny, no clouds, smoke increasing
light breeze.

JTC + DBM measured pH, and on unamended
water / sample mixtures prepared last night,
then stirred up mixture once more. Samples
were left to settle.

Site visit to Grum to inspect floating
structure for pit lake bioremediation testing.
No structure present - only a shack to
be used as working space. No tables/benches
in shack, shack not level.

Return to Faro, inspect structure for pit lake
work. Incomplete - JTC to discuss w/ M.
Bryson

09:45 JTC to continue lab work, PBM to
finish sampling fines.

FoF12

Fines stockpile adj to road to Mt Mung
dumps. Material has orange fines, dry
at surf to moist @ depths. Clasts are py Sx,
minor schist/phyll, 25% brn stained on surface.
In TP, Sx clasts are weath s.t. ind. min.
grains break 15% > 1cm

Photo 1

Sample pit in side of safety berm on top
of pile

FoF13

June 23, 2004

Same pile as FoF12. Sample TP in traffic
surface on top of pile. TP contains
compact material in layers:
ol. brn (10cm) over dk gry (5cm) over
orange (base of pit). Clasts appear
to be py Sx, with abundant black schist
chips. Fines are moist to surface.
< 5% > 1cm.

Photo 2

FoF14

10:37

Side slope adj to Mt Mung Rd, same pile as
FoF13. Material is moist, orange brn,
15% > 1cm, clasts are py Sx, with sph, some
chips of schist. Surface clasts: 25% w/
brn ex stain

Photo 3

FoF15

Same pile, top @ Sand. Material is
orange brn at 0-15cm, then orange below.
Brn is prob surface wash down slope. Clasts
are py sph Sx = dark schist. 10% > 1cm.
In TP, most Sx clasts weath s.t. ind min
grains are free.

Photo 4

FLG 01

June 23, 2004

TP is what appears to be a low grade stockpile, small volume, NE of ox fire pile sampled on FOF 11 for FOF 16.

Surface is py sand, Sx pebbles thru boulders. In TP, material is mottled orange, dk orange, orangish yellow, dk brn, w/ 20% Sx clasts that appear fresh but are only accumulations of ind. grains and are cut by shovel. Material is moist, 25% to 71 cm.

Photo 5.

FOF 16

Same pile as FOF 11 - FOF 15. At base of dump face on SE side of pile. Material is mottled orange and orangish brn fines and Sx/Schist clasts Abundant sph. Moist, 10% to 71 cm. Runnels forming on dump face

Photo 6 (new CF card)

Contact Testing on Recent TP Samples June 27, 2004

Sample ID	pH	Cond (µS/cm)	Appendix A
LGSPC02	2.05	4880	
FOF08	2.04	6300	
MGSP03	3.10	2680	
FOF04	1.96	8340	
LGSPC03	2.35	3080	
CHSP02	5.30	2340	
FOF02	2.24	8550	
FOF09	2.07	5530	
FOF05	2.04	8180	
BF02	2.38	6210	
LGSPA03	2.36	5560	
FLG01	2.15	5140	
FOF12	2.21	4680	
FOF13	2.23	5360	
FOF14	2.13	6046	
FOF15	2.02	6060	
FOF16	1.96	5130	
VPOF03	2.33	5370	
BF01	3.47	5250	
VPOF01	2.60	5800	

June 23, 2004

SAMPLE ID	pH (s.v.)	cond (µS/cm)
LGSPA01	2.15	5010
LGSPA02	2.25	4860
LGSPA04	2.37	2250
LGSPA05	2.23	3980
LGSPC01	2.94	2120
LGSPC04	2.60	2460
LGSPC05	2.42	4740
BF03	3.17	4690
FOF03	2.31	7040
FOF07	2.34	4090
FOF01	2.03	9950
FOF06	2.16	8510
FOF11	2.08	4680
FOF10	2.14	4740
MGSP071	2.77	1940
MGSP02	4.77	2250 → pH: 5.25 cond: 2120
CHSP01	5.55	1612
CHSP03	5.80	2560

June 24, 2004

Line Demand Testing

Sample ID	Flask	pH (s.v.)	cond µS/cm	ALK Add #1	ALK Add #2	final pH
VTP28A	1	6.12	4	9.73	1	9.66
		0.651				
VTP28B	2	6.04	8	9.67	3	9.59
		1.677				
VTP33	3	7.39	3	9.59	2	9.61
		0.228				
VTP25B	4	6.69	3	9.76	2	10.22
* 2 samples w/ this ID		1.722				
VTP27BB	5	6.82	3	9.76	2	9.83
		1.770				
VTP44	6	6.94	2	9.79	1	9.51
		0.895				
VTP30A	7	7.19	5	9.61	1	9.54
		0.582				
VTP36	8	7.22	5.5	9.62	1.5	9.55
		0.741				
VTP41	9	4.80	30	9.87	3.5	9.59
		1.775				
VTP17B	10	6.33	4	9.68	1.5	9.63
		0.814				
VTP30C	11	6.70	5	9.69	2	9.75
		0.685				
	12	7.65	3	9.60	1	9.51
		0.250				
Calibrating pH probe		Slope	-58.6 mV/pH			
		asym	2 mV			

Reading cond std

Stand 1413 µS/cm
Probe 1407

Appendix A

Sample ID	Flask #	Cond (mS/cm)	mL 10g/L	mL 100g/L	Final pH	mL 10g/L	mL 100g/L	Final pH
VTP24A	1	6.32	8	9.62	2.5			9.53
		19.35						
VTP22	2	4.80	26	9.64	4.5			9.53
		1.864						
VTP31	3	7.65	3	9.61	1			9.49
		0.333						
LGSPC05	4	2.62	8	9.82	17			9.50
		3.45						
BF03	5	3.46	2	9.57	5			9.50
		3.86						
VPOFO1	6	2.75	3	9.61	16			9.53
		5.66						
FOFO7	7	2.41	27	9.69	20			9.51
		6.65						
FOFO3	8	2.48	5	9.65	67			9.55
		5.50						
BF01	9	3.68	14	9.55	30			9.52
		5.99						
VPOFO3	10	2.58	16	9.75	40			9.52
		4.91						
FOF15	11	2.22	5	9.59	20			9.57
		4.94						
FOF16	12	2.04	17	9.50	16			9.53
		5.11						
FOF14	13	2.10	16	9.89	10			9.57
		4.99						
FOF12	14	2.28	11	9.51	40			9.54
		4.06						
FOF13	15	2.31	4	9.55	15			9.57
		4.23						

Sample ID	Flask	Cond (mS/cm)	mL 10g/L	mL 100g/L	Final pH	mL 10g/L	mL 100g/L	Final pH
FOF10	1	2.41	15	9.85				
		5.77						
FOF11	2	2.28	7	18	9.67			
		6.18						
FOFO6	3	2.26	3	35	9.54			
		7.66						
FOFO1	4	2.19	26	9.76				
		7.07						
CHSP03	5	5.95	6.5	9.62				
		2.63						
CHSP01	6	5.90	12.5	9.51				
		2.020						
MGSP02	7	5.81	21.5	9.56				
		2.18						
MGSP01	8	3.07	6	9.89				
		2.17						
LGSPA01	9	2.30	3	19	9.54			
		5.75						
LGSPA02	10	2.30	3	17	9.58			
		4.016						
LGSPA04	11	2.57	21	5.5	9.53			
		2.620						
LGSPA05	12	2.42	17	16	9.54			
		4.69						
FLG01	13	2.28	8	12.5	9.53			
		4.620						
LGSPC01	14	3.25	39	9.54				
		1.944						
LGSPC04	15	2.89	79	9.53				
		1.781						

Appendix A

Appendix B

Laboratory Test Procedures

Memorandum

To:	Sohan Basra, CEMI	Date:	September 17, 2004
cc:		From:	John Chapman
Subject:	Vangorda Backfill Program Phase II Testing – Columns	Project #:	1CD003.48

1 Terms of Reference

This memorandum describes the testing requirements for the second phase of testing for the Vangorda Backfill Evaluation Testing Program. Two series of saturated column tests are to be completed as follows:

- i) Waste rock composites amended with lime; and,
- ii) Waste rock composites amended with limestone.

Both sets of tests are to be completed under anoxic conditions. Sample preparation and testing procedures are described below.

2 Sample Preparation

Prepare composite waste rock samples from the as received (do not dry the samples) waste rock samples by blending the samples as shown in Table 1. Blend the samples well until homogenous conditions are achieved.

Submit sub-samples of each of the composites for a 3:1 distilled water to solids extraction test, using a 200 g waste rock sample. Analyze the solution for acidity, conductivity, sulphate, pH and metals by ICP.

Split the remainder of the waste rock composite samples into two equal portions of about 1400 g each. Proceed as follows:

- Label the first portion –A (e.g. VG1-A) and obtain the exact weight in kg. Multiply the weight in kg with the LIME amendment rate given in Table 2 for the given composite (i.e. for VG1 with a sample weight of 1.4 kg, the lime amendment would be $1.4 \times 0.65 = 0.91$ g of Lime) to obtain the weight of lime required for the composite sample. Add this amount of lime to the sample by spreading out the composite sample in a thin layer, and then sprinkling the lime evenly over the sample. Blend well and split out a 200 g sample for a 3:1 leach extraction test as above. The balance of the sample (about 1200 g) will be used for the column test as described below in the next section.
- Label the second portion –B (e.g. VG1-B) and obtain the exact weight in kg. Multiply the weight in kg with the LIMESTONE (CaCO_3) amendment rate given in Table 2 for the given

composite (i.e. for VG1 with a sample weight of 1.4 kg, the limestone amendment would be $1.4 \times 0.72 = 2.17$ g of limestone) to obtain the weight of limestone required for the composite sample. Add this amount of limestone to the sample by spreading out the composite sample in a thin layer, and then sprinkling the limestone evenly over the sample. Blend well and split out a 200 g sample for a 3:1 leach extraction test as above. The balance of the sample (about 1200 g) will be used for the column test as described below in the next section.

Table 1. Preparation of Composite Samples

Composite VG1		Composite VG2		Composite FOF1		Composite LGS1	
Sample	Wt (g)	Sample	Wt (g)	Sample	Wt (g)	Sample	Wt (g)
VTP24A	280	VTP26A	750	FOF01	500	LGSPA01	600
VTP27A	280	VTP37A	750	FOF02	500	LGSPA02	600
VTP28B	280	VTP21	750	FOF03	500	LGSPA03	600
VTP29B	280	VTP42	750	FOF04	500	LGSPA04	600
VTP17A	280			FOF05	500	LGSPA05	600
VTP23B	280			FOF06	500		
VTP30B	280						
VTP25A	280						
VTP43	280						
VTP35A	280						
VTP23A	280						
Total Wt. (g)	3080		3000		3000		3000

Table 2 Alkali Amendment Rates

Amendment	Units	Composite VG1	Composite VG2	Composite FOF1	Composite LGS1
Lime*	gCaO/kg	0.65	2.53	17.99	7.34
Limestone**	gCaCO ₃ /kg	1.55	6.01	42.84	17.48

3 Apparatus

Prepare eight columns 300 mm long with an internal diameter of 50 mm (2 inches), with removable head-plates each equipped with an inlet. A fine mesh distribution ‘plate’ (3 to 4 nylon mesh disks) is required at each end of the column as shown schematically in Figure 1. The internal volume of each column will be about 620 mL and should accommodate about 1,000 g of waste rock (at < 20 mm). The porevolume of the contained waste rock is estimated to be about 300 mL.

4 Procedure

The operational set-up of the column test is shown in Figure 2. The tests will be carried out using distilled de-aerated water. The proposed procedure for the preparation of the column is as follows:

1. Seal the base plate of the column in place.
2. Place the nylon disks at the bottom of the column, and weigh the apparatus.
3. Fill with rock to the top of the column, ensuring that an even compact fill density is achieved. Weigh the apparatus and subtract the column weight to obtain the rock load.
4. Seal the top of the column in place, and pressure test.
5. Attach the inlet and outlet pipes of the column and displace the pore gases with nitrogen, and seal valves A, B and C.
6. Open Valves A, B, and C and flood the tube to Valve C. Shut Valve C and slowly flood the column with de-aerated distilled water to prime the column. Measure the volume of water used to saturate the column. (This can be done either by weight or by starting with a known volume of water and measuring the volume taken up in the column).

7. Seal off Valve A and Valve B and open Valve C. Set the pump so that about 1 to 2 pore volume equivalents are recycled through the column over a 24 hour period. Continue recycling for a 14 day period while ensuring that no air enters the system.
8. At day 14, stop the pump, shut of Valve C. Open Valves A and then B and slowly displace the porewater equal to one pore volume (about 300 mL) with fresh de-aerated distilled water. Sample should be extracted under anoxic conditions. Immediately obtain the pH conductivity and redox of the solution. Submit solution for alkalinity, acidity, and metals by ICP.
9. Continue the column in recycle mode for a further 7 days, and repeat step 8. Once sample has been obtained and submitted for the required analysis, seal off re-saturated column until further notice.

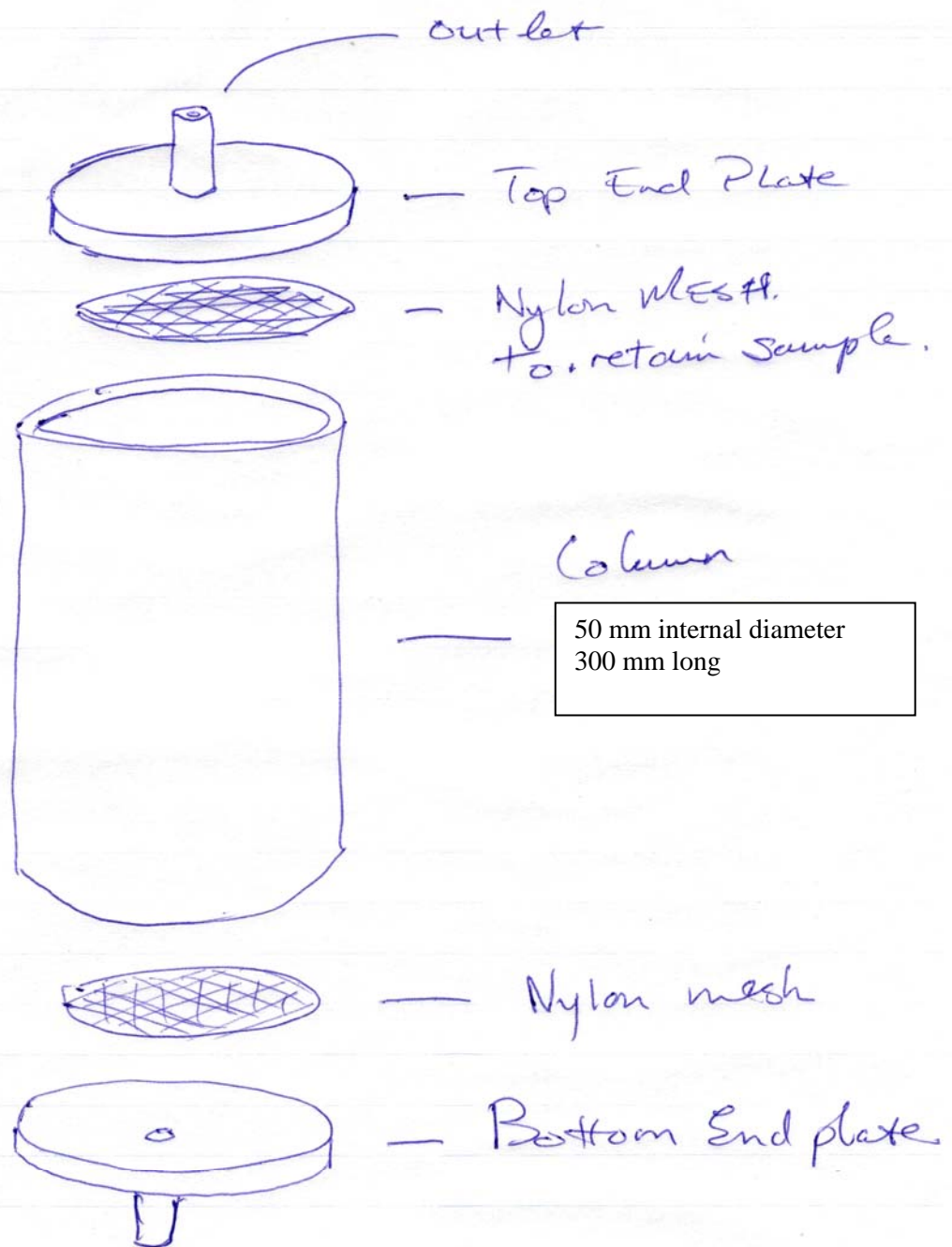


Figure 1 Column Schematic.

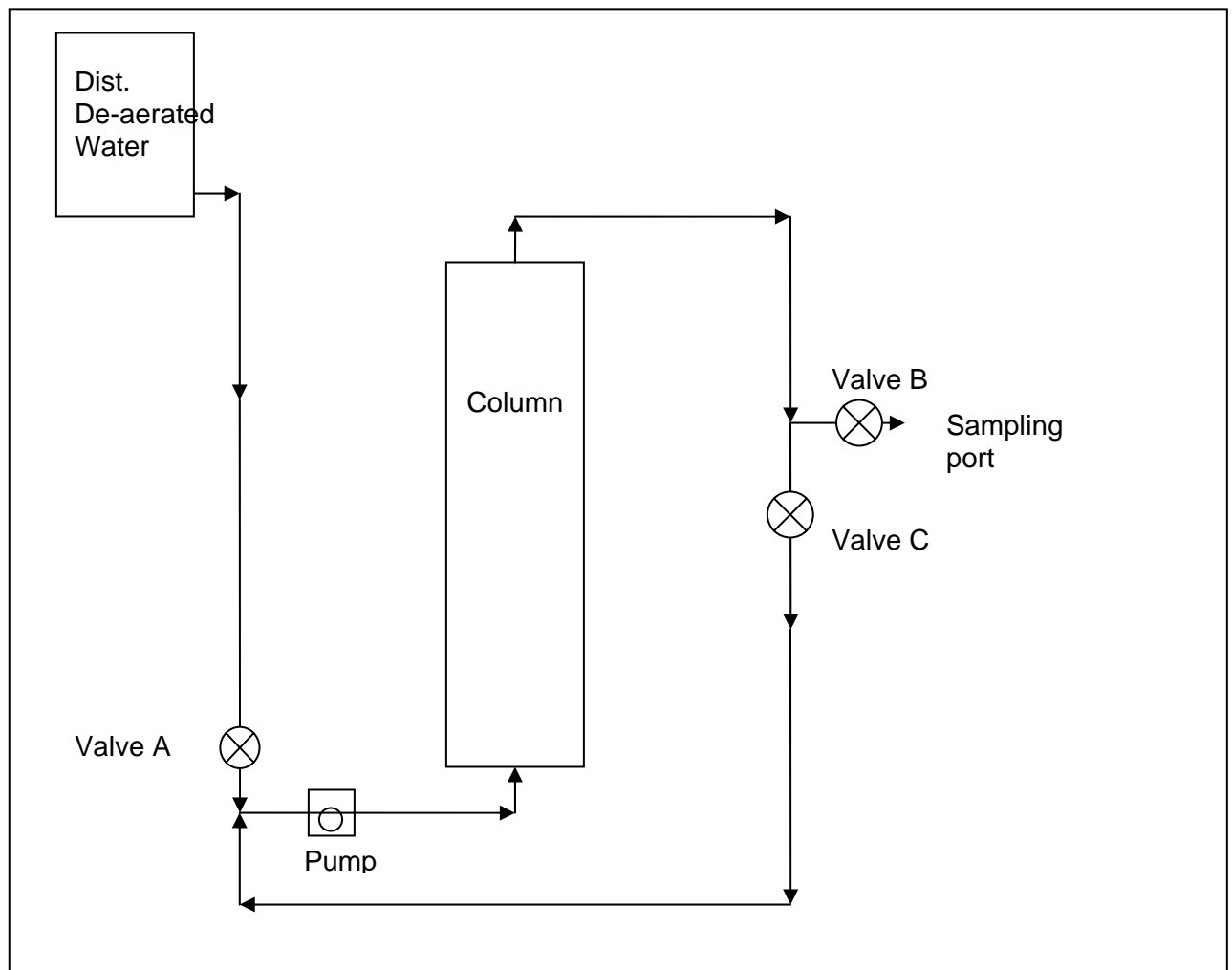


Figure 2. Operational Set-up

Appendix C

Acid Base Account and Metal Analyses

Table C-1 Acid Base Account and Net Acid Generation Test Results

Sample	Paste pH	S(T) %	S(SO ₄) %	AP	NP	Net NP	NP/AP	TIC %	CO ₃ NP	NAG pH	NAG pH 4.5	NAG pH 7.0
Oxide Fines												
VPOF (VANGORDA) 01	-	19.40	1.52	558.8	-	-	-	-	-	-	-	-
VPOF (VANGORDA) 02	2.8	14.00	1.74	383.1	-11.5	-394.6	<0.1	<0.01	0.8	1.99	87.7	106.8
VPOF (VANGORDA) 03	-	14.00	1.76	382.5	-	-	-	-	-	-	-	-
Baritic Fines												
BF (VANGORDA) 01	-	15.40	0.88	453.8	-	-	-	-	-	-	-	-
BF (VANGORDA) 02	2.8	21.30	1.89	606.6	-11.1	-617.6	<0.1	<0.01	0.8	1.90	125.0	136.7
BF (VANGORDA) 03	-	22.30	1.19	659.7	-	-	-	-	-	-	-	-
Till/Overburden												
VPT 31	8.5	0.07	0.02	1.6	65.9	64.3	42.2	0.75	62.5	6.68	0.0	1.2
VPT 32	-	0.01	<0.01	0.3	-	-	-	0.23	19.2	-	-	-
VPT 33	8.6	0.02	0.01	0.3	59.0	58.7	188.8	0.63	52.5	6.44	0.0	2.2
VPT 34	-	0.04	0.01	0.9	-	-	-	1.49	124.2	-	-	-
VPT 35A	7.5	0.70	0.24	14.4	27.5	13.1	1.9	1.72	143.3	5.37	0.0	2.2
VPT 35B	-	0.11	0.04	2.2	-	-	-	0.67	55.8	-	-	-
VPT 17A	-	0.58	0.41	5.3	-	-	-	-	-	-	-	-
VPT 17B	7.9	0.79	0.14	20.3	89.9	69.6	4.4	1.03	85.8	6.70	0.0	0.3
VPT 36	8.1	0.11	0.04	2.2	56.9	54.7	26.0	0.66	55.0	6.62	0.0	1.2
Wasterock												
VPT 18A	-	2.25	0.16	65.3	-	-	-	-	-	-	-	-
VPT 18B	7.1	4.27	0.15	128.8	29.8	-98.9	0.2	0.83	69.2	3.04	7.0	39.7
VPT 19	-	18.50	0.20	571.9	-	-	-	-	-	-	-	-
VPT 20	5.5	22.90	0.41	702.8	19.3	-683.6	<0.1	0.48	40.0	2.37	35.6	85.3
VPT 21	-	7.55	0.74	212.8	-	-	-	0.20	16.7	-	-	-
VPT 22	5.3	4.81	0.37	138.8	9.7	-129.1	0.1	0.27	22.5	2.38	22.9	53.7
VPT 23A	-	8.80	0.49	259.7	-	-	-	0.54	45.0	-	-	-
VPT 23B	6.2	21.90	0.37	672.8	45.5	-627.3	0.1	1.04	86.7	2.94	11.1	66.2
VPT 24A	-	10.90	0.39	328.4	-	-	-	1.27	105.8	-	-	-
VPT 24B	5.8	25.10	0.40	771.9	45.9	-726.0	0.1	1.20	100.0	2.73	17.0	82.7
VPT 25A	-	9.37	0.46	278.4	-	-	-	0.02	1.7	-	-	-
VPT 25B	5.7	4.54	0.39	129.7	1.8	-127.9	<0.1	0.07	5.8	2.54	21.2	38.9
VPT 25B-2*	7.7	0.94	0.10	26.3	15.6	-10.7	0.6	0.35	29.2	4.45	0.1	2.9
VPT 26A	-	2.05	0.37	52.5	-	-	-	0.05	4.2	-	-	-
VPT 27A	6.3	3.67	0.11	111.3	20.5	-90.8	0.2	0.54	45.0	3.52	3.6	40.7
VPT 27B	-	1.89	0.13	55.0	-	-	-	1.23	102.5	-	-	-
VPT 27C	6.7	2.00	0.15	57.8	19.8	-38.0	0.3	1.00	83.3	3.12	6.4	26.1
VPT 28A	-	2.75	0.08	83.4	-	-	-	0.82	68.3	-	-	-
VPT 28B	6.3	1.48	0.24	38.8	14.8	-23.9	0.4	0.66	55.0	4.04	0.7	10.6
VPT 29A	-	5.40	0.15	164.1	-	-	-	0.91	75.8	-	-	-
VPT 29B	6.6	6.26	0.21	189.1	29.1	-160.0	0.2	0.96	80.0	3.12	8.7	41.6
VPT 30A	-	0.18	0.04	4.4	-	-	-	0.78	65.0	-	-	-
VPT 30B	6.1	2.48	0.23	70.3	4.4	-65.9	0.1	0.07	5.8	2.47	27.0	37.7
VPT 30C	-	1.26	0.07	37.2	-	-	-	0.50	41.7	-	-	-
VPT 37A	3.6	1.98	0.52	45.6	-4.1	-49.7	<0.1	0.17	14.2	2.60	25.2	32.4
VPT 37B	-	2.57	0.18	74.7	-	-	-	1.12	93.3	-	-	-
VPT 38	-	1.15	0.06	34.1	-	-	-	0.20	16.7	-	-	-
VPT 39	6.8	2.67	0.13	79.4	19.9	-59.5	0.3	0.52	43.3	3.03	9.3	30.3
VPT 40	-	1.04	0.10	29.4	-	-	-	0.03	2.5	-	-	-
VPT 41	4.8	1.04	0.29	23.4	2.3	-21.1	0.1	0.03	2.5	2.94	8.0	12.1
VPT 42	-	11.60	1.01	330.9	-	-	-	0.01	0.8	-	-	-
VPT 43	4.1	2.16	0.23	60.3	1.8	-58.5	<0.1	0.06	5.0	2.56	22.1	32.9
VPT 44	-	1.06	0.06	31.3	-	-	-	0.52	43.3	-	-	-

AP = Acid potential in tonnes CaCO₃ equivalent per 1000 tonnes of material. AP is determined from calculated sulphide sulphur content: S(T) - S(SO₄), assuming total conversion of sulphide to sulphate.

AP = When calculated AP is 0, it is reported as <0.01.

NP/AP= When AP is 0, then 0.01 value is used for AP to calculate NP/AP.

NP = Neutralization potential in tonnes CaCO₃ equivalent per 1000 tonnes of material.

NET NP = Net neutralization potential = Tonnes CaCO₃ equivalent per 1000 tonnes of material.

NOTE: Where S(T) is reported as <0.01%, a S(T) value of 0.01% is used for the AP calculation.

Where S(SO₄) is reported as <0.01%, it is assumed to be zero for the AP calculation (ie. if S(SO₄) is less than 0.01% or is not analyzed, AP is calculated from S(T) only).

TIC = Total Inorganic Carbon as %C.

Carbonate NP calculated from total inorganic carbon (TIC) assay. TIC value of 0.01 is used in calculation if TIC <0.01%.

NAG = Net Acid Generation (kg H₂SO₄/tonne)

Table C-2 Metals Analysis by ICP

Parameter	Units	Oxide Fines			Baritic Fines			Till/Overburden								
		VPOF (VG) 01	VPOF (VG) 02	VPOF (VG) 03	BF (VG) 01	BF (VG) 02	BF (VG) 03	VPT 17A	VPT 17B	VPT 31	VPT 32	VPT 33	VPT 34	VPT 35A	VPT 35B	VPT 36
Ag	ppm	60.6	45.6	51.8	53	66.9	59.6	2.3	1.7	<0.2	<0.2	<0.2	<0.2	7.1	<0.2	<0.2
Al	%	0.52	0.68	0.72	1.03	0.46	0.54	0.84	1.61	2.02	1.09	1.97	1.93	1.84	1.63	1.57
As	ppm	1334	1014	958	807	1789	1278	1509	165	20	8	10	11	237	28	186
Ba	ppm	18	18	18	21	17	19	369	210	420	129	389	506	305	301	397
Be	ppm	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	0.6	0.6	0.7	0.6	0.7	0.5	0.5
Bi	ppm	23	20	18	15	26	20	<5	<5	<5	<5	<5	<5	<5	<5	<5
Ca	%	0.24	0.3	0.3	0.56	0.19	0.12	0.1	3.2	2.74	1.05	2.53	2.93	1.32	2.29	2.21
Cd	ppm	<1	<1	<1	21	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
Co	ppm	73	60	60	43	109	89	4	14	15	8	16	12	18	13	14
Cr	ppm	39	54	49	54	67	60	98	97	239	211	265	187	187	178	177
Cu	ppm	3855	1999	3370	1455	6409	5582	154	113	29	15	31	32	158	40	40
Fe	%	>15.00	>15.00	14.99	>15.00	>15.00	>15.00	5.88	4.52	3.96	2.29	3.96	3.78	5.22	3.4	3.56
K	%	0.06	0.08	0.08	0.07	0.08	0.05	0.23	0.1	0.17	0.19	0.16	0.17	0.19	0.14	0.13
Mg	%	0.18	0.26	0.3	0.5	0.18	0.19	0.31	1.61	1.89	0.85	1.91	1.7	1.25	1.53	1.54
Mn	ppm	700	545	653	2314	186	723	430	454	610	312	591	578	527	502	563
Mo	ppm	4	4	4	4	2	5	<2	<2	<2	<2	<2	2	4	<2	<2
Na	%	0.01	0.01	0.01	0.02	0.01	0.01	0.02	0.01	0.03	0.02	0.02	0.02	0.02	0.02	0.02
Ni	ppm	26	28	24	41	18	24	8	44	110	58	120	76	79	95	90
P	ppm	570	607	564	604	482	502	963	1003	704	507	740	760	733	646	680
Pb	ppm	>10000	>10000	>10000	>10000	>10000	>10000	2489	1889	88	32	59	73	6305	140	110
Sb	ppm	84	73	79	70	88	88	13	7	11	<5	9	8	13	6	<5
Sc	ppm	2	2	2	3	1	2	2	2	6	3	6	5	5	4	5
Sn	ppm	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Sr	ppm	<1	<1	<1	<1	<1	<1	93	104	96	35	88	101	49	84	83
Ti	%	<0.01	<0.01	<0.01	0.01	<0.01	<0.01	<0.01	<0.01	0.04	0.03	0.04	0.03	0.04	0.03	0.03
V	ppm	35	40	37	39	28	33	27	26	58	27	58	52	48	43	47
W	ppm	302	163	192	560	125	439	<10	<10	<10	<10	<10	<10	26	<10	<10
Y	ppm	2	2	2	6	<1	2	1	8	11	6	10	11	8	9	9
Zn	ppm	>10000	>10000	>10000	>10000	9451	>10000	835	513	187	178	131	137	2321	200	243
Zr	ppm	14	14	13	13	14	14	8	12	10	9	11	11	13	8	9

Table C-2 Metals

			Waste Rock Dump															
Parameter	Units	VPT 18A	VPT 18B	VPT 19	VPT 20	VPT 21	VPT 22	VPT 23A	VPT 23B	VPT 24A	VPT 24B	VPT 25A	VPT 25B	VPT 26A	VPT 27A	VPT 27B	VPT 27C	VPT 28A
Ag	ppm	2.5	6.9	20.4	25.5	17.2	19.2	32.7	19.4	37.8	25.7	60.6	18.7	3.6	16.4	0.9	2.3	5
Al	%	1.49	1.19	0.44	0.52	0.9	1.03	0.95	0.51	0.88	0.21	0.23	0.86	1.41	1.38	0.55	0.59	1.22
As	ppm	151	292	1263	2552	805	781	1241	843	995	1052	3801	1427	2042	342	89	351	588
Ba	ppm	74	37	16	17	32	35	37	87	83	121	17	33	97	48	110	95	43
Be	ppm	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	1	<0.5	<0.5
Bi	ppm	<5	<5	23	28	<5	<5	7	19	8	29	<5	<5	<5	<5	<5	<5	<5
Ca	%	1.27	0.92	0.73	0.84	0.36	0.46	0.92	1.32	1.3	1.48	0.04	0.24	0.43	0.68	2.28	0.56	0.73
Cd	ppm	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	14	<1	<1	<1
Co	ppm	47	59	211	299	51	39	44	270	70	357	21	42	24	32	20	29	53
Cr	ppm	224	196	115	150	91	112	88	114	120	114	179	150	113	119	155	89	103
Cu	ppm	279	593	2439	3641	1121	540	835	3481	1230	3767	>10000	4217	131	400	96	108	329
Fe	%	6.58	7.86	>15.00	>15.00	10.69	7.11	11.33	>15.00	12.7	>15.00	6.95	5.9	5.73	7.99	5.01	6.91	7.54
K	%	0.11	0.13	0.08	0.07	0.11	0.1	0.13	0.06	0.13	0.03	0.07	0.14	0.44	0.24	0.15	0.12	0.15
Mg	%	1.77	1.18	0.42	0.34	0.52	0.67	0.73	0.58	0.74	0.51	0.06	0.42	0.63	0.78	0.92	0.7	0.89
Mn	ppm	1140	1606	3536	2920	1202	1086	2211	9282	2616	8082	171	294	709	3353	729	2664	2217
Mo	ppm	6	6	<2	<2	6	12	4	<2	3	<2	15	8	3	3	10	<2	2
Na	%	0.01	0.02	0.01	0.01	0.01	<0.01	0.02	<0.01	0.02	<0.01	0.01	0.02	0.01	0.02	0.02	0.01	0.01
Ni	ppm	127	94	27	25	38	67	47	31	49	24	32	52	44	49	63	56	63
P	ppm	618	546	340	409	579	461	512	402	536	358	534	564	529	545	972	570	579
Pb	ppm	1859	5503	>10000	9884	>10000	>10000	>10000	>10000	>10000	>10000	>10000	>10000	2580	>10000	734	1676	3370
Sb	ppm	13	19	51	239	41	32	50	26	58	37	328	81	8	28	7	9	11
Sc	ppm	5	3	<1	<1	3	2	3	<1	3	<1	<1	2	2	3	3	3	3
Sn	ppm	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Sr	ppm	49	13	<1	<1	<1	9	<1	<1	<1	<1	<1	10	22	18	70	26	32
Ti	%	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.01	<0.01	0.01	<0.01	<0.01	<0.01	0.03	0.02	<0.01	<0.01	0.01
V	ppm	46	35	21	33	35	32	30	29	32	26	10	23	21	24	42	21	26
W	ppm	75	144	229	141	150	252	323	231	333	255	874	203	57	208	22	57	107
Y	ppm	6	5	3	4	3	4	5	4	5	3	1	4	9	5	12	8	6
Zn	ppm	5762	>10000	>10000	>10000	>10000	>10000	>10000	>10000	>10000	>10000	>10000	>10000	4367	>10000	1910	4516	8080
Zr	ppm	10	10	12	13	14	10	13	14	12	14	7	12	12	14	12	11	13

Table C-2 Metals

		Waste Rock Dump															
Parameter	Units	VPT 28B	VPT 29A	VPT 29B	VPT 30A	VPT 30B	VPT 30C	VPT 37A	VPT 37B	VPT 38	VPT 39	VPT 40	VPT 41	VPT 42	VPT 43	VPT 44	VPT 25B-2
Ag	ppm	1.6	11.1	10.6	<0.2	29.2	7.8	1.1	2	<0.2	3	0.7	3.5	43.4	5.3	1	0.7
Al	%	1.38	1.1	0.9	2.03	1.24	1.72	0.25	0.35	2.19	1.26	1.82	1.58	0.23	1.14	0.98	1.2
As	ppm	229	527	693	37	1049	324	166	191	55	223	111	240	1050	418	178	158
Ba	ppm	141	47	34	511	69	155	77	80	84	82	178	182	21	105	164	167
Be	ppm	<0.5	<0.5	<0.5	0.6	0.6	0.6	<0.5	<0.5	0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Bi	ppm	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	<5	11	<5	<5	<5
Ca	%	0.68	0.67	0.86	3.02	0.33	1.74	0.18	0.34	0.59	0.5	0.12	0.39	<0.01	0.17	0.58	0.53
Cd	ppm	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
Co	ppm	37	53	53	16	19	18	59	90	37	40	15	10	81	26	20	24
Cr	ppm	106	120	76	240	119	149	98	99	120	83	60	110	299	95	111	86
Cu	ppm	185	524	661	41	254	100	446	447	140	352	55	258	440	153	67	51
Fe	%	7.26	9.59	10.17	4.04	6.9	4.83	6.42	7.48	6.52	7.41	5.74	4.84	11.04	6.63	4.72	4.71
K	%	0.18	0.14	0.09	0.17	0.26	0.2	0.09	0.09	0.3	0.11	0.08	0.16	0.15	0.15	0.23	0.3
Mg	%	1.08	0.86	0.79	2.03	0.53	1.28	0.17	0.47	1.18	1.12	0.76	0.72	0.01	0.54	0.76	0.7
Mn	ppm	1284	2279	2358	628	469	651	1090	3019	1003	1473	920	321	29	1096	877	838
Mo	ppm	2	3	4	<2	4	2	<2	<2	<2	<2	<2	6	9	<2	<2	<2
Na	%	0.01	0.01	0.01	0.02	0.02	0.02	<0.01	<0.01	0.01	<0.01	<0.01	0.02	0.03	0.01	<0.01	<0.01
Ni	ppm	79	62	53	115	38	72	30	44	77	49	32	30	10	34	67	55
P	ppm	1030	578	585	845	610	677	302	476	639	456	561	686	242	516	428	548
Pb	ppm	1063	8449	7189	248	>10000	4951	211	364	119	1505	479	3350	>10000	4504	772	609
Sb	ppm	16	21	20	10	47	16	<5	7	<5	6	6	16	92	15	<5	<5
Sc	ppm	4	4	3	5	3	4	2	3	4	3	2	2	<1	3	2	2
Sn	ppm	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Sr	ppm	27	8	12	116	13	55	2	4	6	18	6	21	<1	11	17	11
Ti	%	0.01	<0.01	<0.01	0.04	0.03	0.04	<0.01	<0.01	0.03	<0.01	0.01	0.02	0.01	0.01	0.01	0.02
V	ppm	32	32	32	58	25	44	12	17	30	19	24	28	15	22	17	17
W	ppm	58	182	160	<10	86	54	12	56	<10	51	19	21	<10	60	20	13
Y	ppm	7	5	6	10	7	9	4	6	10	5	3	3	<1	3	5	5
Zn	ppm	4516	>10000	>10000	343	6710	4468	946	4037	473	3759	1490	1683	668	4564	1631	1209
Zr	ppm	15	14	15	10	15	12	9	11	14	9	9	11	13	11	9	9

Appendix D
Lime Demand Test Results

Table D-1. Lime Demand Test Results

Description	SampleID	Initial		60 min		Stage 1 Lime Addition		After Stage 1		60 min	Stage 2	Endpoint	Lime demand (kgCa(OH) ₂ /tonne)		
		pH	EC (mS/cm)	pH	EC (mS/cm)	mL (10 g/L)	mL (100 g/L)	pH	EC (mS/cm)	pH	mL (10 g/L)	pH	Stage 1	Stage 2	Overall
Waste Rock	VTP17A	5.90	0.77	5.01	0.98	12		9.58	1.00	9.39	0	9.39	0.60	0.00	0.60
	VTP17B			6.33	0.81	4		9.68			2	9.63	0.20	0.08	0.28
	VTP18A	6.56	1.50	6.56	1.50	6		9.70	1.87	8.75	1	9.45	0.30	0.05	0.35
	VTP18B			6.57	1.49	7		9.69		8.85	1	9.56	0.35	0.05	0.40
	VTP19			6.06	1.15	4		9.62		8.54	2	9.95	0.20	0.10	0.30
	VTP20	6.12	2.40	5.98	2.51	25		9.50	2.61	7.80	5	9.72	1.25	0.25	1.50
	VTP21			3.86	2.10	54		9.69		9.08	6	9.60	2.70	0.30	3.00
	VTP22			4.80	1.86	26		9.64			5	9.53	1.30	0.23	1.53
	VTP23A			5.71	2.28	18		9.74		9.47			0.90	0.00	0.90
	VTP23B	6.22	1.88	6.22	1.88	11		9.55	2.47	8.53	2	9.75	0.55	0.10	0.65
	VTP24A	6.08	0.96	6.08	0.96	7		9.94	1.32	8.84	1	9.60	0.35	0.05	0.40
	VTP24A			6.32	1.94	8		9.62			3	9.53	0.40	0.13	0.53
	VTP25A	5.31	0.72	6.02	0.81	10		9.55	1.00	7.42	4	9.77	0.50	0.20	0.70
	VTP25B			5.42	1.27	26		9.53		8.9	2	9.61	1.30	0.10	1.40
	VTP38			6.69	1.72	3		9.76			2	10.22	0.15	0.10	0.25
	VTP26A	3.49	2.20	3.48	2.40	35		9.53	2.61	8.07	5	9.88	1.75	0.25	2.00
	VTP27A	6.01	0.75	6.02	0.98	9		9.51	1.25	8.9	2	9.49	0.45	0.10	0.55
	VTP27B			6.82	1.77	3		9.76			2	9.83	0.15	0.10	0.25
	VTP27C			6.68	1.59	7		9.54		8.80	1	9.63	0.35	0.05	0.40
	VTP28A			6.12	0.65	4		9.73			1	9.66	0.20	0.05	0.25
	VTP28B			6.04	1.68	8		9.67			3	9.59	0.40	0.15	0.55
	VTP29A			6.62	1.03	6		9.58		9.16	1	9.49	0.30	0.05	0.35
	VTP29B			6.00	1.33	10		9.80		9.25	1	9.53	0.50	0.05	0.55
	VTP30			6.98	0.45	3		10.13		8.80	1	9.95	0.15	0.05	0.20
	VTP30A			7.19	0.58	5		9.61			1	9.54	0.25	0.05	0.30
	VTP30B	5.95	1.61	6.08	1.72	10		9.47	1.93	8.67	3	9.74	0.50	0.15	0.65
	VTP30C			6.70	0.69	5		9.69			2	9.75	0.25	0.10	0.35
	VTP31			7.65	0.33	3		9.61			1	9.49	0.15	0.05	0.20
	VTP32			7.80	0.24	3		9.56		9.10	1	9.60	0.13	0.03	0.15
	VTP33			7.39	0.23	3		9.59			2	9.61	0.15	0.10	0.25
	VTP35A	6.04	1.10	6.14	1.38	14		9.48	1.47	9.20	2	9.57	0.70	0.10	0.80
	VTP35B	6.76	0.54	7.78	0.64	3		9.8	0.64	8.93	2	9.51	0.15	0.10	0.25
	VTP36			7.22	0.74	6		9.62			2	9.55	0.28	0.08	0.35
	VTP37A	3.15	1.88			34		9.54	2.24	8.71	6	9.61	1.70	0.30	2.00
	VTP37B			5.43	1.97	20		9.67		9.35	1	9.63	1.00	0.05	1.05
	VTP39			6.50	1.08	6		10.04		9.20	1	9.82	0.30	0.05	0.35
	VTP40	4.58	0.20	4.25	0.20	6		9.61	0.42	7.78	2	10.21	0.30	0.10	0.40
	VTP41			4.30	1.78	30		9.87			4	9.59	1.50	0.18	1.68
	VTP42	2.48	2.41	2.50	2.77	60		9.80	2.09	9.05	2	9.51	3.00	0.10	3.10
	VTP43	4.23	0.61			12		9.50	0.94	8.83	2	9.74	0.60	0.10	0.70
	VTP44			6.94	0.60	2		9.79			1	9.51	0.10	0.05	0.15
	VTP34			7.65	0.25	3		9.6			1	9.51	0.15	0.05	0.20
Average													0.63	0.10	0.73

Table D-1. Lime Demand Test Results

Description	SampleID	Initial		60 min		Stage 1 Lime Addition		After Stage 1		60 min	Stage 2	Endpoint	Lime demand (kgCa(OH) ₂ /tonne)		
		pH	EC (mS/cm)	pH	EC (mS/cm)	mL (10 g/L)	mL (100 g/L)	pH	EC (mS/cm)	pH	mL (10 g/L)	pH	Stage 1	Stage 2	Overall
Baritic Fines	BF01	2.55	5.38	3.68	5.99	156	14	9.6	4.31	5.9	30	9.5	7.00	1.50	8.50
	BF02			2.57	4.27			9.9			40	9.7	7.80	2.00	9.80
	BF03			3.46	3.86			9.6			5	9.5	4.60	0.25	4.85
Average													6.47	1.25	7.72
Pit Oxide Fines	VPOF01	2.41	5.38	2.75	5.66	3	16	9.6	4.32	8.7	16	9.5	8.15	0.80	8.95
	VPOF02			2.58	4.31	151		9.6			16	9.5	7.55	0.80	8.35
	VPOF03			2.58	4.91		16	9.8			40	9.5	8.00	2.00	10.00
Average													7.90	1.20	9.10

Appendix E

Leach Extraction Test Results

Table E-1. Leach Extraction Test Results

Sample			As Is		With Lime		With Limestone	
			VG1	VG2	VG1	VG2	VG1	VG2
WATER	VOLUME	(mL)	750	750	750	750	750	750
SAMPLE	WEIGHT	(g)	250	250	250	250	250	250
pH			6.72	2.25	6.94	2.28	6.84	4.59
CONDUCTIVITY	(uS/cm)		915	4930	1079	4250	1086	1312
ALKALINITY	(mg CaCO ₃ /L)		9.5	0.0	21.5	0.0	21.0	1.0
ACIDITY	(pH 4.5)	(mg CaCO ₃ /L)	0.0	3250.0	0.0	1880	0.0	0.0
ACIDITY	(pH 8.3)	(mg CaCO ₃ /L)	52.5	6340.0	16.0	4040	28.5	435
SULPHATE	(mg/L)		1493	7546	1382	5654	1429	2211
Dissolved Metals (mg/L)								
Aluminum	Al	mg/L	<0.20	148	<0.20	110	<0.20	1.29
Antimony	Sb	mg/L	<0.20	<1.0	<0.20	<1.0	<0.20	<0.20
Arsenic	As	mg/L	<0.20	2.8	<0.20	1.4	<0.20	0.35
Barium	Ba	mg/L	0.028	<0.050	0.023	<0.050	0.028	<0.010
Beryllium	Be	mg/L	0.0054	<0.025	<0.0050	0.037	<0.0050	0.0086
Bismuth	Bi	mg/L	<0.20	<1.0	<0.20	<1.0	<0.20	<0.20
Boron	Bi	mg/L	<0.10	<0.50	<0.10	<0.50	<0.10	<0.10
Cadmium	Cd	mg/L	0.233	0.824	0.082	0.772	0.110	0.218
Calcium	Ca	mg/L	417	337	415	497	430	557
Chromium	Cr	mg/L	<0.010	0.406	<0.010	0.225	<0.010	<0.010
Cobalt	Co	mg/L	0.305	1.07	0.102	1.18	0.167	1.46
Copper	Cu	mg/L	0.144	39.9	0.030	28.8	0.042	0.195
Iron	Fe	mg/L	<0.030	1840	0.053	1080	0.043	45.1
Lead	Pb	mg/L	0.843	<0.25	0.247	<0.25	0.317	2.96
Lithium	Li	mg/L	0.036	0.164	0.020	0.069	0.032	0.073
Magnesium	Mg	mg/L	83.6	61.0	72.2	60.5	78.4	114
Manganese	Mn	mg/L	16.2	22.7	8.05	30.5	11.5	63.7
Molybdenum	Mo	mg/L	<0.030	<0.15	<0.030	<0.15	<0.030	<0.030
Nickel	Ni	mg/L	0.253	0.46	<0.050	0.67	0.115	1.24
Phosphorus	P	mg/L	<0.30	3.7	<0.30	<1.5	<0.30	<0.30
Potassium	K	mg/L	2.7	<10	2.1	<10	2.7	<2.0
Selenium	Se	mg/L	<0.20	<1.0	<0.20	<1.0	<0.20	<0.20
Silicon	Si	mg/L	1.73	2.10	1.16	3.30	1.24	2.37
Silver	Ag	mg/L	<0.010	<0.050	<0.010	<0.050	<0.010	<0.010
Sodium	Na	mg/L	<2.0	<10	<2.0	<10	<2.0	<2.0
Strontium	Sr	mg/L	0.912	0.142	0.973	0.767	0.850	0.319
Thallium	Tl	mg/L	<0.20	<1.0	<0.20	<1.0	<0.20	<0.20
Tin	Sn	mg/L	<0.030	<0.15	<0.030	<0.15	<0.030	<0.030
Titanium	Ti	mg/L	<0.010	<0.050	<0.010	<0.050	<0.010	<0.010
Vanadium	V	mg/L	<0.030	0.18	<0.030	<0.15	<0.030	<0.10
Zinc	Zn	mg/L	40.7	585	7.09	549	12.0	158

Appendix F

Column Test Results

Table F-1. Column Setup

Description #	Column #	Weight (kg)	Lime/ Limestone (grams)	Pore Volume (ml)
VG1with Lime	1	1.40	1.20	300
VG1with Limestone	2	1.48	2.29	310
VG 2 with Lime	3	1.39	4.64	310
VG 2 with Limestone	4	1.40	8.41	330

Table F-2. Column Test Results

Parameter	Units	VG1				VG 2			
		Lime		Limestone		Lime		Limestone	
Time	Days	15	21	15	21	15	21	15	21
Vol.	(mL)	305	320	300	335	295	340	310	310
Redox	(mV)	188	282	176	277	274	335	350	264
Immediate	pH	6.43	6.46	6.36	7	2.32	2.64	4.5	4.79
Before	pH	7.51	7.22	7.62	7.49	2.37	2.39	4.51	4.84
CONDUCTIVITY	(uS/cm)	1445	1424	1312	1336	6910	8300	1757	1650
ALKALINITY	(mg CaCO ₃ /L)	40	33	43.5	41	0	0	0	1.5
ACIDITY (pH 4.5)	(mg CaCO ₃ /L)	0	0	0	0	4430	2455	0	0
ACIDITY (pH 8.3)	(mg CaCO ₃ /L)	87	73	79	45.5	13995	9030	406.5	264
SULPHATE	(mg/L)	2800	1845	2490	1700	14165	10565	3065	21135
Total Metals (mg/L)									
Aluminum	mg/L	<0.20	1.88	<0.20	0.26	474	302	1.04	0.87
Antimony	mg/L	<0.20	<0.20	<0.20	<0.20	<2.0	<2.0	<0.40	<0.20
Arsenic	mg/L	<0.20	<0.20	<0.20	<0.20	5	<2.0	<0.40	<0.30
Barium	mg/L	0.07	0.234	0.03	0.098	<0.10	<0.10	0.092	0.111
Beryllium	mg/L	0.0052	<0.0050	<0.0050	<0.0050	0.106	0.062	<0.010	<0.0050
Bismuth	mg/L	<0.20	<0.20	<0.20	<0.20	<2.0	<2.0	<0.40	<0.20
Boron	mg/L	<0.10	<0.10	<0.10	<0.10	<1.0	<1.0	<0.20	<0.10
Cadmium	mg/L	0.307	0.262	0.253	0.18	2.58	1.68	0.374	0.292
Calcium	mg/L	478	441	431	405	293	255	455	450
Chromium	mg/L	<0.010	<0.010	<0.010	<0.010	1.13	0.75	<0.020	<0.010
Cobalt	mg/L	0.508	0.362	0.479	0.298	4.33	2.73	2.94	1.76
Copper	mg/L	0.678	0.65	0.452	0.364	96.5	59	0.134	0.087
Iron	mg/L	0.464	2.68	0.192	0.416	4060	2650	6.62	5.2
Lead	mg/L	1.05	1.51	0.598	0.643	<0.50	<0.50	1.95	2.28
Lithium	mg/L	0.103	0.079	0.094	0.076	0.45	0.3	0.153	0.111
Magnesium	mg/L	364	194	352	198	260	182	415	218
Manganese	mg/L	52.4	39.4	50.7	35.4	114	77	199	115
Molybdenum	mg/L	<0.030	<0.030	<0.030	<0.030	<0.30	<0.30	<0.060	<0.030
Nickel	mg/L	0.398	0.253	0.379	0.203	2.4	1.99	2.11	1.34
Phosphorus	mg/L	<0.30	<0.30	<0.30	<0.30	<3.0	<3.0	<0.60	<0.30
Potassium	mg/L	10.5	9	9.1	9	<20	<20	5.9	4.4
Selenium	mg/L	<0.20	<0.20	<0.20	<0.20	<2.0	<2.0	<0.40	<0.20
Silicon	mg/L	2.94	5.29	2.65	2.7	33.1	35.4	8.27	10.1
Silver	mg/L	<0.010	<0.010	<0.010	<0.020	<0.10	<0.10	<0.020	<0.010
Sodium	mg/L	4.3	2.2	4.1	2.2	<20	<20	<4.0	<2.0
Strontium	mg/L	2.26	2.02	2.04	1.85	0.39	0.297	0.33	0.316
Thallium	mg/L	<0.20	<0.20	<0.20	<0.20	<2.0	<2.0	<0.40	<0.20
Tin	mg/L	<0.030	<0.030	<0.030	<0.030	<0.30	<0.30	<0.060	<0.030
Titanium	mg/L	<0.010	0.079	<0.010	<0.010	<0.10	<0.10	<0.020	0.033
Vanadium	mg/L	<0.030	<0.030	<0.030	<0.030	0.58	0.34	<2.0	<0.030
Zinc	mg/L	52.7	41.4	45.8	29.7	1820	1190	224	147

Appendix G

Cost Estimation Supporting Calculations

Table G-1 Vangorda Pit Backfill and Vangorda Crek Chute

Item		Chute	Totals	
Drainage area (km ²)		20.1		
Design flood discharge (m ³ /s)		31		
HIDE-computed flood discharge (m ³ /s)		32		
GEOMETRY				
Channel length (m)		720		
Channel longitudinal slope (m/m)		0.05		
Channel bottom width (m)		5		
Channel sideslope (H:V)		2		
HYDRAULICS				
Depth of water (m)		1.03		
Flow cross-sectional area (m ²)		7.3		
Wetted perimeter (m)		9.6		
HIDE-top width (m)		9.12		
Hydraulic radius (m)		0.76		
Manning's n (s/m ^{1/3})		0.043		
Average cross-sectional velocity (m/s)		4.4		
Froude number		1.56		
Typical freeboard allowance (m)		0.50		
HIDE - Freeboard on outside bends (m)		1.09		
EROSION PROTECTION				
Riprap median diameter (m)		0.48		
Minimum thickness of riprap layer (m)		0.96		
Minimum thickness of filter layer (m)		0.3		
HIDE - COMPUTED RIPRAP SIZE				
D50 for mild slope (m)		0.66355405		
Angle of repose of riprap (radians)		0.698131701		
Longitudinal angle of chute (radians)		0.049958396		
D50 for steep slope (m)		0.706480509		
HIDE - HEIGHTS OF TRAPEZOIDAL SHAPES				
Channel (m)		1.53		
Channel + Riprap (m)		2.49		
Channel + Riprap + Filter (m)		2.79		
HIDE - TOP WIDTHS OF TRAPEZOIDAL SHAPES				
Channel (m)		11.12		
Channel + Riprap (m)		15.41325052		
Channel + Riprap + Filter (m)		16.7548913		
HIDE - BOTTOM WIDTHS OF TRAPEZOIDAL SHAPES				
Channel (m)		5		
Channel + Riprap (m)		5.453250517		
Channel + Riprap + Filter (m)		5.594891303		
HIDE - AREA OF TRAPEZOIDAL SHAPES				
Channel (m2)		12.3318		
Channel + Riprap (m2)		25.97879379		
Channel + Riprap + Filter (m2)		31.17794674		
QUANTITIES				
	Volume of soil excavation (1000 m ³)	0.0	0.0	
	Volume of fill material (1000 m ³)	9,073	9,073	
	Volume of placed riprap (1000 m ³)	9.8	9.8	
	Volume of placed filter material (1000 m ³)	3.7	3.7	
	Volume of placed till material (1000 m ³)	555	555	
	Cost Estimates		Totals	Units Unit rates
1	Rock Excavation: drill, blast, muck, load and haul 1 km	\$0	\$0	m ³ \$23.00
2	Soil Excavation: load, haul and dump locally	\$0	\$0	m ³ \$6.00
3	Waste Rock: load, haul 3km and dump	\$36,291,896	\$36,292,000	m ³ \$4.00
4	Riprap: drill, blast and stockpile locally	\$98,258	\$98,000	m ³ \$10.00
5	Riprap: screen, load, haul and stockpile (10km)	\$294,775	\$295,000	m ³ \$30.00
6	Riprap: load from local stockpile, haul and place (.5km)	\$137,562	\$138,000	m ³ \$14.00
7	Filter layer: supply and stockpile locally	\$24,332	\$24,000	m ³ \$6.50
8	Filter layer: load , haul, place and compact (3km)	\$52,407	\$52,000	m ³ \$14.00
9	Till: Load from stockpile, haul, place and compact	\$4,439,696	\$4,440,000	m ³ \$8.00
10	Soil : place and compact	\$0	\$0	m ³ \$4.50
11	Access Road: clearing and grubbing	NA		m ³ \$2.50
12	Access Road: construction	NA		m ³ \$6.00
13				
Grand Total		\$41,338,927	\$41,339,000	

Table G-2. Estimation of Lime Amendment Costs

Item	Waste Rock	Oxide / Baritic Fines	Units
Volume	8,786,974	286,000	m3
Lime Demand	0.84	8.41	kg Ca(OH) ₂ /tonne
Lime Demand	1.43	14.30	kg Ca(OH) ₂ /m3
Total Lime	12,548	4,089	tonnes
Cost	\$ 4,417,000	\$ 1,439,000	
Total Cost			\$ 5,856,000

Assumptions

Item	Value	Units
Bulk Density	1.7	
Lime costs	320	\$/short tonne
	352	\$/tonne

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Table G-3. Faro Oxide Fines Relocation and Lime Amendment

Item	Low Estimate	High Estimate	Units
Unit Cost - Haulage	8.00	\$/m3	
Lime Cost	352	\$/tonne	FOB the site
Lime Demand	14.3	14.3	kg/m3
Volume	680000	1770000	m3
Haul and Place	\$ 5,440,000	\$ 14,160,000	
Lime Amendment	\$ 3,422,000	\$ 8,908,000	
Total	\$ 8,862,000	\$ 23,068,000	