

## TECHNICAL MEMORANDUM

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### TAILINGS & WASTE CHARACTERISATION REVIEW - NALUNAQ, GREENLAND

#### 1.0 BACKGROUND

Nalunaq is a past producing gold mine, located in the valley Kirkespirdalen, near Napasorsuaq (Kirkespiret), 40 km NE of the town of Nanortalik, in South Greenland. Initially opened in 2004 and closed in 2013, Nalunaq A/S are preparing to re-open the mine. In this Technical Memorandum is presented a summary of the assessment of historical waste rock and tailings geochemical characterisation together with that from recent testing undertaken by SGS (2021). The recent geochemical characterisation of the tailings was undertaken to support Nalunaq A/S Environmental Impact Assessment reporting as a permitting requirement.

The approach taken was to re-evaluate historic available data and incorporate consideration of the results of the additional static test work conducted by SGS in 2021 (SGS, 2021) to assess the necessity of further kinetic test work based on guidance published by the Danish Centre for Environment and Energy (DCE, 2018). This Technical Memorandum therefore fulfils the requirement of the DCE with respect to the second step of the three-step approach regarding geochemical test work. Based on the information available recommendations are made with regard to the need for additional geochemical characterisation.

#### 2.0 ASSESSMENT

##### Geological Setting, Vein Composition and Historic Characterisation

The future extraction waste (waste rock) will mainly consist of fine-grained amphibolite (meta-basalt) with lesser quantities of coarse-grained dolerite (meta-gabbro) and cross-cutting aplite dykes. In the tailings a proportion of the gold-bearing quartz will also be present. The quartz vein (known as the Main Vein) hosting the gold mineralisation averages 0.7 m thickness.

A geological review of the composition differences between future waste rock (to be mined) and the existing waste rock on site has been made by SRK (2021a). The conclusion is that:

*“SRK does not anticipate any significant variations between either the Main Vein or the host rock between or within the various mine Blocks. With this in mind, it is likely that the geochemical behaviour of any future waste rock materials in terms of its acid rock drainage and metal leaching (ARDML) potential will also be similar to that predicted and / or experienced from the existing waste rock on site. Geochemical characterisation work pertains to the historically mined areas already exists from previous work and have been submitted to the MLSA. While SRK cannot rule out the possibility for small scale local variations, these would likely be minor”.*

In addition to the above, SRK (2021b) also undertook a vein material description. The Main Vein slightly crosscuts the foliation at Nalunaq and can be traced at surface for over 1 km on the east- and north-facing slopes of the mountain. About 250 m across the western face of the mountain and at least 800 m down the south-western slopes. The Main Vein is represented by a 0.5 - 2 m thick quartz vein with alteration envelope extending for up to 1 - 1.5 m in width on both sides of the vein. The Main Vein is observed as holding distinct high-grade shoots (gold) that generally dip in the plane of the vein towards the SE. These plunges, where identified, are the key targets for past and potential new mining operations. These include the Mountain, Target and South Blocks and the newly identified Valley Block as a possible fourth objective. The conclusion of the geological review of the Main Vein is that:

*“Based upon SRK’s experience at the project, it is possible to conclude that while these plunges are separate targeted ore bodies within the wider Main vein, the material remains very similar and as a result of the same gold precipitation event. As such, SRK does not anticipate any significant variations in geochemistry between Main vein material from either the older mine areas or the newly defined Valley block”.*

The conclusion from the geochemical characterisation undertaken by SRK from available multi-element assaying is that the addition of the Valley Block to the previously known resource would result, possibly, only in minor changes of the geochemical composition of any tailings material. However, the quantity of data from the Valley Block is currently smaller than from other areas, and it is assumed that as the total data count increases in the Valley Block following further exploration the geochemical populations evaluated will further align. Presently, at an average, concentrations of As, W, Cu, S, Pb for example are lower in the Valley Block whilst Co is slightly higher (but for example still below the limit value for less sensitive land use in Sweden<sup>1</sup>).

Historic test work (Kvaerner, 2002) reported neutralisation potential/acid generation potential (NP/AP) ratios of 2.9 – 3.4 for waste rock and ore materials, indicating a substantial net buffering capacity and no potential for acid generation where the lower value corresponded to waste rock and not the ore.

The mineralogical and elemental composition of the waste rock should mimic that of the tailings except for the quartz that is associated with the mineralisation. Evaluation of the tailings should therefore give valuable input to the waste rock characteristics and future behaviour.

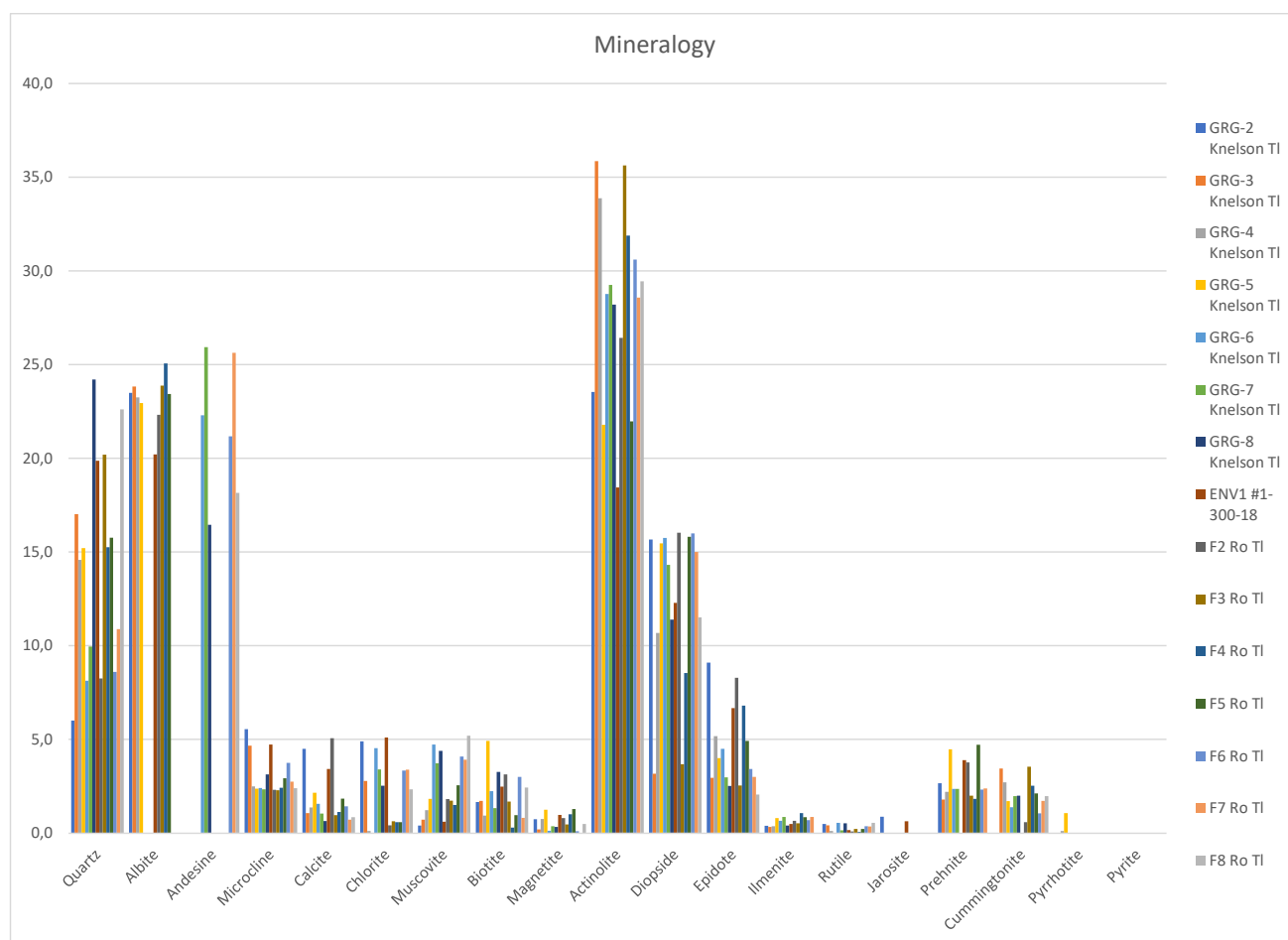
### **Tailings: 2021 Geochemical Characterisation**

A total of 15 samples representative of future tailings were scheduled by Nalunaq A/S personnel for geochemical test work by SGS (2021) in line with the recommendations for mining projects in Greenland (DCE, 2018). The test results are presented in Appendix A.

The mineralogy reported by SGS (2021) reflects the geological setting, with calcium-rich silicates (actinolite, diopside, epidote), sodium-aluminium silicates and quartz, pyrrhotite and trace of pyrite in two out of 15 samples (Figure 1). The concentration levels of potential contaminants of concern are much too low for the mineralogical investigation to identify any specific mineral-bearing phases. It is possible these exist as specific minerals but it is more likely that they are also present in other minerals e.g. pyrite (identified in two samples at low concentrations) as impurities.

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<sup>1</sup> <https://www.naturvardsverket.se/upload/stod-i-miljoarbetet/vagledning/fororenade-omraden/berakning-riktvarden/generella-riktvarden-20160707.pdf>



**Figure 1: Mineralogy of the tailings samples (weight %)**

The full analyses of the tailings samples indicate that the only potential constituents of concern (PCOC) might be arsenic (As) and possibly tungsten (W) (SGS, 2021; Appendix A).

Variations in trace element content of the tailings are normal for small-scale testwork due to its limited ore usage. In full scale a blend of ore would be fed into the beneficiation plant in order to keep an even ore mixture. Thus, the expected tailings would also align towards the average/median values of trace element content. Variations are normal to be present in different parts of the ore body and are managed during the beneficiation process by blending.

Modified acid base accounting (ABA) (Appendix A) on the 15 tailings samples shows a NP/AP ratio between 3.23 and 29.2 whereas the sulphide-S is between <0.04 and 0.21 wt-%. CO<sub>3</sub>-NP/AP ratio varies from 2 (three samples of gravity tailings) with the other samples varying between 3.1 up to 24.2. All samples are thus net buffering based on NP/AP and for CO<sub>3</sub>-NP/AP, and this is verified by the net acid generating (NAG) tests conducted.

The sequential extractions exhibited very low recovery grades, typically below 50% and often below 20%. This is a strong indicator that the elements are largely locked into silicate matrix and thereby unavailable for leaching. This should be even more pronounced for the waste rock exhibiting a smaller surface area than the fine-grained tailings and thereby preventing leaching of non-exposed leachable trace elements.

Some trace elements are leachable though although the leached amounts (summarising the extractable, reducible, and oxidisable phase ranges) are still very low. It is also considered unlikely that geochemical

conditions will prevail that enable the extractable, reducible and oxidisable phases to be leached simultaneously. Thus, the deposition methodology may, in due time, enable leaching of a portion of the trace element but only the relevant fraction depending on the environmental context into which it is placed. For example, dry stacking of filtered tailings may over time (without any oxygen control) generate a mobilisation of adsorbed and reduced fractions where over time the adsorbed fraction is washed out by rain and snowmelt whereas the reduced fraction is mobilised by oxidation. Water management and a cover decreases this potential transport and mobilisation pathway.

Arsenic that was identified as a PCOC based on somewhat elevated concentrations compared to five times the crustal abundance (Smith & Huyck, 1999) is predominantly leached in reducible and extractable phases (particularly if gravity and flotation tailings are added together) indicating presence in non-sulfidic minerals. Thus, it is considered that the potential for leaching under atmospheric conditions is likely to be negligible considering the total amount in the tailings (260 ppm in gravity tailings and 52 ppm in flotation tailings).

A range of leaching tests were conducted, including toxicity characteristic leaching procedure (TCLP) which is a very aggressive leaching test under acidic conditions. Such conditions are not expected in the field setting since rainwater and snowmelt on site will only be slightly acid and ion-weak, and will not create sufficient acidity to dissolve the matrix of the rocks in the way the test is designed to do. The pH of the leachate solution remained buffered, which is indicative of the net buffering capacity of the tailings. Arsenic varied for the gravity tailings between 0.1 - 1 mg/L whereas the flotation tailings reported an order of magnitude lower values. This is explained by the higher (6.15) pH of the flotation leachate compared to the gravity leachate (4.5). The results were within the limits for all controlled parameters specified for this test procedure.

The synthetic precipitation leaching procedure (SPLP, leaching with pH 4.2 solution) test as well as shake flask extraction (SFE) and process water all reported arsenic above the Canadian Council of Ministers for the Environment (CCME, 2017) guidelines and in some cases equivalent to the less stringent Metal and Diamond Mining Effluent Regulations (MDMER; Government of Canada, 2021) guidelines, and overall, below the International Finance Corporation's effluent quality guidelines (IFC, 2007). These guidelines are set with safety factors to avoid any impacts to selected sensitive species (i.e. the most sensitive species). Moreover, the seepage from existing as well as future extraction waste will be transported via groundwater into the Kirkespir River where the dilution is approximately 600 times. This means that instead of reviewing the seepage itself it should be reviewed in the light of any potential impacts to the river, during and after ongoing production.

The released arsenic in the SPLP and SFE tests mimics most closely arsenic mobilised during processing (process water is partially contained in dewatered tailings as well as adsorbed onto mineral surfaces). Toxicity tests verified that, except for a single anomalous Daphnid mortality, both composite process water samples reported 100% survival rates and non-lethal designations for both species tested.

### 3.0 CONCLUSIONS AND RECOMMENDATIONS

The following conclusions and recommendations are made:

- The mineralogy and composition of the waste rock and tailings samples reflects their geological denomination. The concentrations of most constituents are low and the only identified PCOC was arsenic, but still in low concentration (average 149 µg/g and median 98 µg/g). The samples show a net buffering capacity and low sulphide content thus classifying as inert based on the European Union's classification of ARD potential (European Commission, 2009)<sup>2</sup>.

<sup>2</sup> <https://eur-lex.europa.eu/legal-content/EN/TXT/HTML/?uri=CELEX:32009D0359&from=EN>

- Sequential extractions of the tailings reveal that arsenic largely is reducible (typically oxidises) or in exchangeable phases (typically adsorbed). Thus, exposure to atmospheric conditions should not lead to an increased mobility of arsenic neither in the short or the long term. The available pool of arsenic that is susceptible to future leaching is also small, as was shown by the full analyses.
- Toxicity tests show that process water is designated as non-lethal. Any concerns regarding the toxicity of the process water and its potential impact to the recipient surface water receptors should thus also be negated.
- The potential toxicity of leachate from the waste rock was investigated and its non-lethal characteristics were verified.
- In addition to the above it is noted that an extensive environmental monitoring programme has been in place on site and no detrimental impacts from the historical mining have been identified to date. The similarity between the historically mined areas and future additional mining prospect as well as present mine waste and the future mine waste have been investigated by SRK (2020a and b) and no major change of composition are to be expected in the future.
- Based on the information presented above and based on the inert classification of the materials as per European Union's (European Commission, 2009) definition of ARD potential, the low concentration of the only identified PCOC as well as its leaching characteristics, it is considered that based on the DCE (2018) guidance that kinetic test work is not necessary.

## 4.0 REFERENCES

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**APPENDIX A**

**Tailings Geochemical  
Characterisation Test Data  
(SGS, 2021)**

Table 1: Full geochemical analyses related to 5X Crustal Abundance (CA)

Parameter	Unit	GRG-2 Knelson TI	GRG-3 Knelson TI	GRG-4 Knelson TI	GRG-5 Knelson TI	GRG-6 Knelson TI	GRG-7 Knelson TI	GRG-8 Knelson TI	ENV1 #1- 300- 18	F2 Ro TI	F3 Ro TI	F4 Ro TI	F5 Ro TI	F6 Ro TI	F7 Ro TI	F8 Ro TI	Average	Median	Crustal Abundance, Ca (Smith & Huyck, 1999)	5X CA
Hg	µg/g	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	0,08	0,4
Ag	µg/g	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	<1	<1	0,07	0,35
Al	µg/g	62000	58000	60000	51000	60000	58000	44000	54000	63000	59000	61000	52000	58000	55000	44000	55933	58000	80000	400000
As	µg/g	290	260	150	460	290	210	160	80	71	39	18	98	52	32	22	149	98	2	10
B	µg/g	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1	-	-
Ba	µg/g	99	68	29	82	89	56	120	110	96	73	31	85	83	51	92	78	83	430	2150
Be	µg/g	3,51	0,62	1,31	2,01	1,32	1,3	1,3	2,24	3,43	0,62	1,35	2,16	1,25	1,3	1,2	1,66	1,31	3	15
Bi	µg/g	1,71	0,48	8,99	1,68	2,7	1,0	0,99	3,16	0,89	0,30	3,00	1,17	0,57	0,41	0,4	1,83	1,00	0,2	1
Cd	µg/g	0,31	0,17	0,24	0,36	0,35	0,30	0,18	0,19	0,23	0,16	0,22	0,29	0,21	0,17	0,16	0,24	0,22	0,18	0,9
Co	µg/g	38	39	41	31	39	38	31	19	23	31	29	17	25	27	22	30	31	25	125
Cr	µg/g	310	360	300	330	280	160	220	210	230	290	200	260	240	150	160	247	240	200	1000
Cu	µg/g	140	120	150	170	140	150	140	36	22	30	14	14	24	33	16	80	36	60	300
Li	µg/g	79	63	50	49	46	30	29	60	79	67	48	50	42	28	26	50	49	30	150
Mo	µg/g	6,4	8,1	7,7	10	5,1	1,4	3,6	3,5	3,8	6,6	4,4	7,5	3,8	0,9	1,6	5,0	4,4	2	10
Ni	µg/g	150	160	150	130	150	110	130	88	88	130	94	78	99	77	73	114	110	80	400
Pb	µg/g	4,6	1,3	1,4	1,2	9,0	3,7	9,2	3,0	1,4	1,2	0,53	0,91	2,5	2,3	2,2	3,0	2,2	16	80
Sb	µg/g	3,2	6,2	3,5	2,2	2,4	2,2	2,1	2,0	2,6	2,4	2,2	1,7	1,8	2,0	1,5	2,5	2,2		-
Se	µg/g	< 0.7	< 0.7	< 0.7	0,8	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	< 0.7	0,09	0,45
Sn	µg/g	12	< 6	8,1	8,3	6,6	< 6	< 6	8,7	13	< 6	7,4	8,9	< 6	< 6	< 6	9,1	8,5	2,5	12,5
Th	µg/g	1,2	0,47	0,38	0,36	3,0	1,4	1,4	2,5	0,40	0,21	0,27	0,32	4,1	2,6	1,3	1,3	1,2	10	50
Tl	µg/g	0,60	0,42	0,22	0,83	0,42	0,24	0,53	0,50	0,57	0,44	0,20	0,84	0,36	0,22	0,50	0,5	0,44	1	5
U	µg/g	0,78	0,12	0,16	0,48	0,67	0,56	0,87	0,61	0,57	0,12	0,15	0,48	0,57	0,55	0,58	0,5	0,56	3	15
W	µg/g	57	3,7	1,6	13	7,07	33	2,3	8,5	69	3,5	1,6	13	6,19	28	2,1	16,6	7,07	1	5
Y	µg/g	16	14	15	16	16,6	15	13	13	16	14	14	16	16,4	15	13	14,9	15	30	150
Zn	µg/g	94	71	79	84	98	83	74	67	83	65	66	79	74	68	60	76,3	74	70	350

Notes: Red shading indicates exceedance of parameter concentration relative to Crustal Abundance.



**Table 2: Modified ABA test work on tailings**

Parameter	Unit	GRG-2 Knelson TI	GRG-3 Knelson TI	GRG-4 Knelson TI	GRG-5 Knelson TI	GRG-6 Knelson TI	GRG-7 Knelson TI	GRG-8 Knelson TI	ENV1 #1- 300- 18	F2 Ro TI	F3 Ro TI	F4 Ro TI	F5 Ro TI	F6 Ro TI	F7 Ro TI	F8 Ro TI
Paste pH	no unit	8,51	8,77	8,14	8,51	9,29	8,74	8,70	8,26	8,85	8,88	8,74	8,85	9,45	9,06	8,99
Fizz Rate	no unit	2	2	2	2	2	2	2	3	3	2	2	3	2	2	2
Sample weight	g	2,00	2,04	1,99	2,01	1,99	1,99	2,03	2,02	2,02	1,98	2,01	2,01	2,05	2,01	1,98
HCl added	mL	30,00	20,00	20,00	20,00	24,20	20,00	20,00	25,60	20,00	20,00	20,00	20,00	20,00	20,00	20,00
HCl	Normality	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10
NaOH	Normality	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10	0,10
Vol NaOH to pH=8.3	mL	12,75	15,29	13,97	11,49	13,99	11,54	13,64	10,85	10,28	15,31	14,16	12,44	10,82	13,09	15,08
Final pH	no unit	1,50	1,24	1,36	1,36	1,67	1,66	1,38	1,50	1,50	1,13	1,21	1,22	1,90	1,47	1,20
NP	t CaCO <sub>3</sub> /1000 t	43,1	11,6	15,1	21,2	25,7	21,3	15,7	36,5	24,1	11,8	14,5	18,8	22,4	17,2	12,4
AP	t CaCO <sub>3</sub> /1000 t	5,94	1,25	3,12	6,56	5,31	2,19	2,81	1,25	1,25	1,25	1,25	1,25	1,25	1,25	1,25
Net NP	t CaCO <sub>3</sub> /1000 t	37,2	10,4	12,0	14,6	20,4	19,1	12,9	35,2	22,8	10,6	13,2	17,6	21,2	16,0	11,2
NP/AP	ratio	7,26	9,28	4,83	3,23	4,84	9,74	5,58	29,2	19,3	9,44	11,6	15,0	17,9	13,8	9,92
S	%	0,346	0,077	0,204	0,521	0,221	0,124	0,159	0,048	0,023	0,009	0,014	0,020	0,019	0,014	0,012
Acid Leachable SO <sub>4</sub> -S	%	0,16	< 0.04	0,10	0,31	0,05	0,05	0,07	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04
Sulphide	%	0,19	0,04	0,10	0,21	0,17	0,07	0,09	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04	< 0.04
C	%	0,447	0,116	0,149	0,222	0,193	0,197	0,135	0,408	0,436	0,116	0,133	0,190	0,162	0,149	0,092
CO <sub>3</sub>	%	3,02	0,420	0,580	0,909	0,674	0,550	0,330	1,82	2,00	0,440	0,590	0,804	0,550	0,420	0,295
CO <sub>3</sub> NP (calc'd)	t CaCO <sub>3</sub> /1000 t	50,1	7,0	9,6	15,1	11,2	9,1	5,5	30,2	33,2	7,3	9,8	13,3	9,1	7,0	4,9
CO <sub>3</sub> Net NP (calc'd)	t CaCO <sub>3</sub> /1000 t	44,2	5,72	6,51	8,53	5,88	6,94	2,67	29,0	32,0	6,05	8,54	12,1	7,9	5,7	3,6
CO <sub>3</sub> NP/AP (calc'd)	ratio	8,44	5,58	3,09	2,30	2,11	4,17	1,95	24,2	26,6	5,84	7,84	10,7	7,3	5,6	3,9
NP attributed to CO <sub>3</sub>	%	116	60	64	71	44	43	35	83	138	62	68	71	41	41	39