Appendix A

DRAWING LIST

Drawing register

Ramböll Sverige AB Dragarbrunnsgatan 78 B SE-753 20 Uppsala Telefon +46 (0)10 615 60 00	Technical sector Roads and Surfaces	Document number	Page/Pages 1/1
	Commision LUDVIKA GRUVOR PFS Nordic Ore Iron	Administrator J.Brask	
		Date 2015-03-27	
		Commision number 1320011785	
	Status Technical description	Revision date	Bet.

Drawing number	Bet.	Drawing title	Scale	Date	Revision date
		Blötberget			
M-30-1-01		Overview roads	1:5000	2015-03-27	
M-32-1-01		Industrial area 1	1:1000	2015-03-27	

o:\sto2\sgm\2014\1320011785\3_teknik\m\dokument\drawing register.doc

Drawing List

RAMBØLL	Discipline Buildings	Document no. Annex no. 1 — Page/Pages 1/1
		Administrator Hans Lundberg
	Project LUDVIKA MINES, NORDIC IRON ORE AB BLÖTBERGET BUILDINGS	Date 2015-03-06
		Commision no. 1320011785
	Status Preliminary design	Revision date 2015-03-27 — Rev.

Ramböll Sverige AB
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791 19 Falun
Phone 010-615 60 00
Fax 010-615 20 00

Drawing no.	Rev.	Description	Scale	Date	Revision date
		INDUSTRIAL AREA 1			
		Office/Locker-room			
A-40.1-101		Plan	1:100	PD 15-03-06	PD 15-03-27
A-40.2-111		Section	1:50	PD 15-03-06	
A-40.3-121		Facades	1:100	PD 15-03-06	PD 15-03-27
A-40.3-122		Facades	1:100	PD 15-03-06	
		Drillcore archive			
A-40.1-201		Plan	1:100	PD 15-03-06	
A-40.2-211		Section	1:100	PD 15-03-06	
A-40.3-221		Facades	1:100	PD 15-03-06	
A-40.3-222		Facades	1:100	PD 15-03-06	
		Stores			
A-40.1-301		Plan	1:200	PD 15-03-06	
A-40.2-311		Section	1:100	PD 15-03-06	
A-40.3-321		Facades	1:100	PD 15-03-06	
A-40.3-322		Facades	1:100	PD 15-03-06	
		Process plant			
A-40.1-501		Plan 1	1:250	PD 15-03-06	PD 15-03-27
A-40.1-502		Plan 2	1:250	PD 15-03-06	PD 15-03-27
		General maintenance workshop			
A-40.1-701		Plan	1:100	PD 15-03-06	
A-40.2-711		Section	1:50	PD 15-03-06	
A-40.3-721		Facades	1:100	PD 15-03-06	
A-40.3-722		Facades	1:100	PD 15-03-06	
		Vehicle workshop			
A-40.1-801		Plan	1:100	PD 15-03-06	
A-40.2-811		Section	1:100	PD 15-03-06	
A-40.3-821		Facades	1:100	PD 15-03-06	
A-40.3-822		Facades	1:100	PD 15-03-06	
		BS-SHAFT			
		Supply air building			
A-40.1-901		Plan	1:100	PD 15-03-06	

REGISTER & ISSUE SHEET

PROJECT NAME	Blotberget Iron Ore Project Interim Technical Study
PROJECT NUMBER	C22
OFFICE ADDRESS	Icon Business Centres Lake View Drive Sherwood Park Nottingham NG 15 0DT United Kingdom
SHEET NUMBER	**Report Issue**

DATE OF ISSUE	
DAY	10
MONTH	4
YEAR	15

REF NO.	TITLE	LAST REV	REVISION
C22-001	Schematic of Mine Infrastructure		O
C22-002	Schematic of Mine Conveyors Phase 1		O
C22-003	Schematic of Materials Handling from U/G Crusher to Railway		O
C22-004	G.A. of Portal Outfeed Conveyor/Crusher Arrangement -19 level		O
C22-005	G.A. of Dirt Belt from Surface Crusher to Rock Waste Dump		O
C22-006	G.A. of Conveyor Feed to ROM Silo		O
C22-007	G.A. of Load Out Arrangement to Railway		O
C22-008	Roadway Profiles - Combined Ramp (10m x 5.5m)		O
C22-009	Roadway Profiles - Conveyor Decline (6m x 5.5m)		O
C22-010	Roadway Profiles - Access Decline (5m x 5.5m)		O
C22-011	Roadway Profiles - Combined Ramp Passing Point (10m x 5.5m)		O
C22-012	G.A. of Crusher Station (-420 Level)		O
C22-013	Crusher Station Showing Plan Levels		O
C22-014	G.A. of Belt Transfer Station CV2 to CV1 (-353 Level)		O
C22-015	G.A. of Maintenance Workshop		O
C22-016	G.A. of Muster Station		O
C22-017	Typical G.A. of Detonator Store		O
C22-018	Conceptual Design of Portal Entrance to Combined Ramp		O
C22-019	G.A. of Pumping Ranges to Existing Shaft Phase 1 & 2		O
C22-020	G.A. of Belt Transfer Pocket at -420 level post crusher removal		O
C22-021	Electrical Schematics - Stage One Initial Development		O
C22-022	Electrical Schematics - Stage Two Initial Development		O
C22-023	Electrical Schematics - Stage Three Development Complete to -310m		O
C22-024	Electrical Schematics - Stage Four Development Complete to -420m		O
C22-025	Electrical Schematics - Stage Five Development Complete to -500m		O
C22-026	Electrical Schematics - Stage Six Development Complete to -580m		O
C22-027	Electrical Schematics - Stage Seven Development Complete to -660m		O

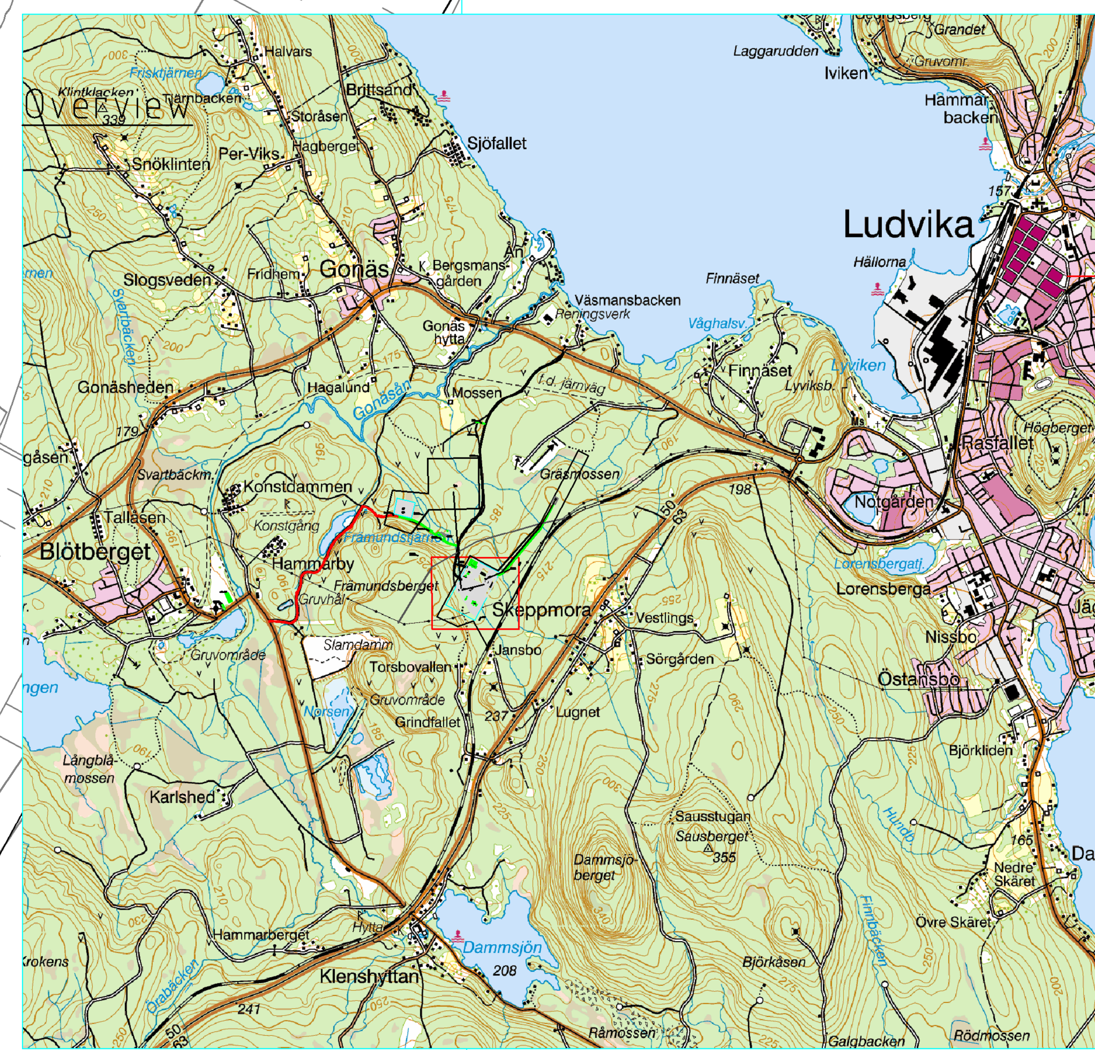
METHOD OF ISSUE	R
PURPOSE OF ISSUE	R

DISTRIBUTION		NO OF COPIES
	Issued as part of the Interim Technical Study	R

Method: (if not by post) F = Fax R = As Part of Report E = E-mail D = Disk * - A3 Size O=Original copy

Purpose of Issue A = Approval R= Report Production I = Information T = Tender C = Construction

Appendix B
SURFACE DRAWINGS



SWEREF 99 TM

Legend

	ROADSIDE ASPHALT
	REINFORCES ROAD
	INDUSTRIAL FENCE
	ROADSIDE GRAVEL
	AREA FOR LOCAL PLAN /BUILDING POINT
	GRAVEL AREA

BET	ANT	ÄNDRINGEN AVSER	DATUM	SIGN

Technical Report

LUDVIKA GRUVOR

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UPPDRAG NR	RITAD/KONSTR AV	HANDLÄGGARE
1320011785	J.BRASK	J.BRASK
DATUM 2015-03-09	ANSVARIG LISA FERNIUS	

BLÖTBERGET
INDUSTRY AREA 1

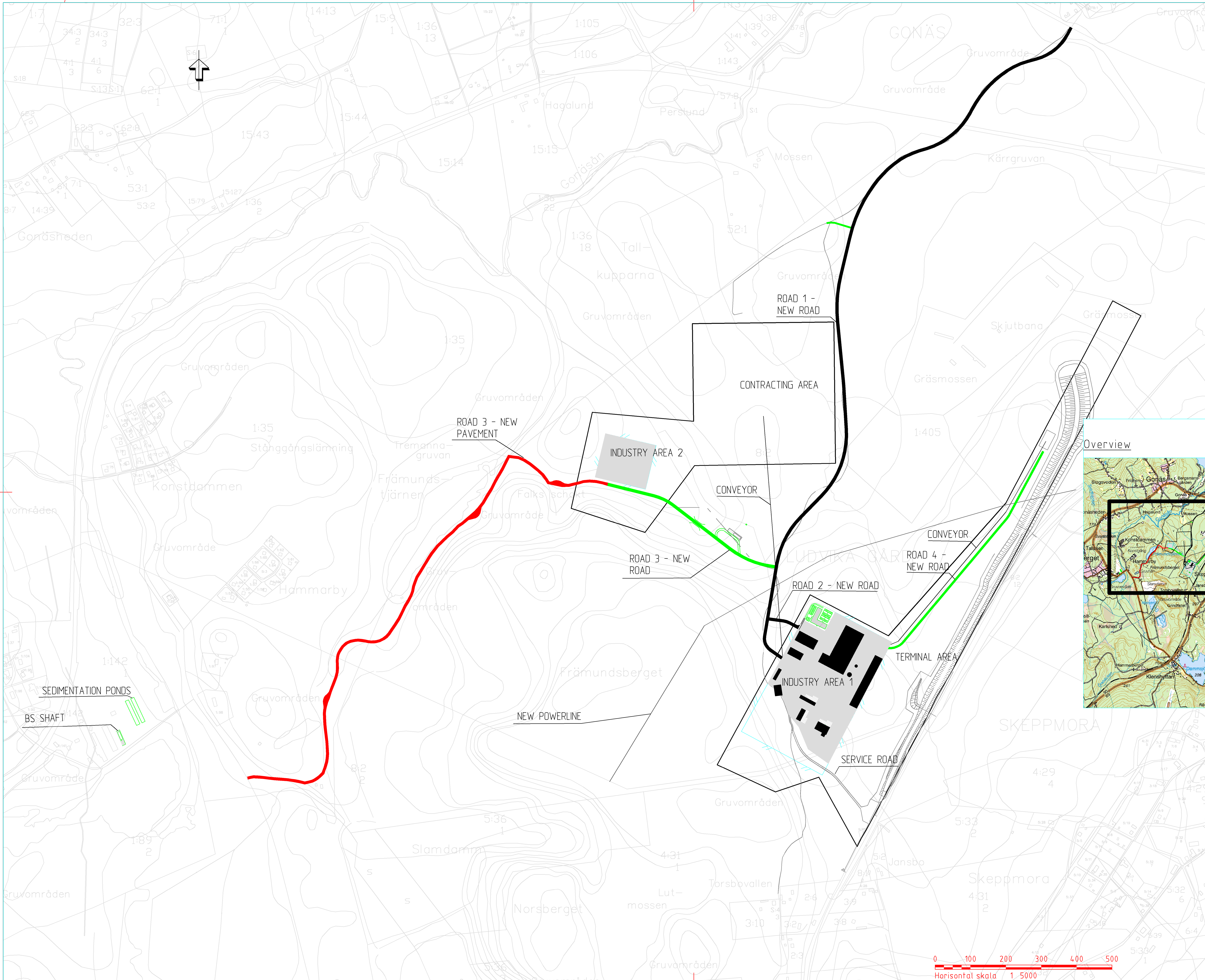
SKALA	NUMMER	BET
1:1000 (A1)	M-32-1-01	

FIL: O:\STO2\SGM\2014\1320011785\3_TEKNIK\M\RITDEF\M-32-1-01.DWG PLOTTAD: 2015-03-27 16:45 AV ANVÄNDARE JNBKSE

SWEREF 99 TM

Legend

ROADSIDE ASPHALT
REINFORCES ROAD
INDUSTRIAL FENCE
ROADSIDE GRAVEL
AREA FOR LOCAL PLAN
/BUILDING POINT
GRAVEL AREA



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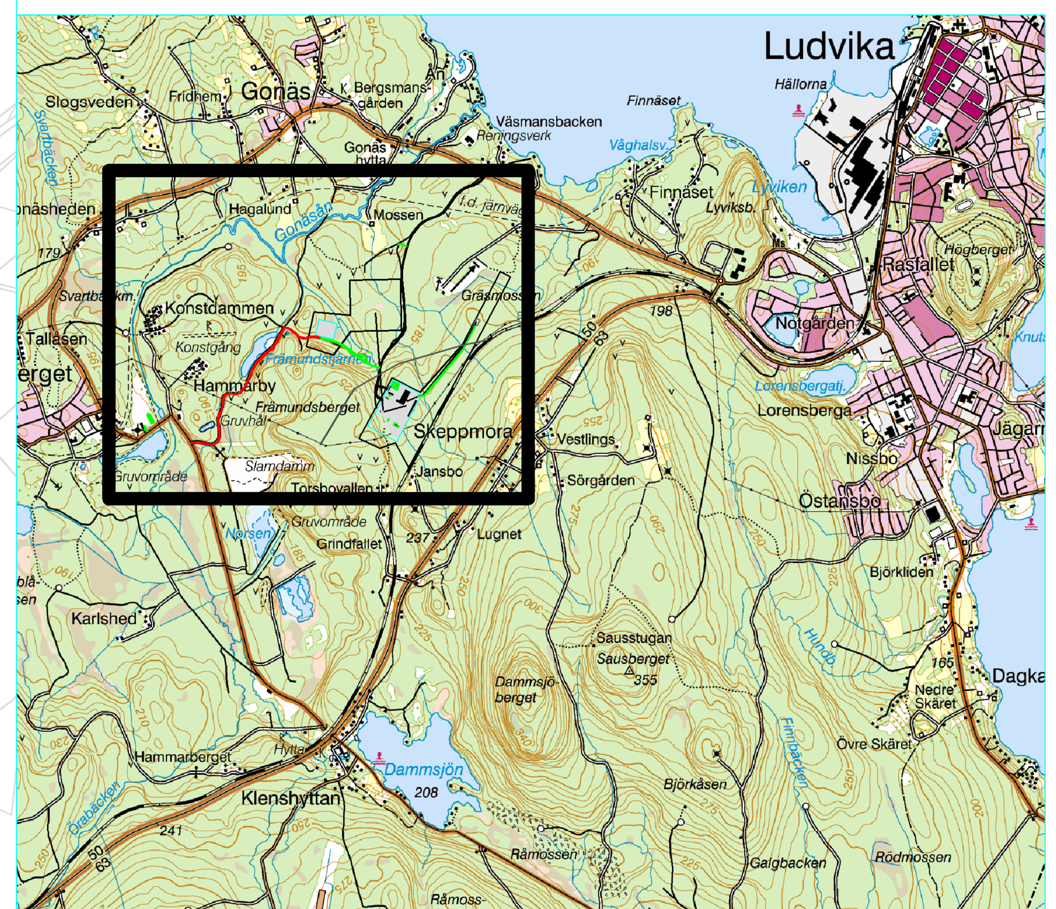

BET	ANT	ÄNDRINGEN AVSER	DATUM	SIGN

Technical Report

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UPPDRAG NR	RITAD/KONSTR AV	HANDLÄGGARE
1320011785	J.BRASK	J.BRASK
DATUM	ANSVARIG	
2015-03-27	LISA FERNIUS	

BLÖTBERGET
OVERVIEW ROADS

SKALA	NUMMER	BET
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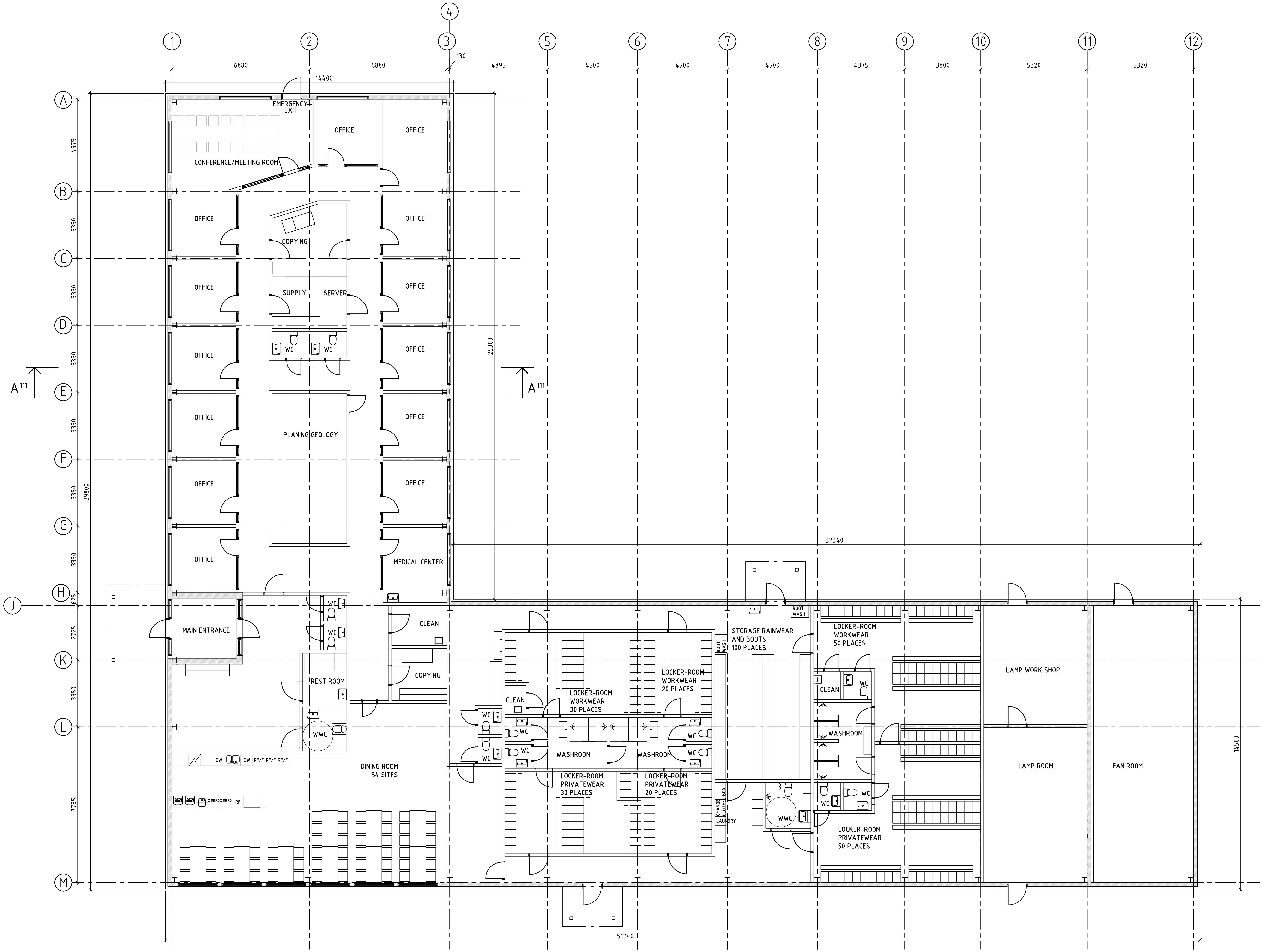

OFFICE/LOCKER-ROOM
1:100

SCALE A1 1:100 / A3 1:200

2015-03-27
~~2015-03-06~~

REV	ÄNT	CHANGE INTENDS	DATE	SIGN

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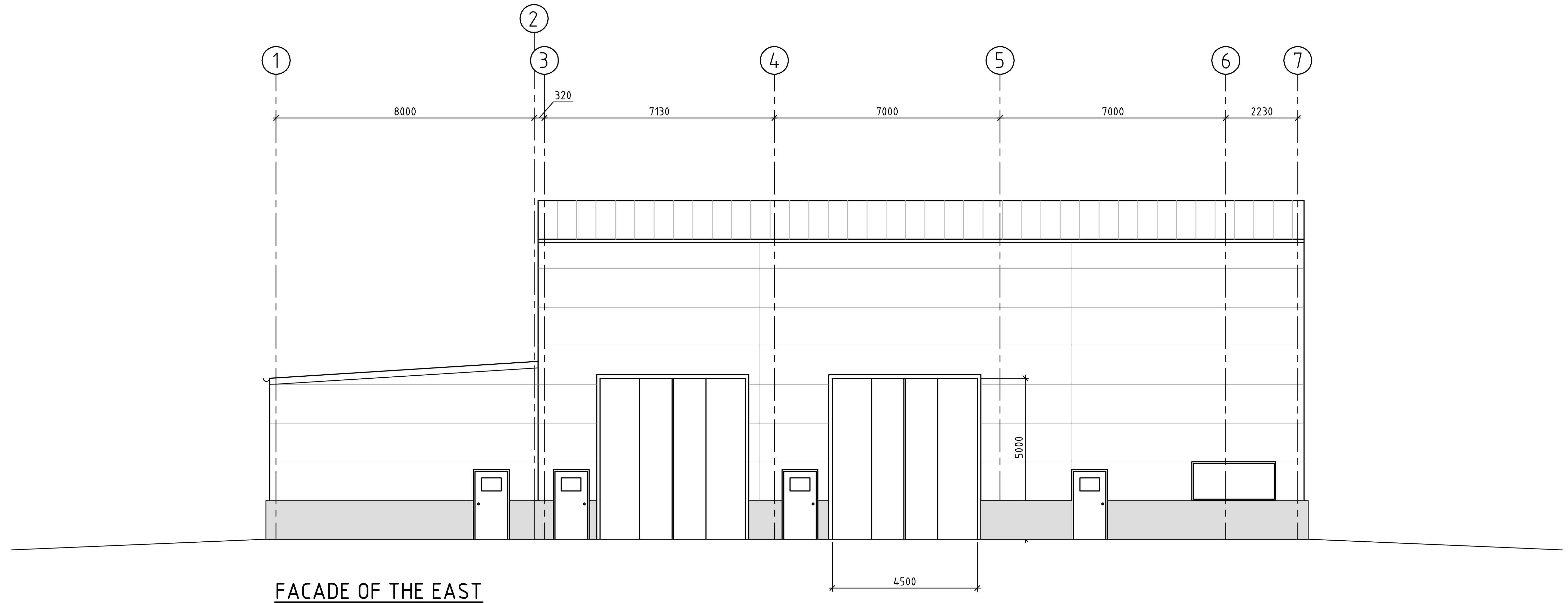
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
OFFICE/LOCKER-ROOM
PLAN

SCALE	DRAWING NO	REV
1:100/1:200	A-40.1-101	

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.



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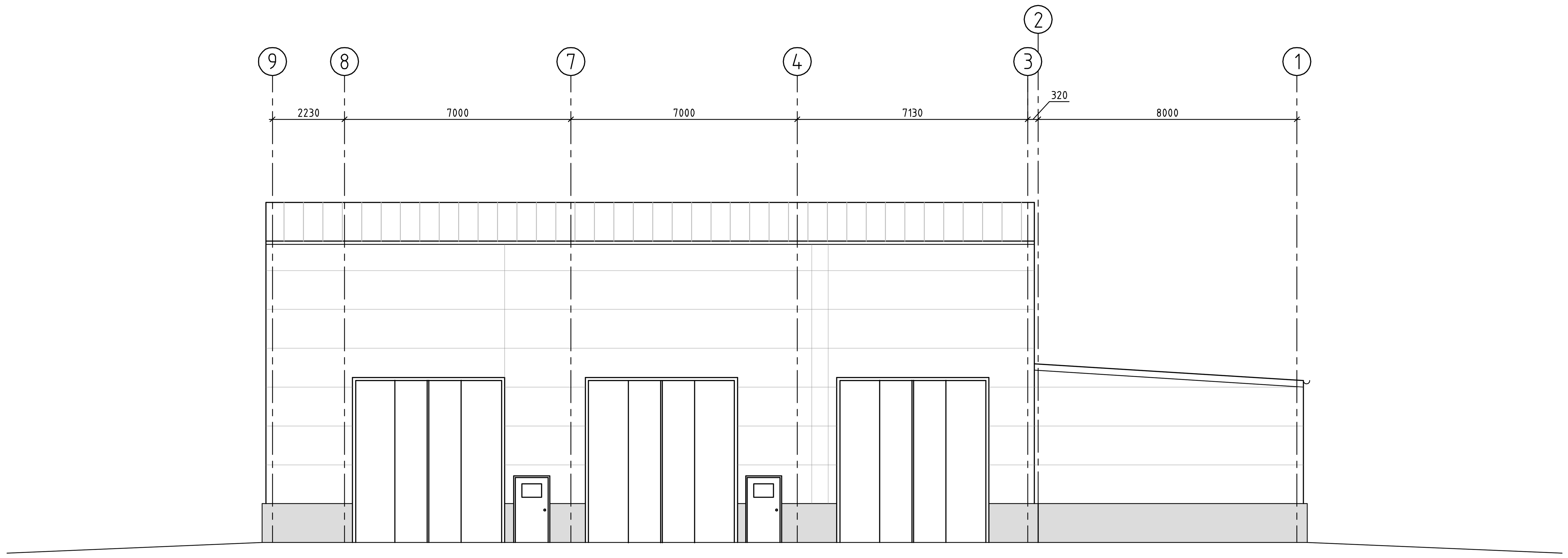

FACADE OF THE WEST
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2015-03-06

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BLÖTBERGET
INDUSTRIAL AREA 1
VEHICLE WORKSHOP
FACADES

SCALE	DRAWING NO	REV
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PLO.

SCALE A1 1:100 / A3 1:200

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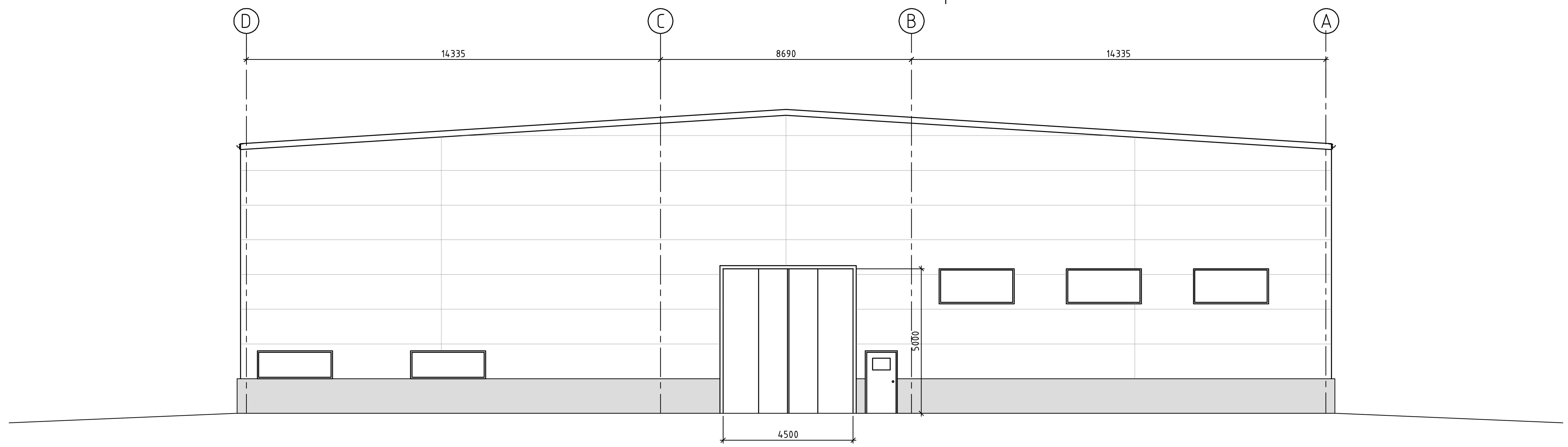

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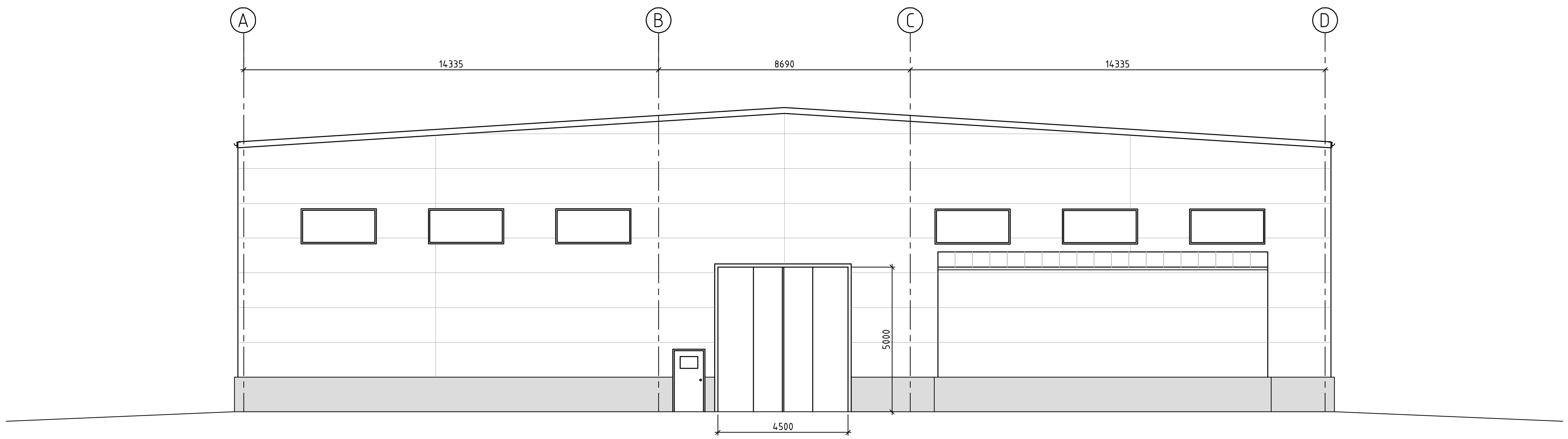

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SCALE A1 1:100 / A3 1:200

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

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DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
VEHICLE WORKSHOP
FACADES

SCALE	DRAWING NO	REV
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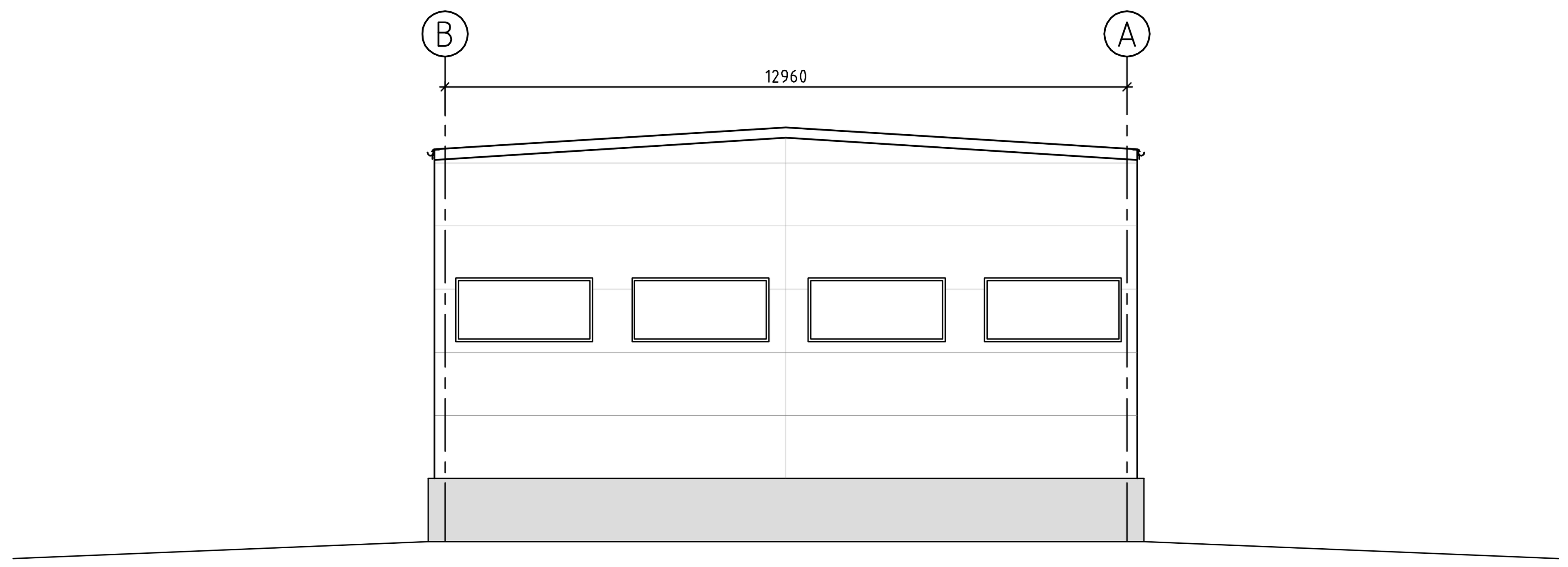

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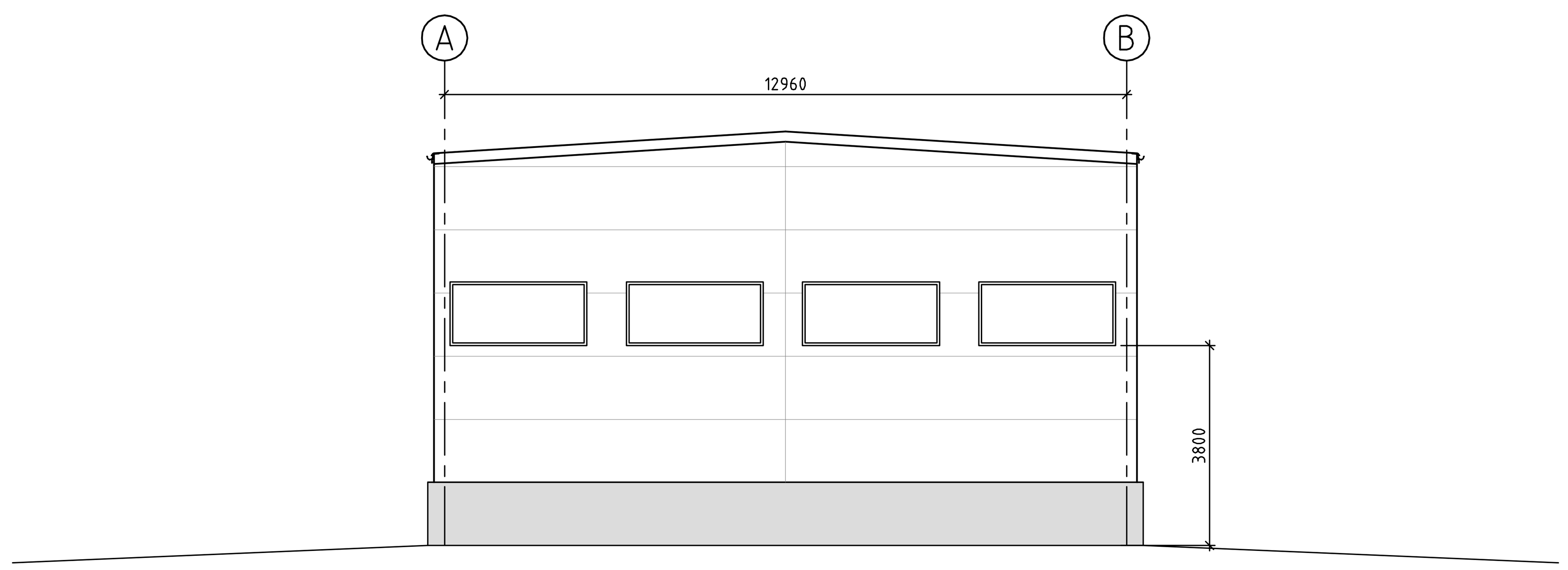

FACADE OF THE WEST
1:100

SCALE A1 1:100 / A3 1:200

2015-03-06

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1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
GENERAL MAINTENANCE WORK SHOP
FACADES

SCALE	DRAWING NO	REV
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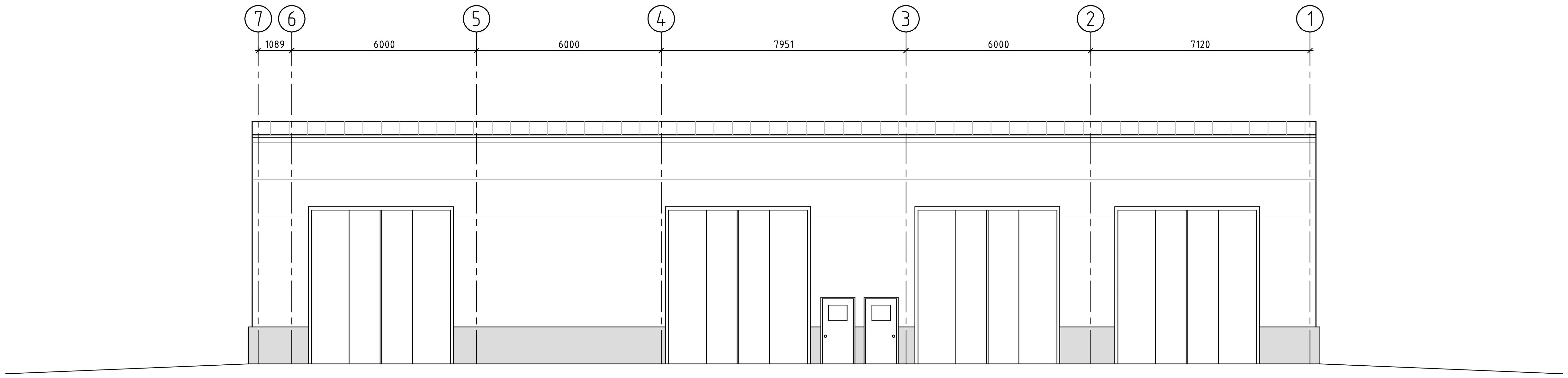

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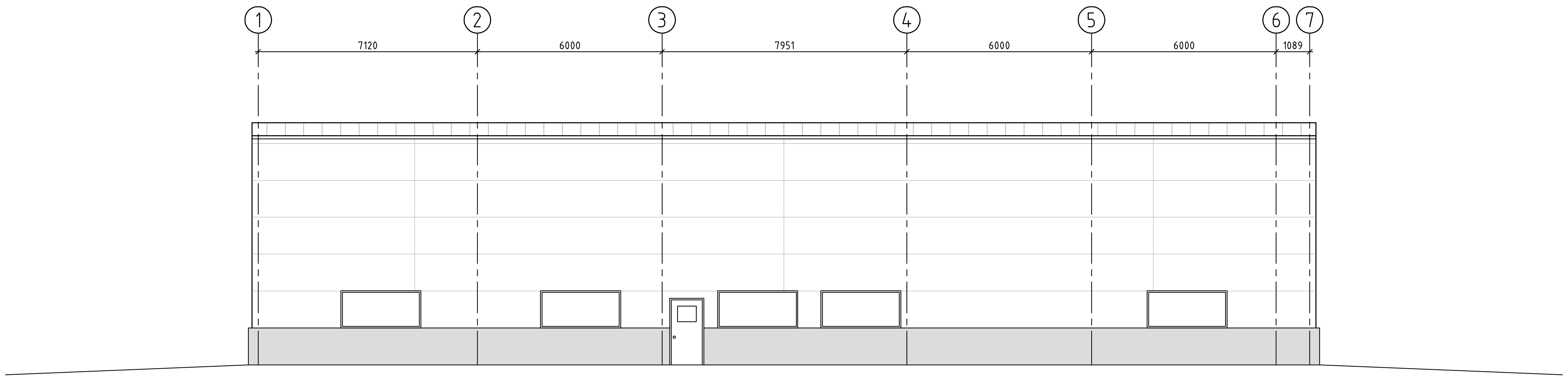

FACADE OF THE SOUTH
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EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

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1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
GENERAL MAINTENANCE WORK SHOP
FACADES

SCALE	DRAWING NO	REV
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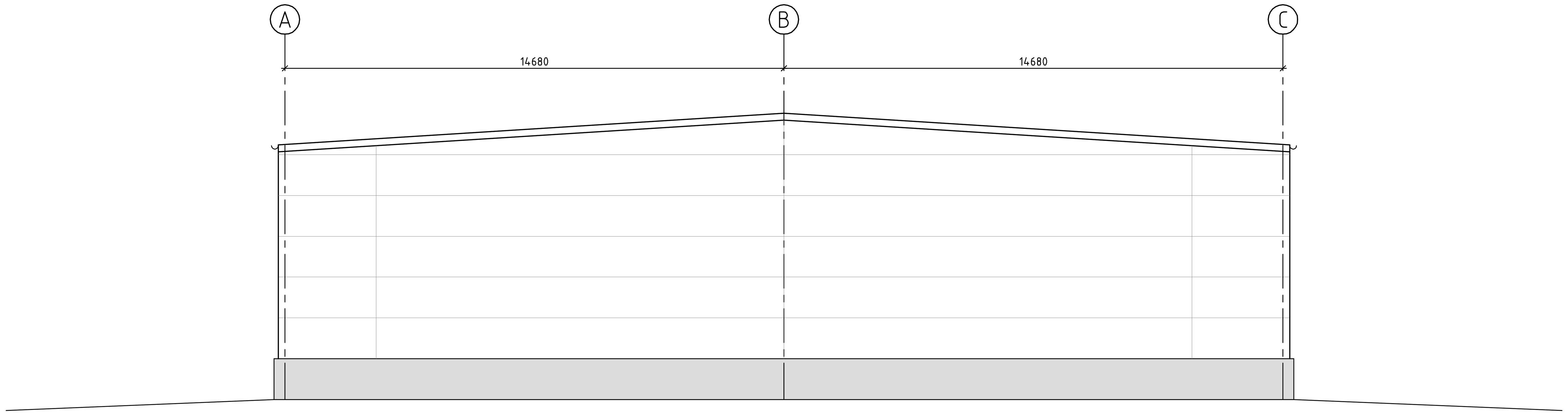
EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.



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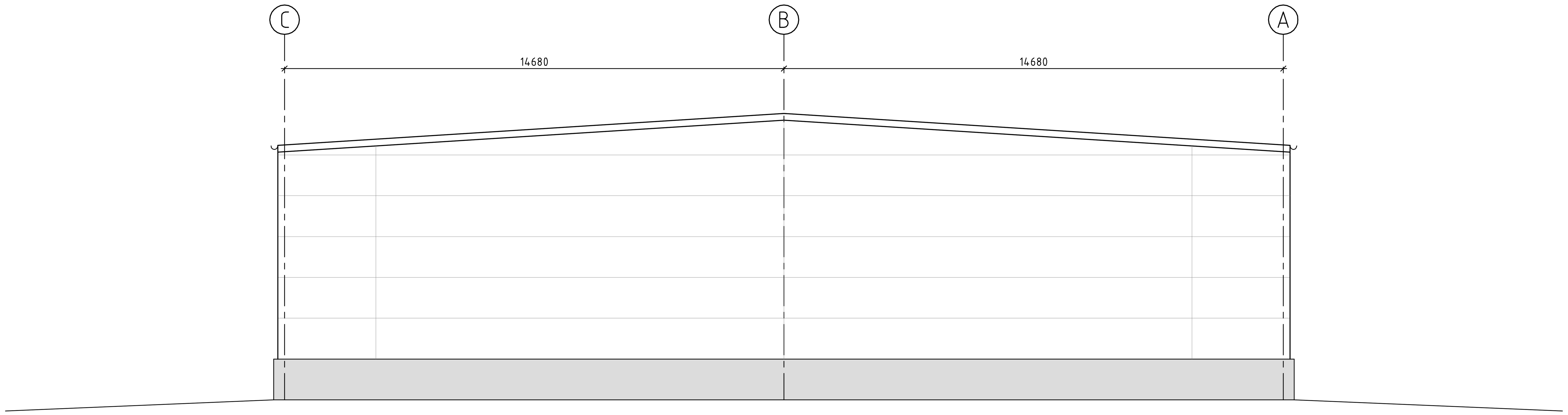

FACADE OF SOUTHEAST
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2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

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1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
STORES
FACADES

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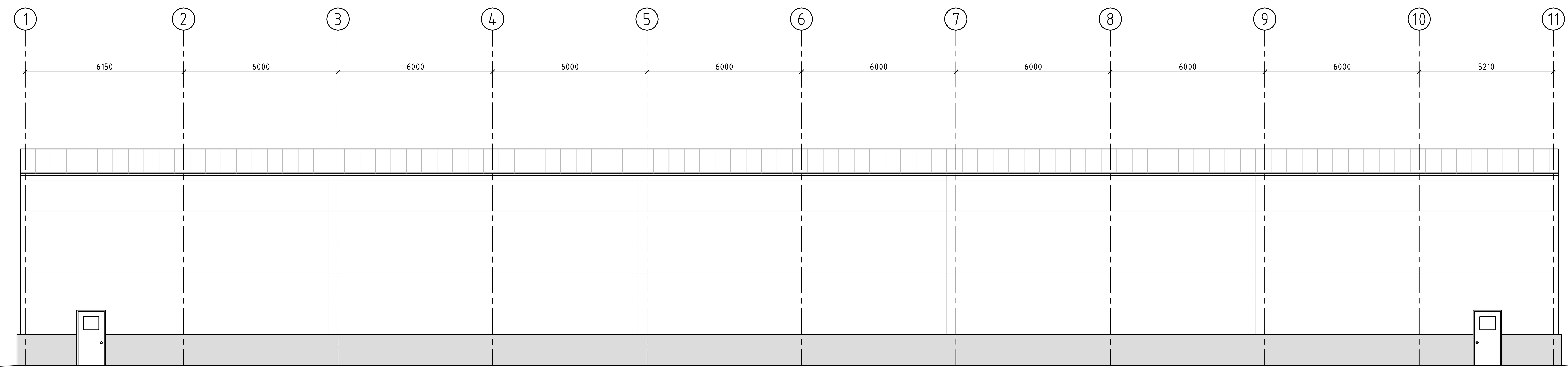

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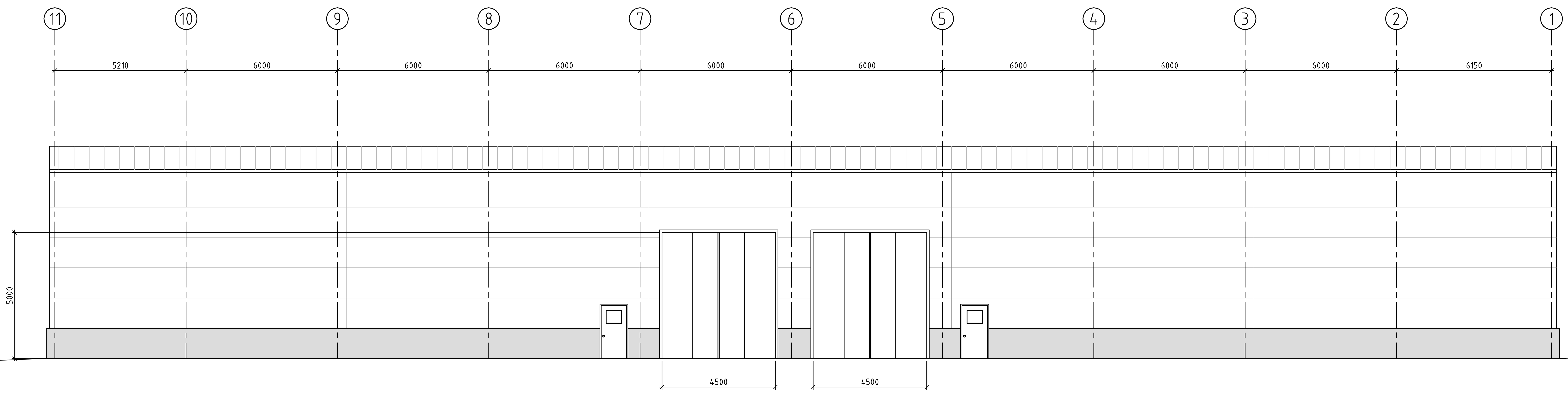

FACADE OF SOUTHWEST
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SCALE A1 1:100 / A3 1:200

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

INSTRUCTIONS

REFERENCES

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

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COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
STORES
FACADES

SCALE	DRAWING NO	REV
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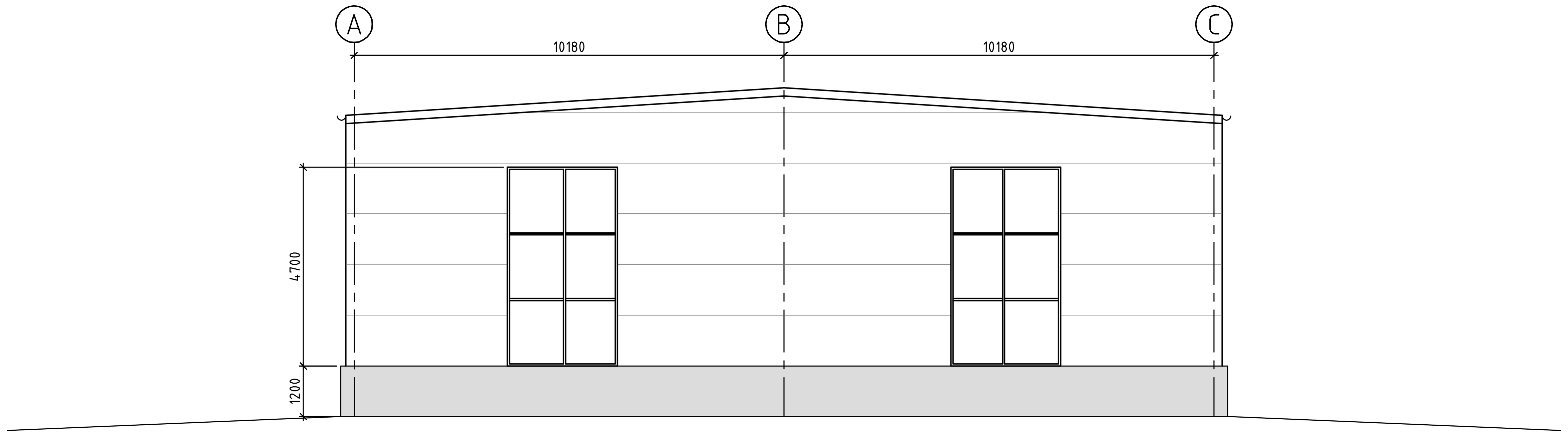

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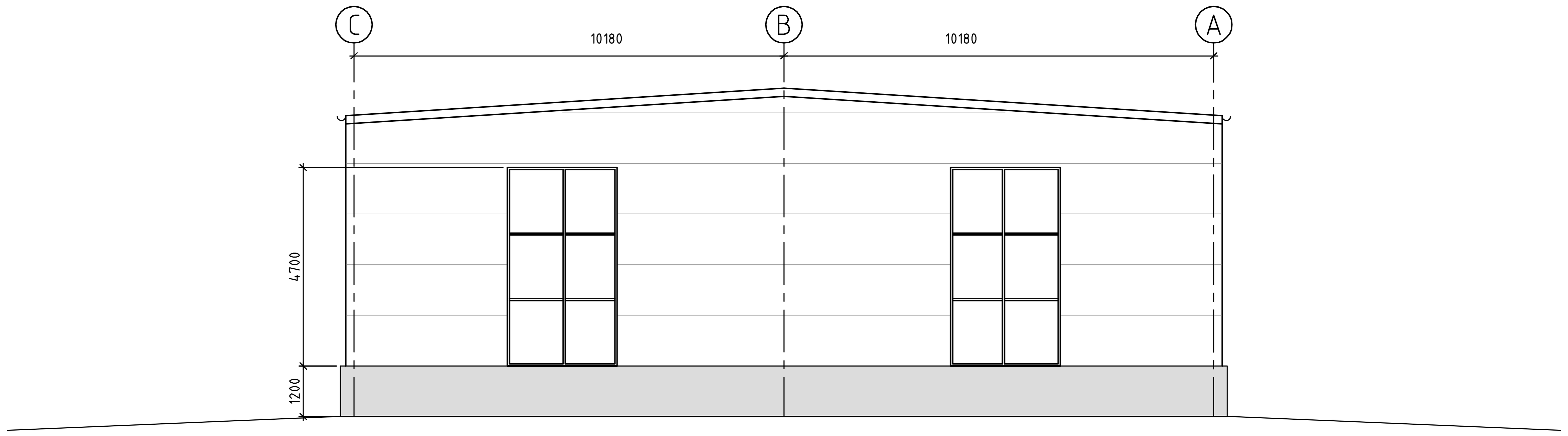

FACADE OF SOUTHEAST
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EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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RAMBOLL

COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

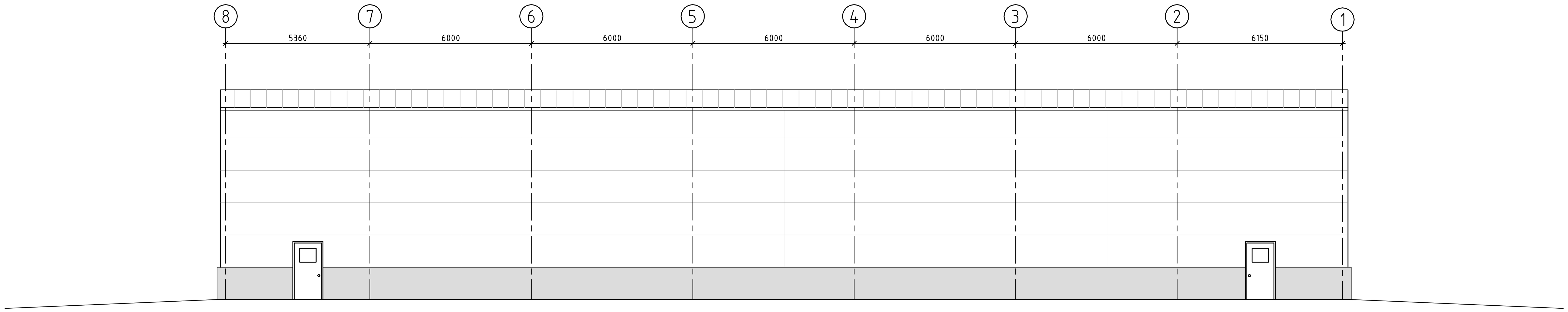
BLÖTBERGET
INDUSTRIAL AREA 1
DRILLCORE ARCHIVE
FACADES

SCALE	DRAWING NO	REV
1:100/1:200	A-40.3-222	

SCALE A1 1:100 / A3 1:200

REF: LAGER: PLO:

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.



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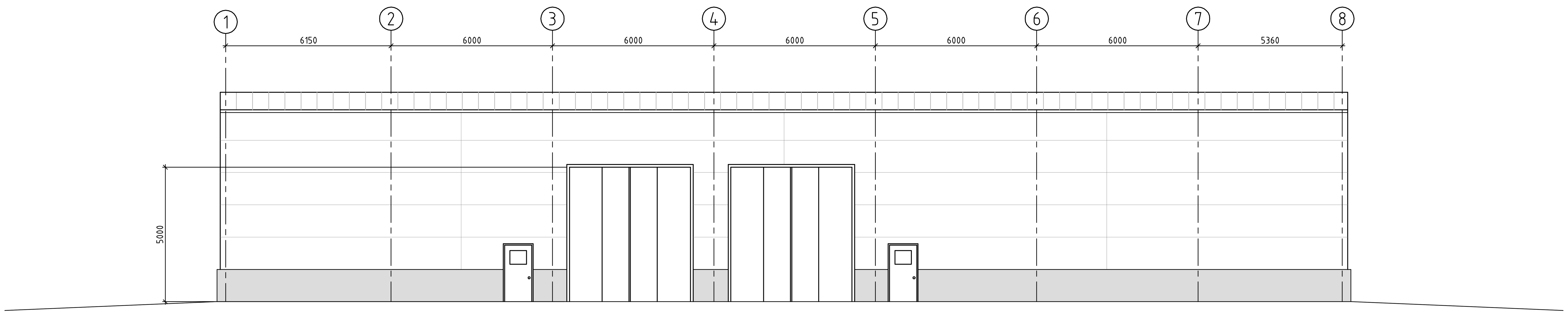

FACADE OF SOUTHWEST
1:100

SCALE A1 1:100 / A3 1:200

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
DRILLCORE ARCHIVE
FACADES

SCALE	DRAWING NO	REV
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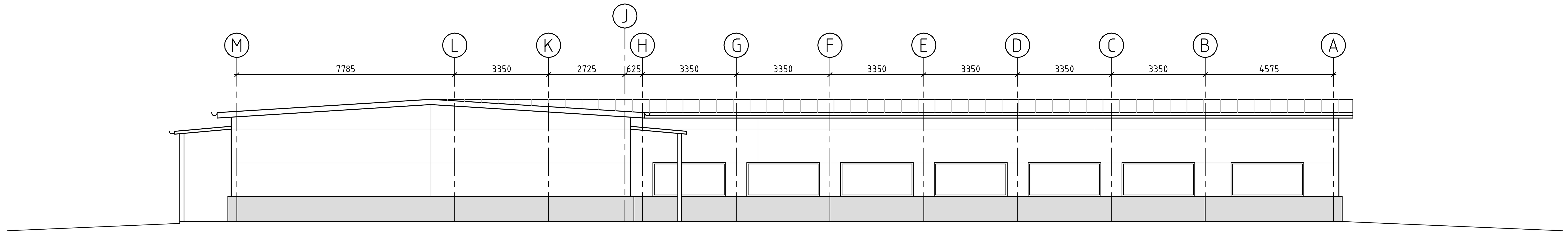

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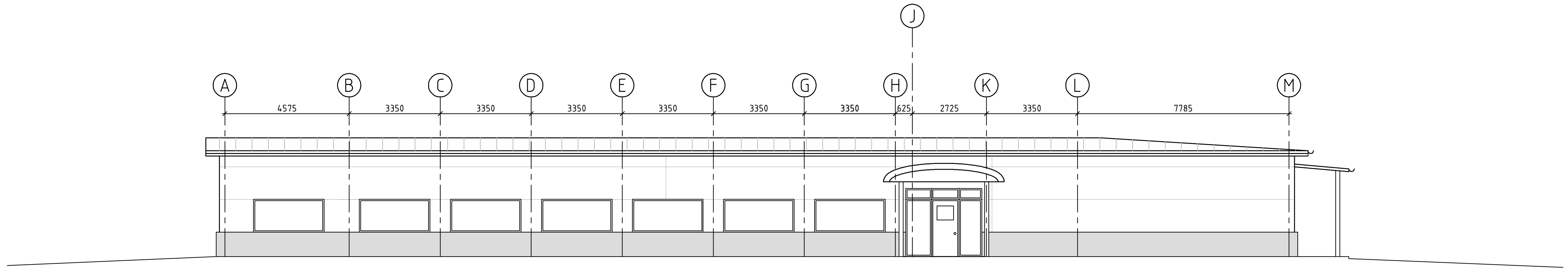

FACADE OF NORTHWEST
1:100

SCALE A1 1:100 / A3 1:200

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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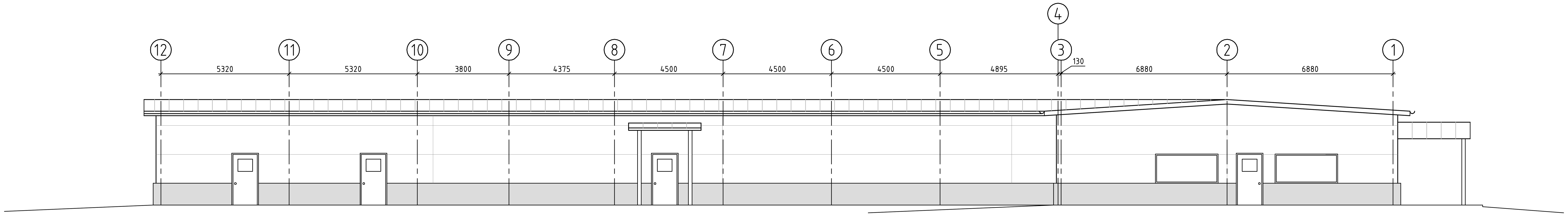
RAMBOLL

COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
OFFICE/LOCKER-ROOM
FACADES

SCALE	DRAWING NO	REV
1:100/1:200	A-40.3-122	

REF.
LAGER
PLO.



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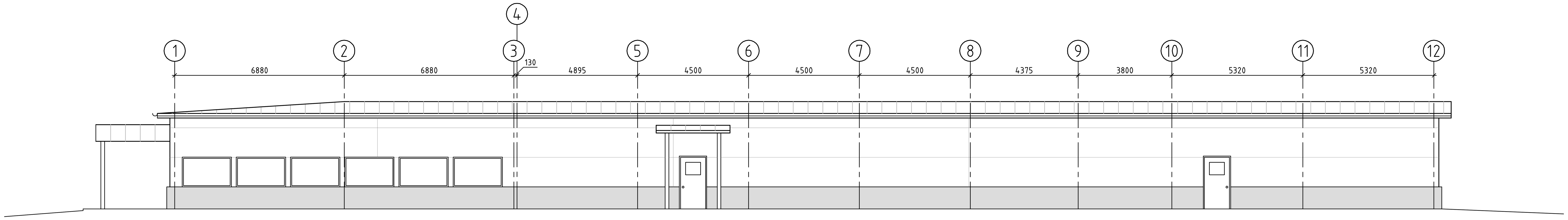

FACADE OF SOUTHWEST
1:100

SCALE A1 1:100 / A3 1:200

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-27
~~2015-03-06~~

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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RAMBOLL

COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

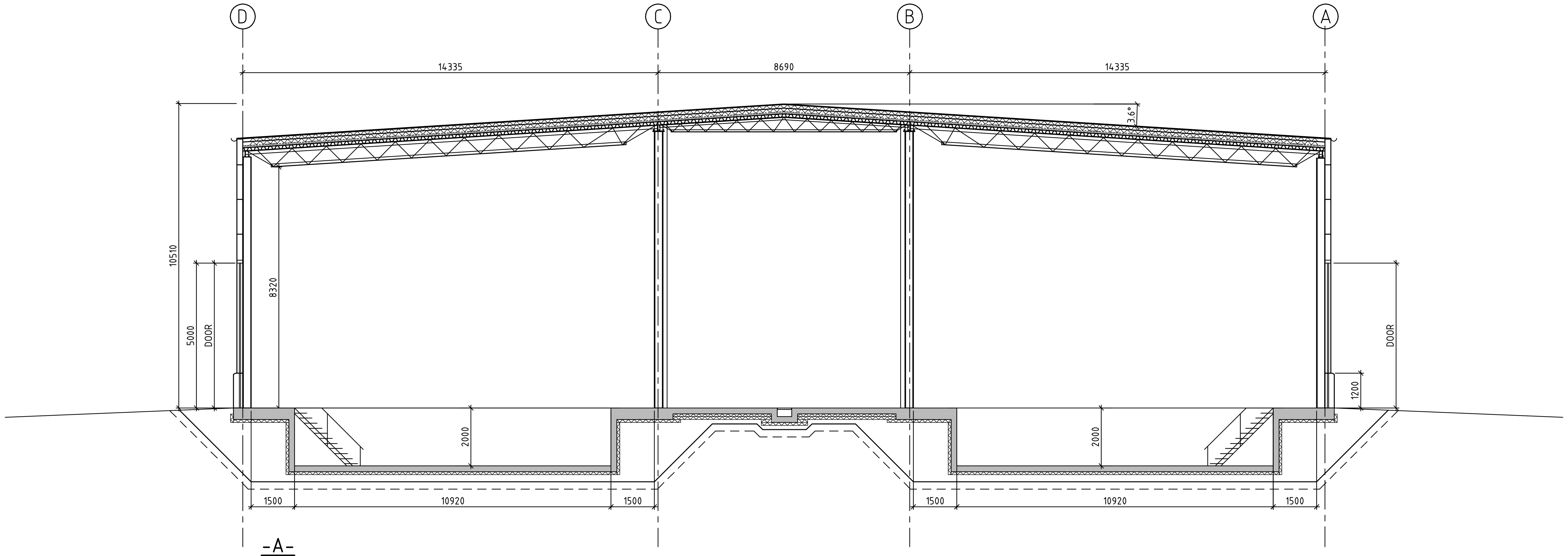
BLÖTBERGET
INDUSTRIAL AREA 1
OFFICE/LOCKER-ROOM
FACADES

SCALE	DRAWING NO	REV
1:100/1:200	A-40.3-121	

PLO:

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.



-A-
1:100

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

LUDVIKA MINES
NORDIC IRON ORE

Ramböll Sverige AB
Pelle Bergs Backe 3
BOX 1932
791 19 FALUN

Phone:010-615 60 00
Fax:010-615 20 00
www.ramboll.se

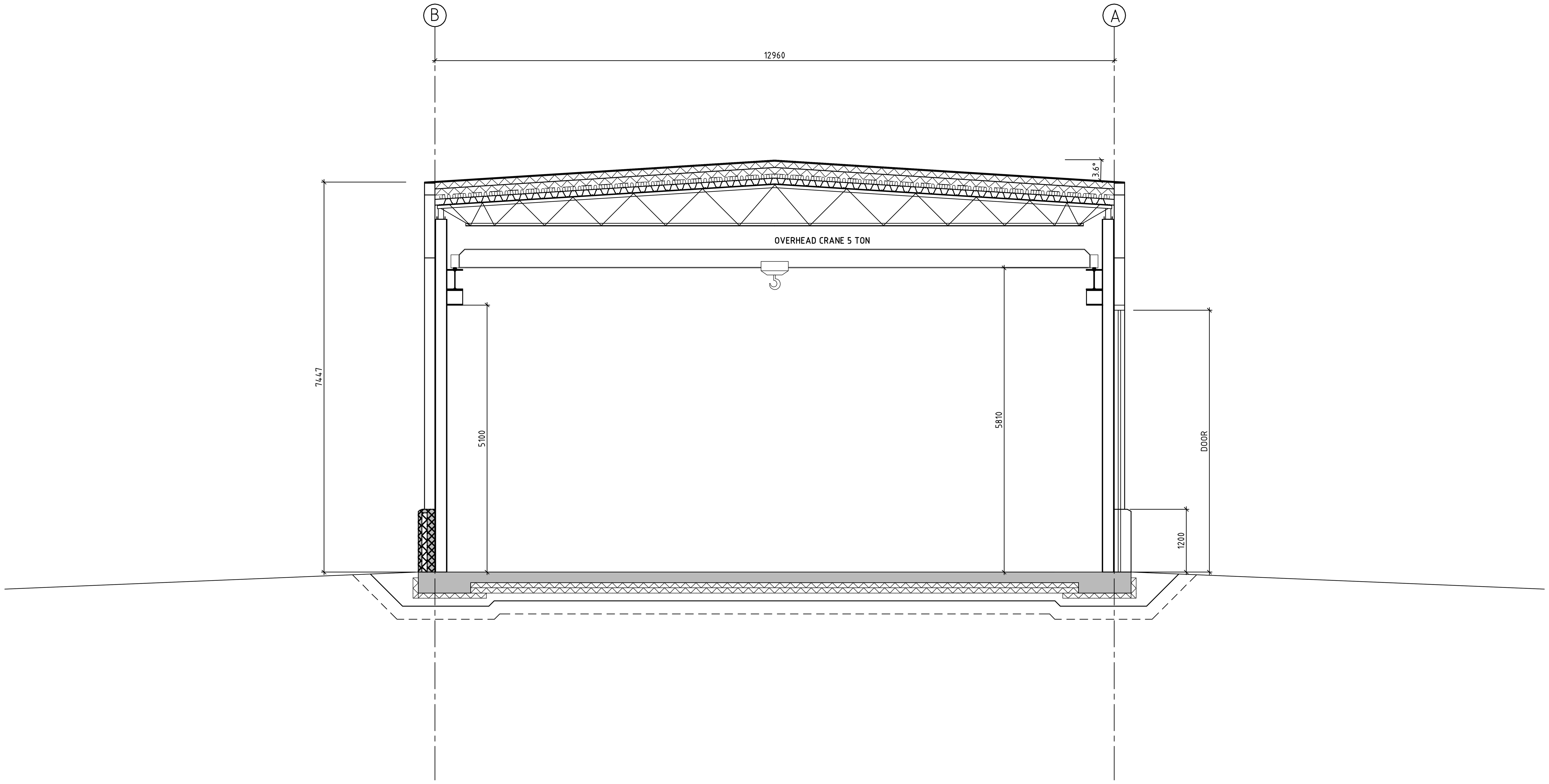
COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
VEHICLE WORKSHOP
SECTION

SCALE	DRAWING NO	REV
1:100/1:200	A-40.2-811	

SCALE A1 1:100 / A3 1:200

REF: LAGER: PLO:



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1:50

SCALE A1 1:50 / A3 1:100

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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Fax:010-615 20 00
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COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
GENERAL MAINTENANCE WORK SHOP
SECTION

SCALE	DRAWING NO	REV
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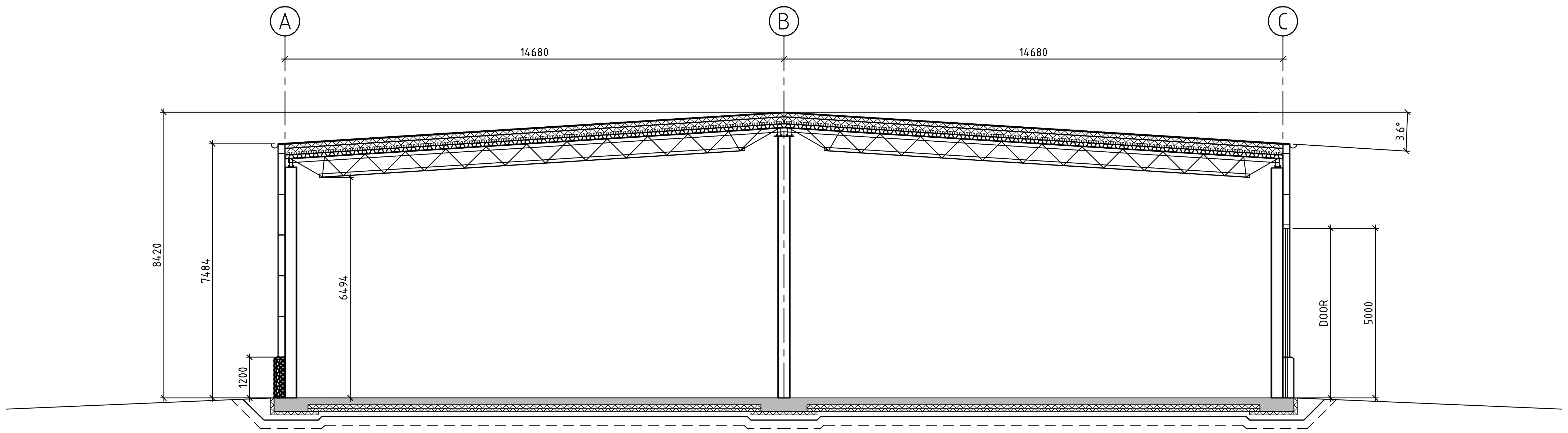

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1:100

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

LUDVIKA MINES
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RAMBOLL

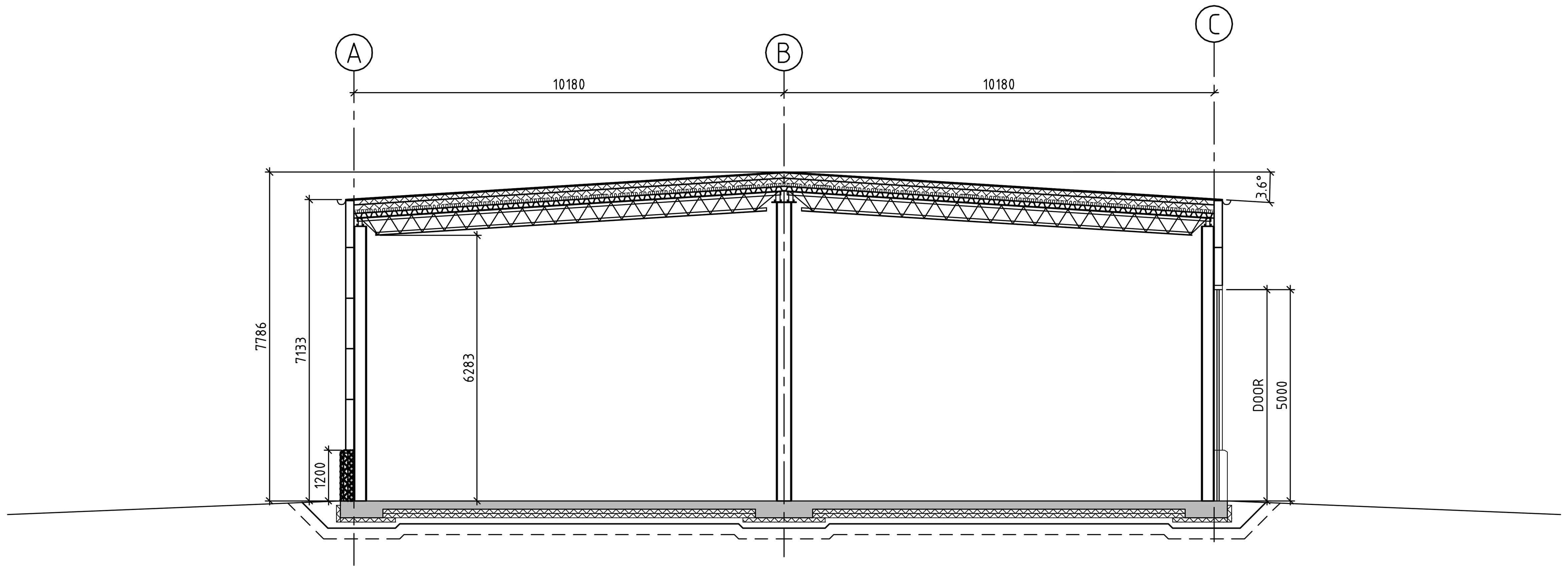
COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
STORES
SECTION

SCALE	DRAWING NO	REV
1:100/1:200	A-40.2-311	

SCALE A1 1:100 / A3 1:200

REF.
LAGER:
PLO:



A-
1:100

SCALE A1 1:100 / A3 1:200

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

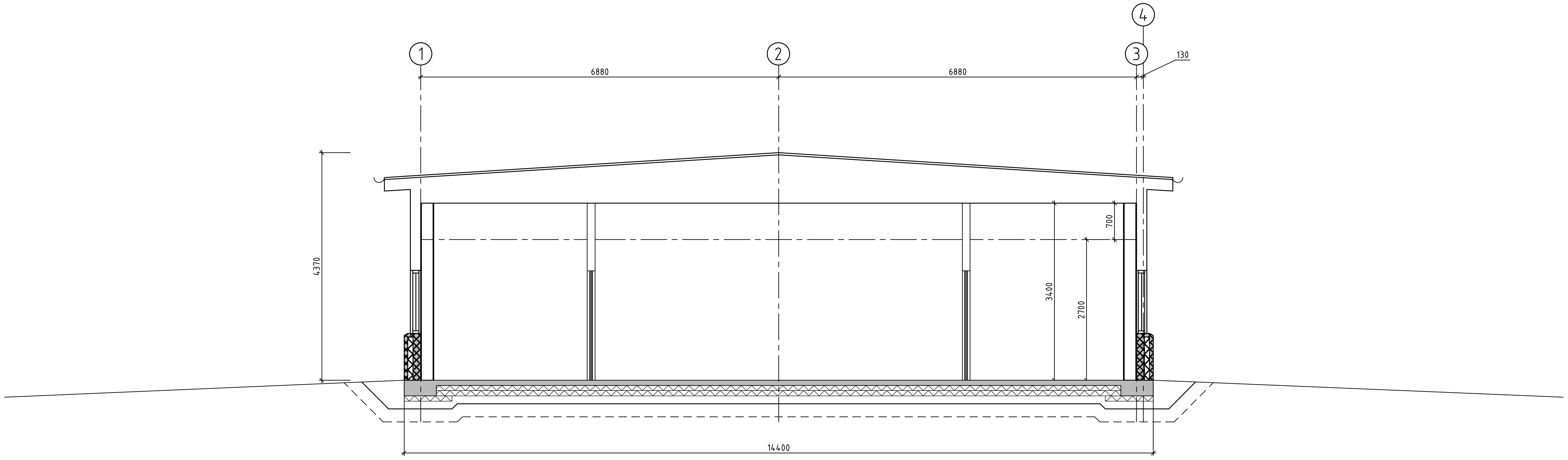
BLÖTBERGET
INDUSTRIAL AREA 1
DRILLCORE ARCHIVE
SECTION

SCALE	DRAWING NO	REV
1:100/1:200	A-40.2-211	

REF:
LAGER:
PLO:

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.



-A-
1:50

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
OFFICE/LOCKER-ROOM
SECTION

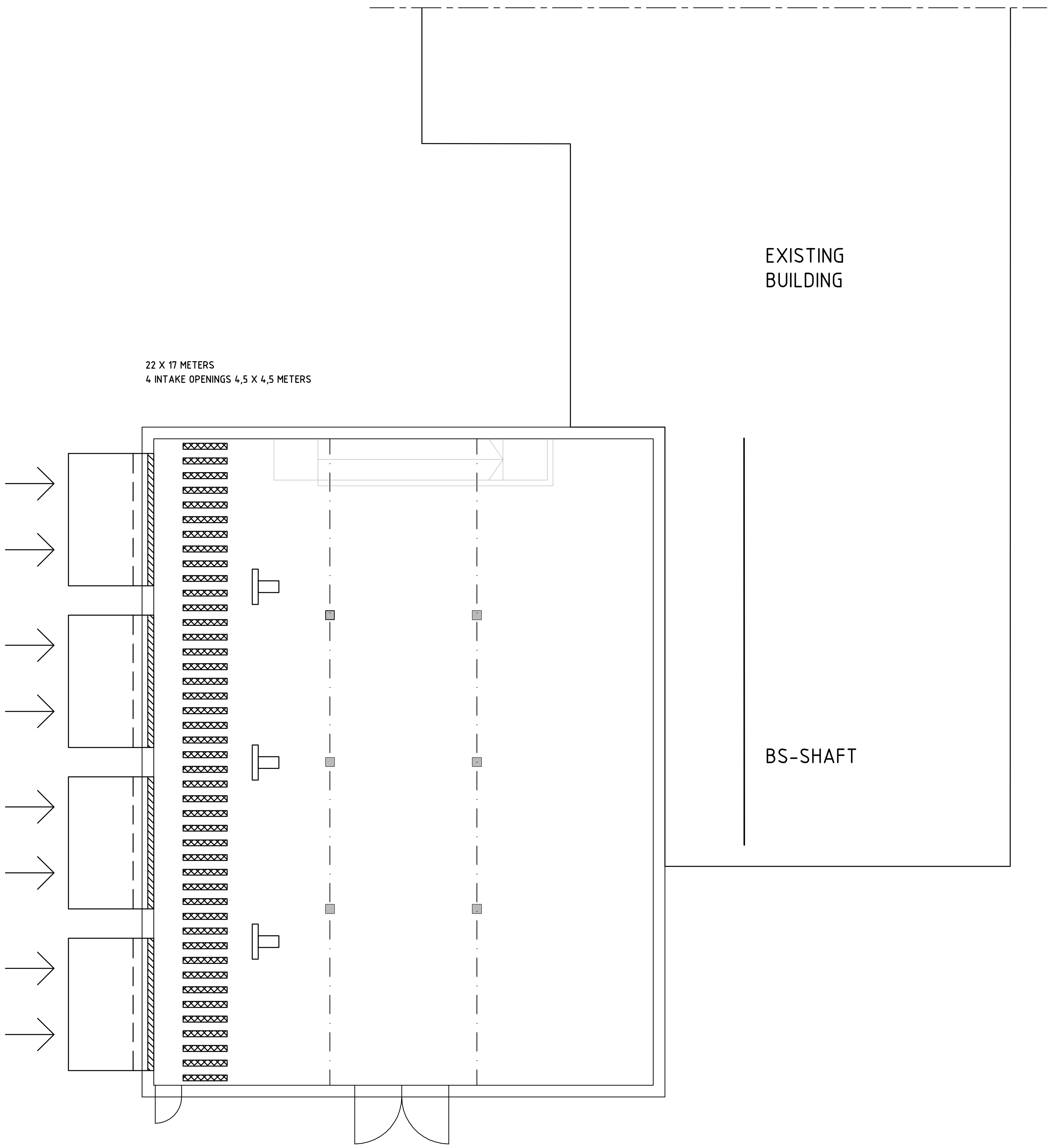
SCALE	DRAWING NO	REV
1:50/1:100	A-40.2-111	

PLO:

REF:
LAGER:

0 1 2 3 4 5 10

SCALE A1 1:50 / A3 1:100



SUPPLY AIR BUILDING BS SHAFT
1:100

SCALE A1 1:100 / A3 1:200

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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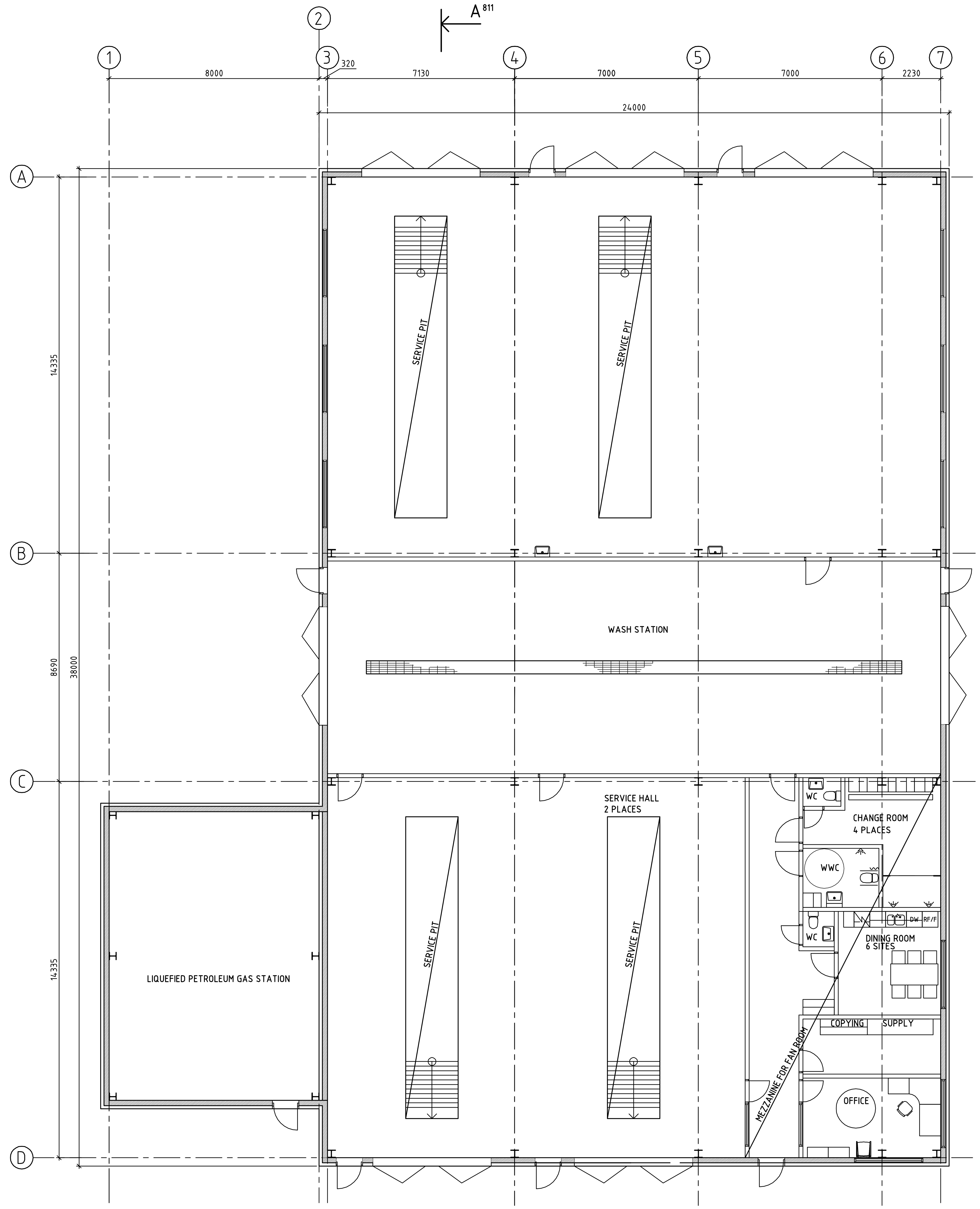
COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
BS SHAFT
SUPPLY AIR BUILDING
PLAN

SCALE	DRAWING NO	REV
1:100/1:200	A-40.1-901	



VEHICLE WORKSHOP, PLAN
1:100

SCALE A1 1:100 / A3 1:200

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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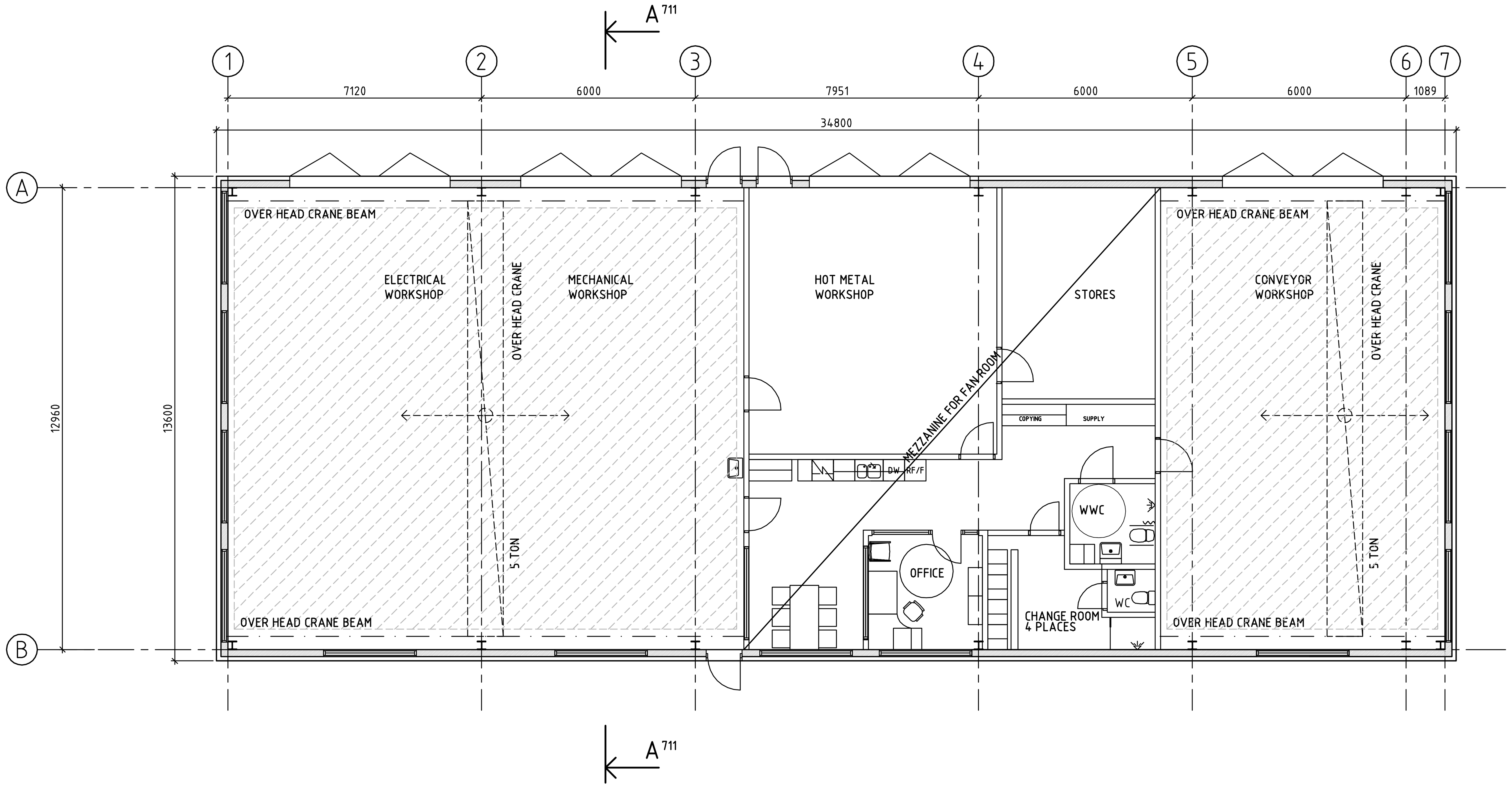
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
VEHICLE WORKSHOP
PLAN

SCALE	DRAWING NO	REV
1:100/1:200	A-40.1-801	



GENERAL MAINTENANCE WORK SHOP, PLAN
1:100

EXPLANATIONS
ALL MEASUREMENTS ARE IN MILLIMETERS.

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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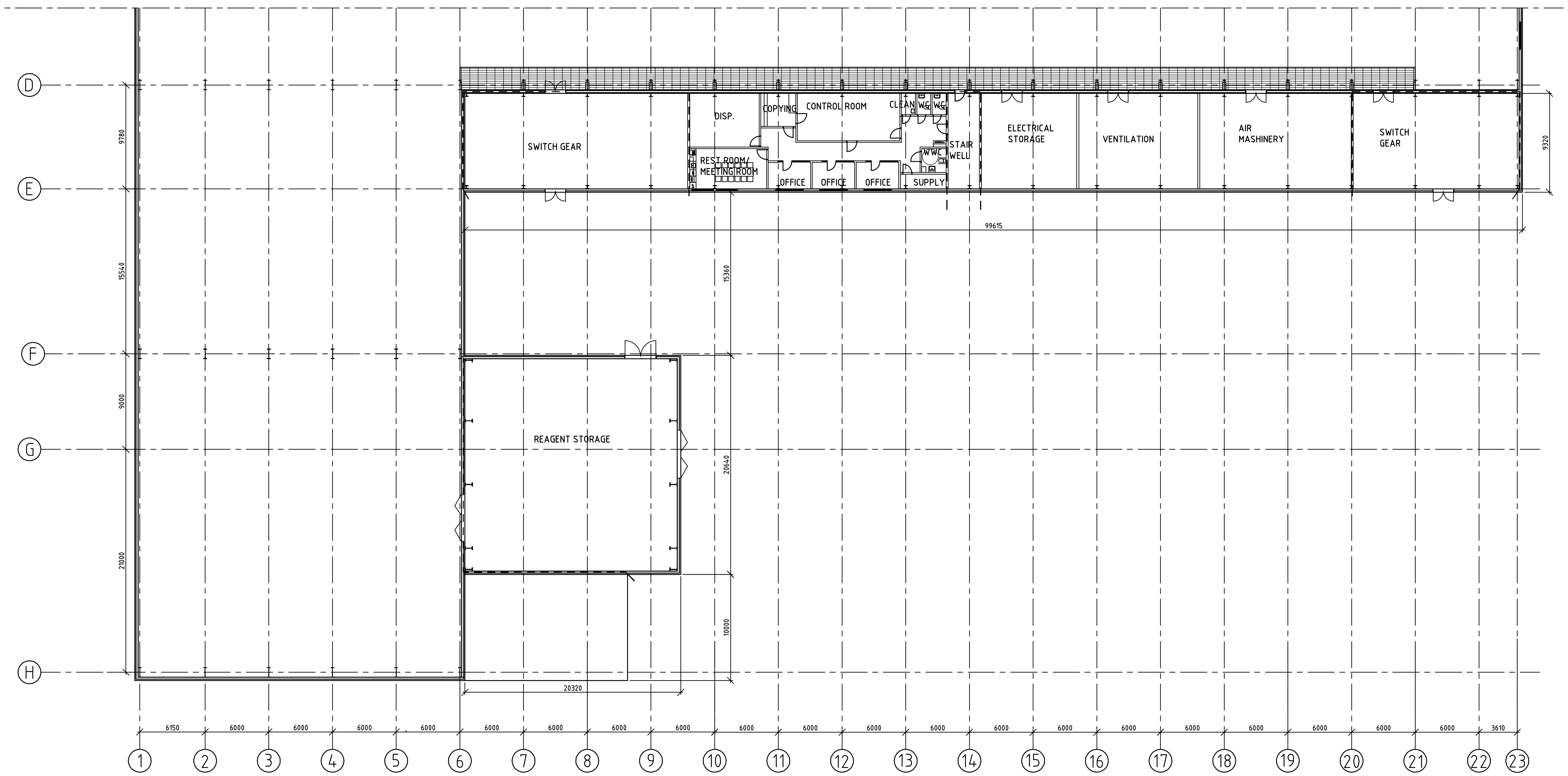
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
GENERAL MAINTENANCE WORK SHOP
PLAN

SCALE	DRAWING NO	REV
1:100/1:200	A-40.1-701	

SCALE A1 1:100 / A3 1:200

REF:
LAGER:
PLO:



PLAN 2
1:250

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.

FIRE WALLS (EI60)

GRATING FLOOR

2015-03-27
~~2015-03-06~~

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

LUDVIKA MINES
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COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
PROCESS PLANT
PLAN 2

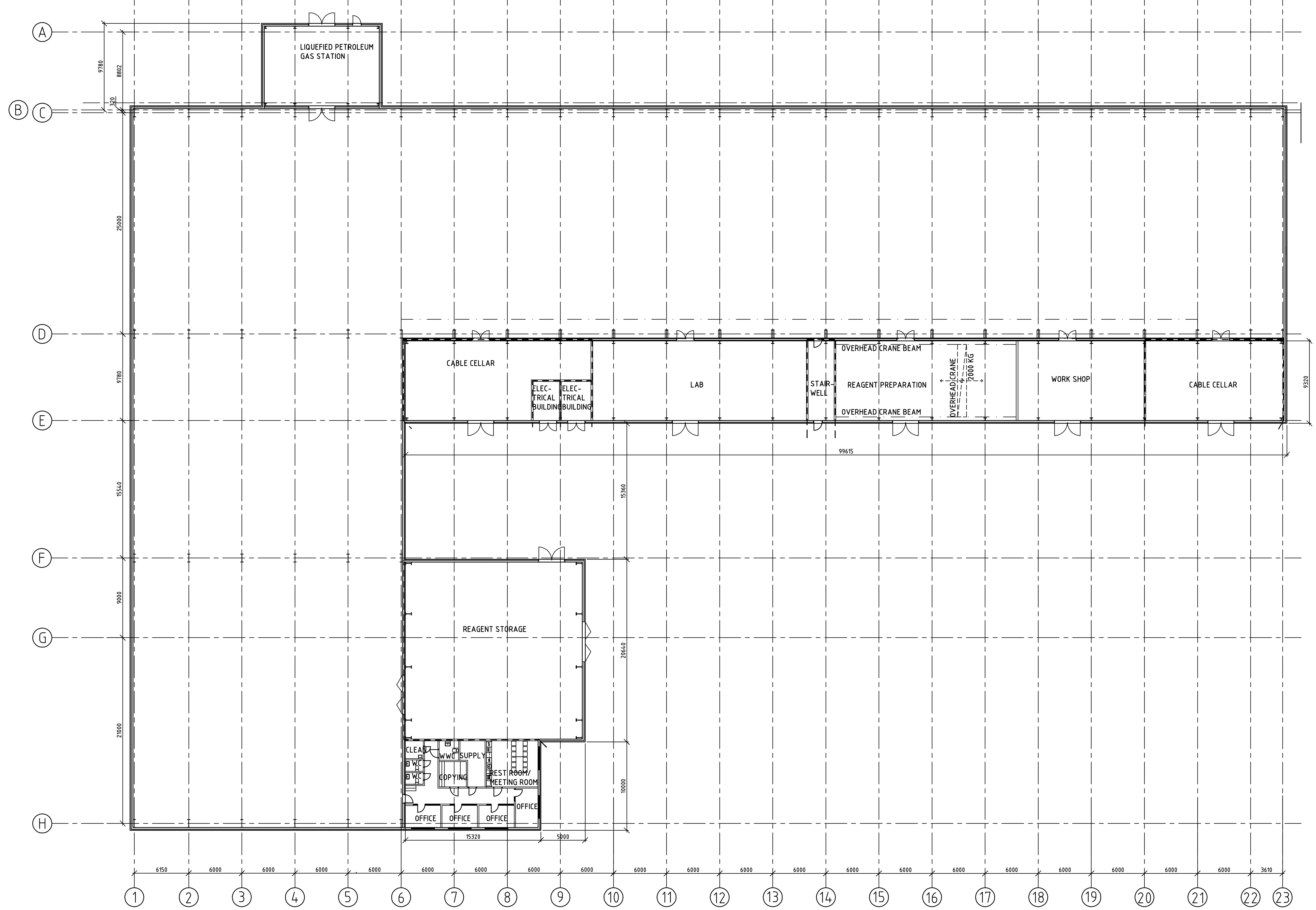
SCALE	DRAWING NO	REV
1:250/1:500	A-40.1-502	

SCALE A1 1:250 / A3 1:500

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.

FIRE WALLS (EI60)



PLAN 1
1:250

SCALE A1 1:250 / A3 1:500

2015-03-27

~~2015-03-06~~

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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RAMBOLL

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1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
PROCESS PLANT
PLAN 1

SCALE	DRAWING NO	REV
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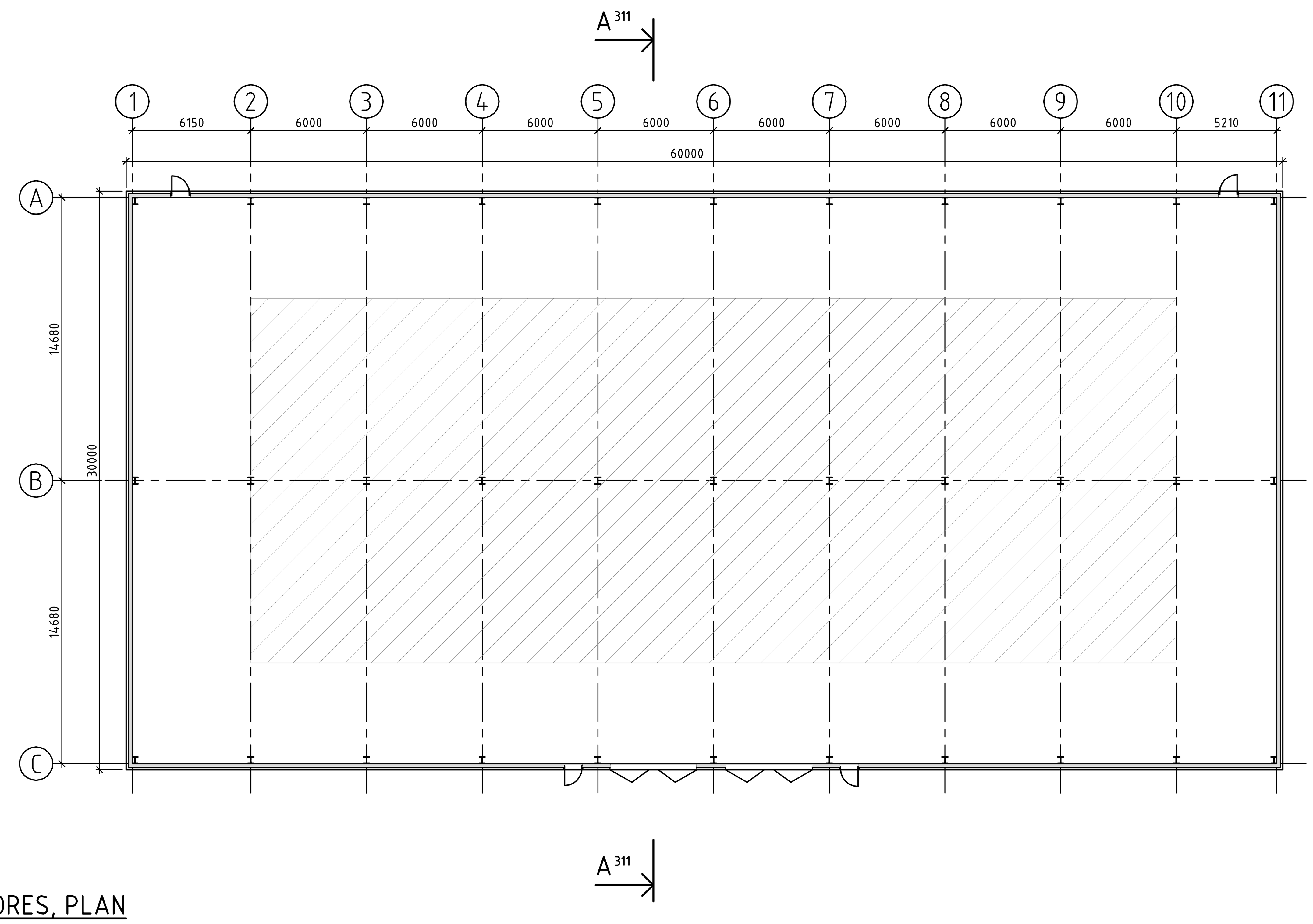

STORES, PLAN
1:200

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.

AREA FOR STORAGE RACK
AREA 907 m2

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

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NORDIC IRON ORE

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Fax:010-615 20 00
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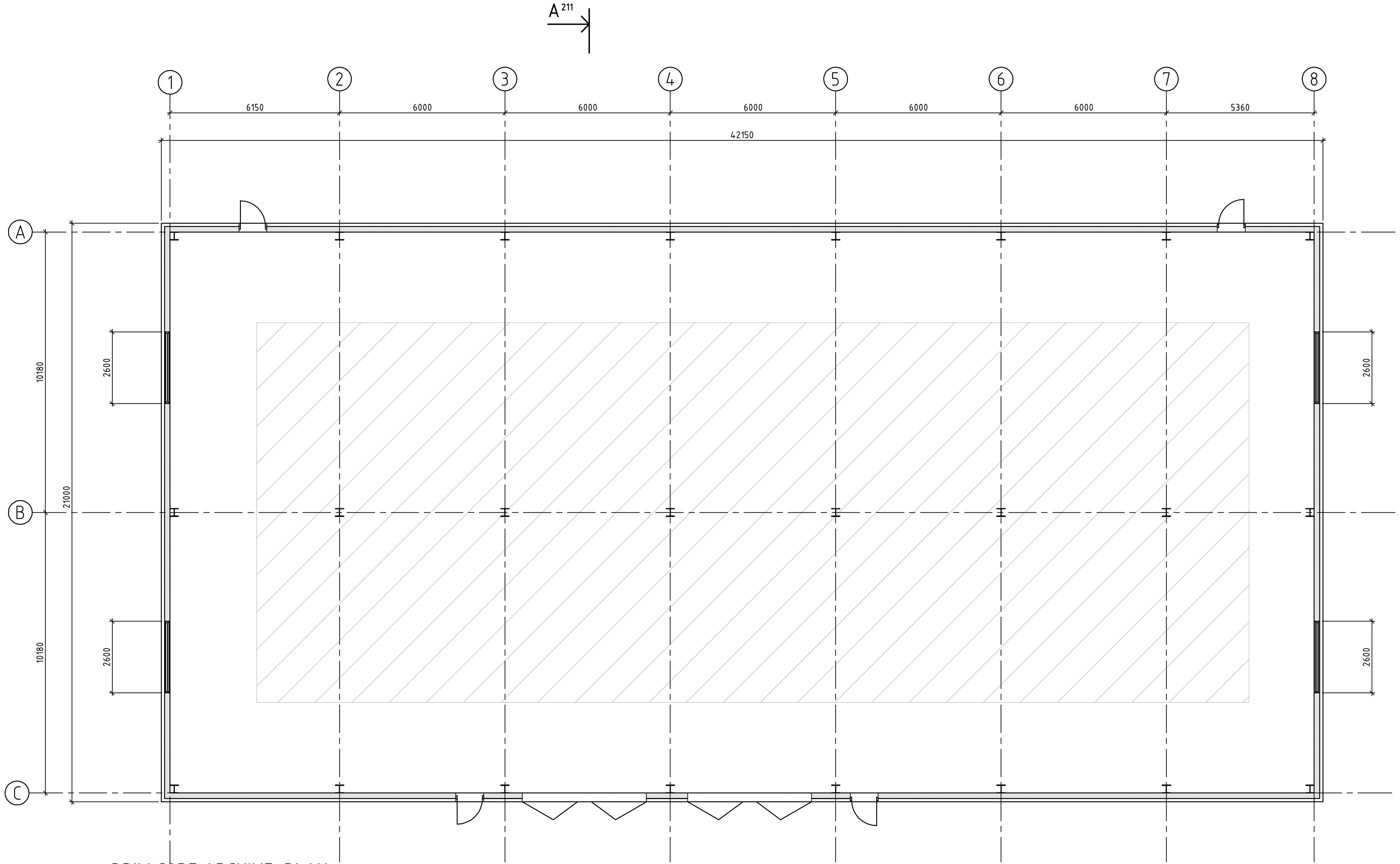
COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
STORES
PLAN

SCALE	DRAWING NO	REV
1:200/1:400	A-40.1-301	

SCALE A1 1:200 / A3 1:400

REF: LAGER: PLO:



DRILLCORE ARCHIVE, PLAN
1:100

SCALE A1 1:100 / A3 1:200

EXPLANATIONS

ALL MEASUREMENTS ARE IN MILLIMETERS.

AREA FOR STORAGE RACK
AREA 496 m2

2015-03-06

REV	ANT	CHANGE INTENDS	DATE	SIGN

PRELIMINARY DESIGN

LUDVIKA MINES
NORDIC IRON ORE

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RAMBOLL

COMMISION NO	DRAWN/CONSTRUCTED BY	ADMINISTRATOR
1320011785	KS	HL
DATE	LIABLE	

BLÖTBERGET
INDUSTRIAL AREA 1
DRILLCORE ARCHIVE
PLAN

SCALE	DRAWING NO	REV
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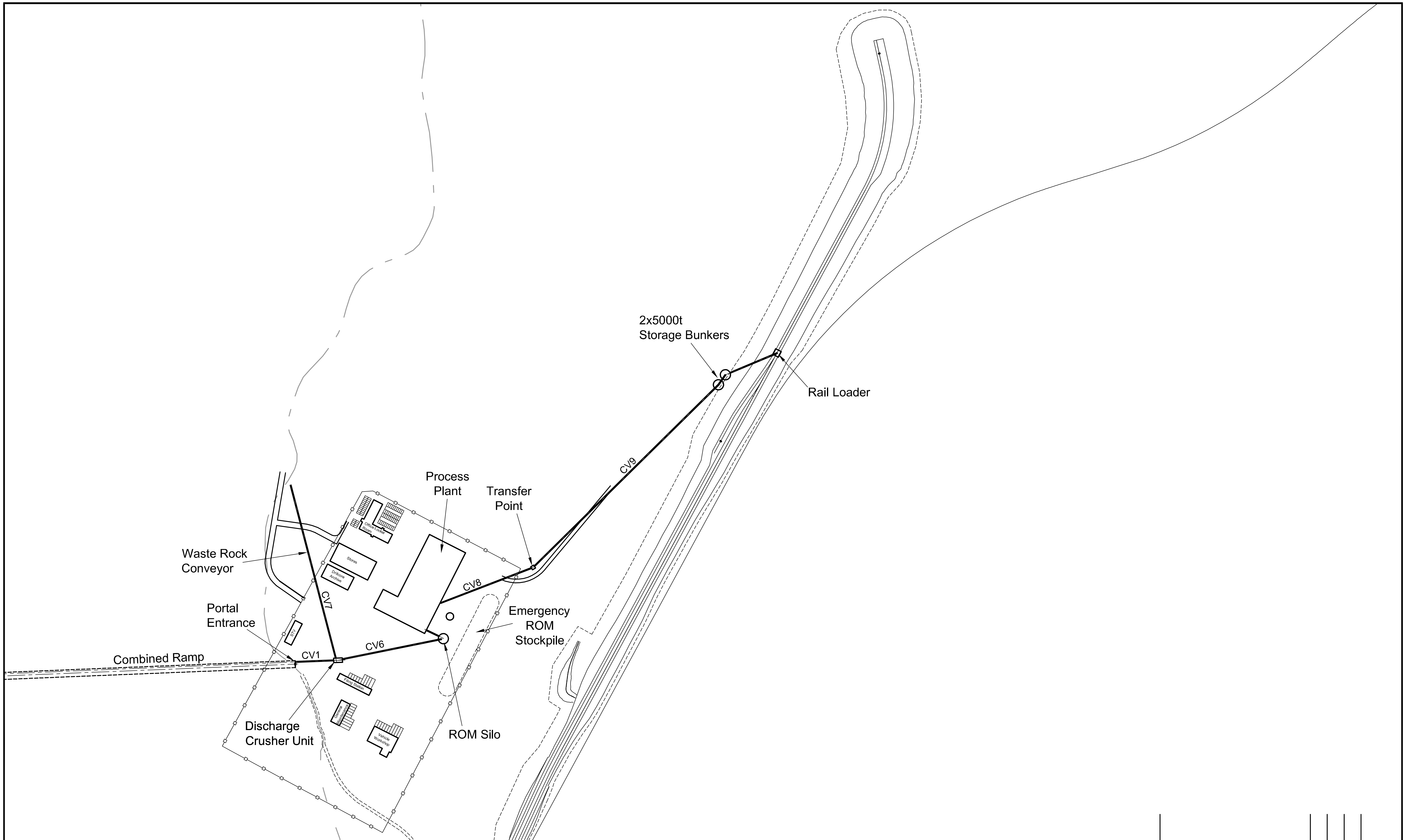

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e-mail: uk@dmt-group.com

Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

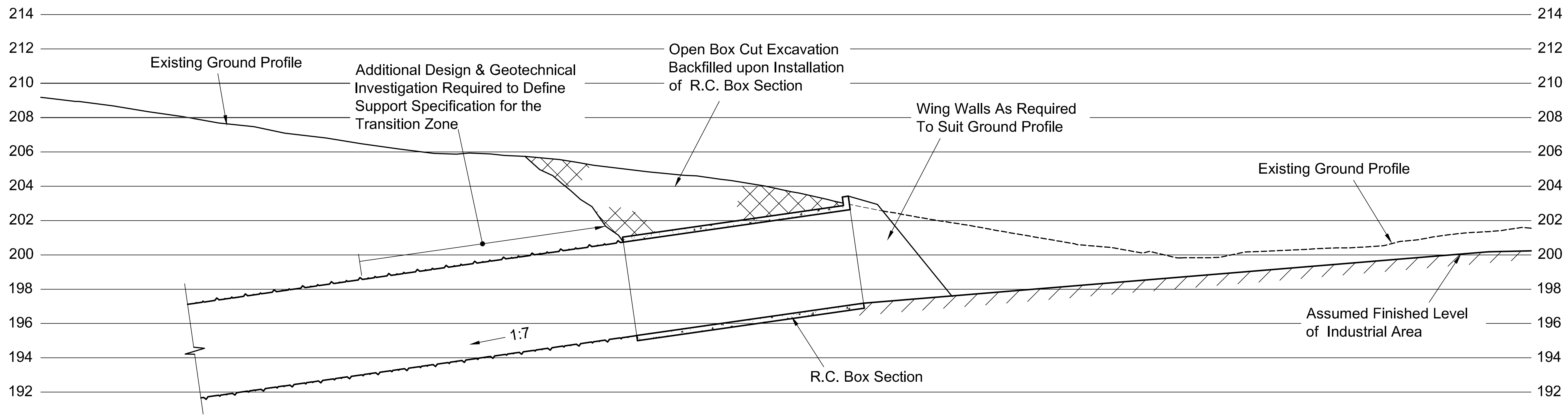
Drawing Title:

Industrial Area 1 Showing Conveyor and Rail Loading

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
1:5000	RS	Nov 14	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	020	



Section

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DMT

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Earth. Insight. Values

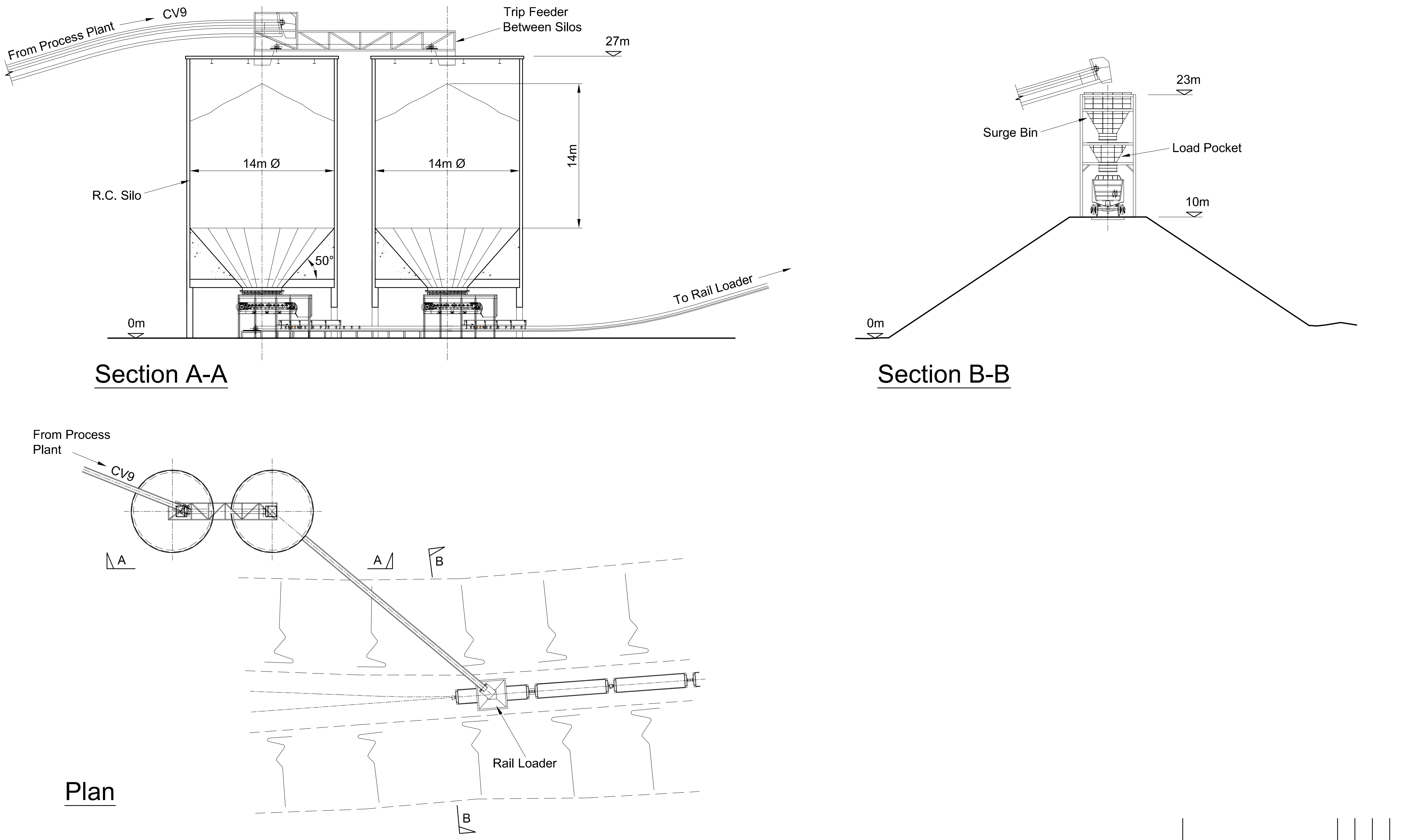
Project: Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Conceptual Design of Portal Entrance of Combined Ramp From Industrial Area 1

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
1:200	RS	April 14	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	018	



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Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

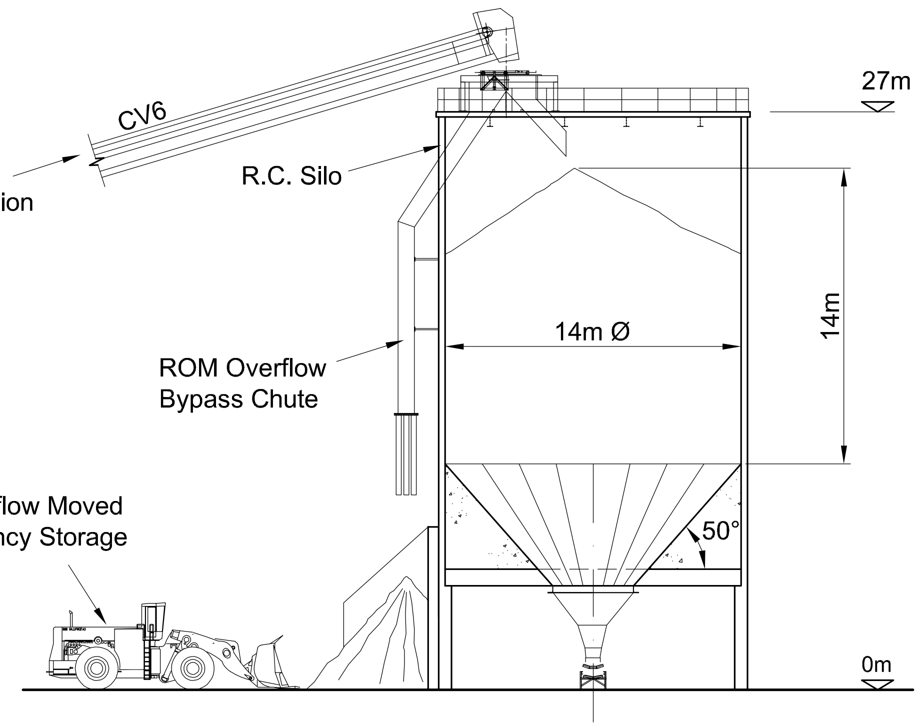
Drawing Title:

G.A. of Load Out System To Railway

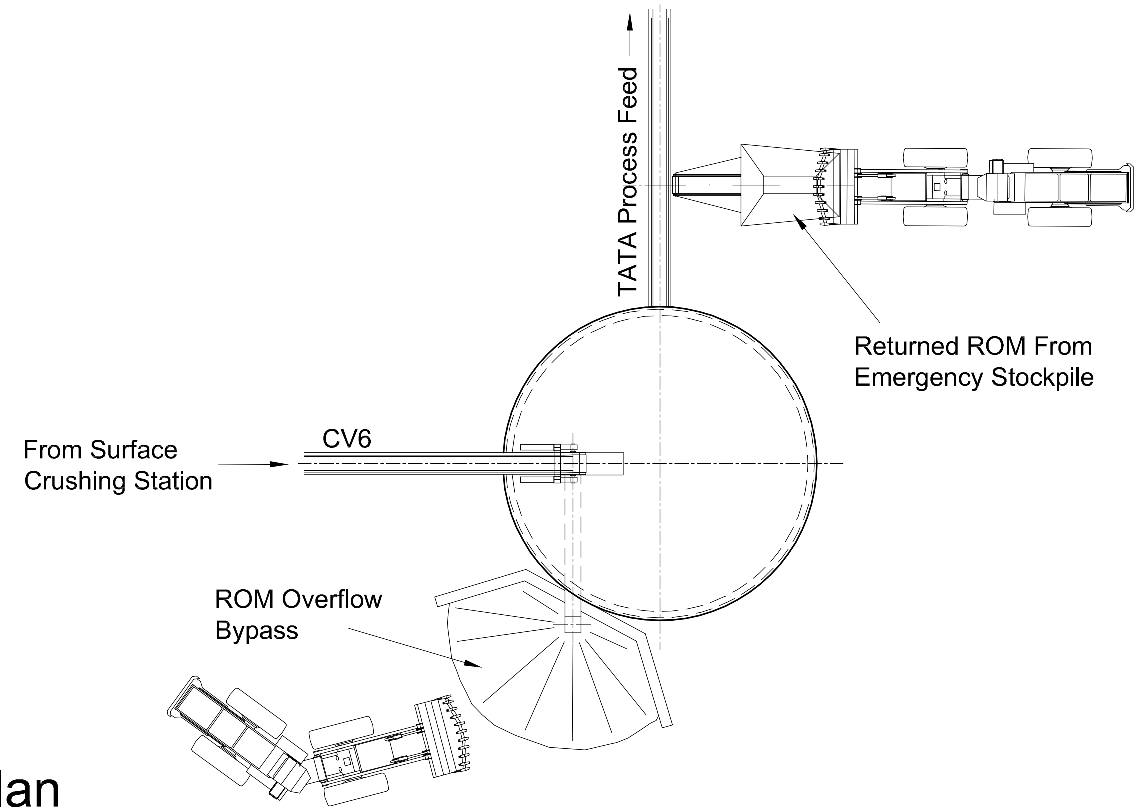
Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	April 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	007	



Elevation



Plan

Pure Offices,
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Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

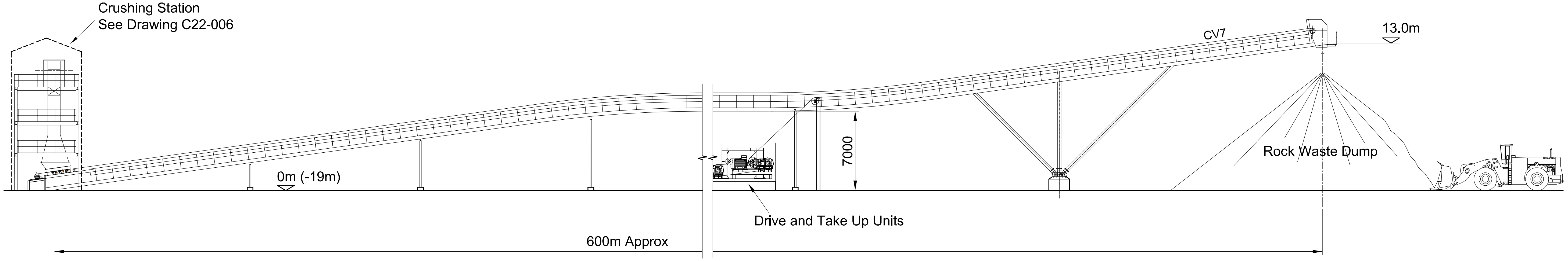
Drawing Title:

G.A. of ROM Silo with Emergency Overflow

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	March 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	006	



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United Kingdom
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Earth. Insight. Values

Project:
Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Elevation of Waste Rock Conveyor (CV7) From Surface Crusher

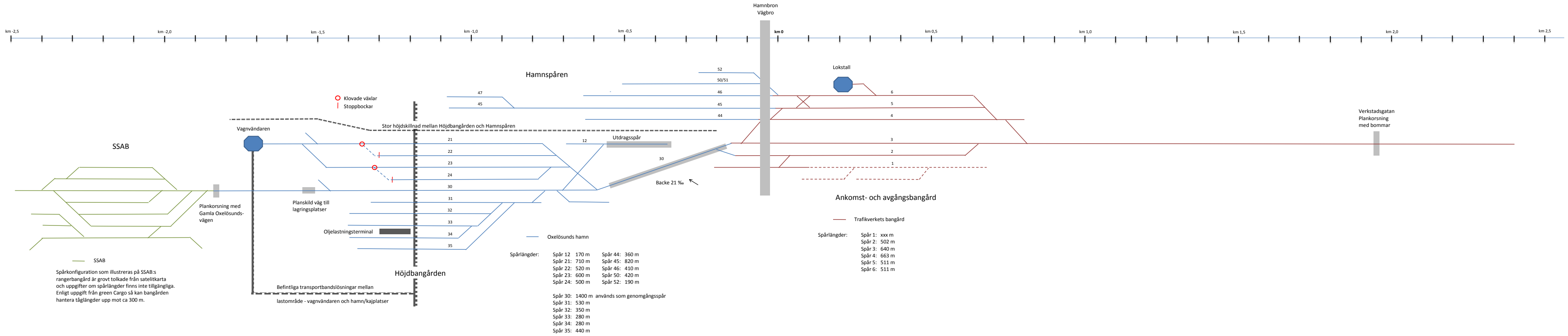
Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	March 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	005	

OXELÖSUNDS BANGÅRDAR Rev. 2012-05-23



KONTRAKT NR	
RITAD	Leif Lidevad
GODKÄND	
GODKÄND AV	
SKAPAD	2008-03-10

Oxelösunds Hamn AB

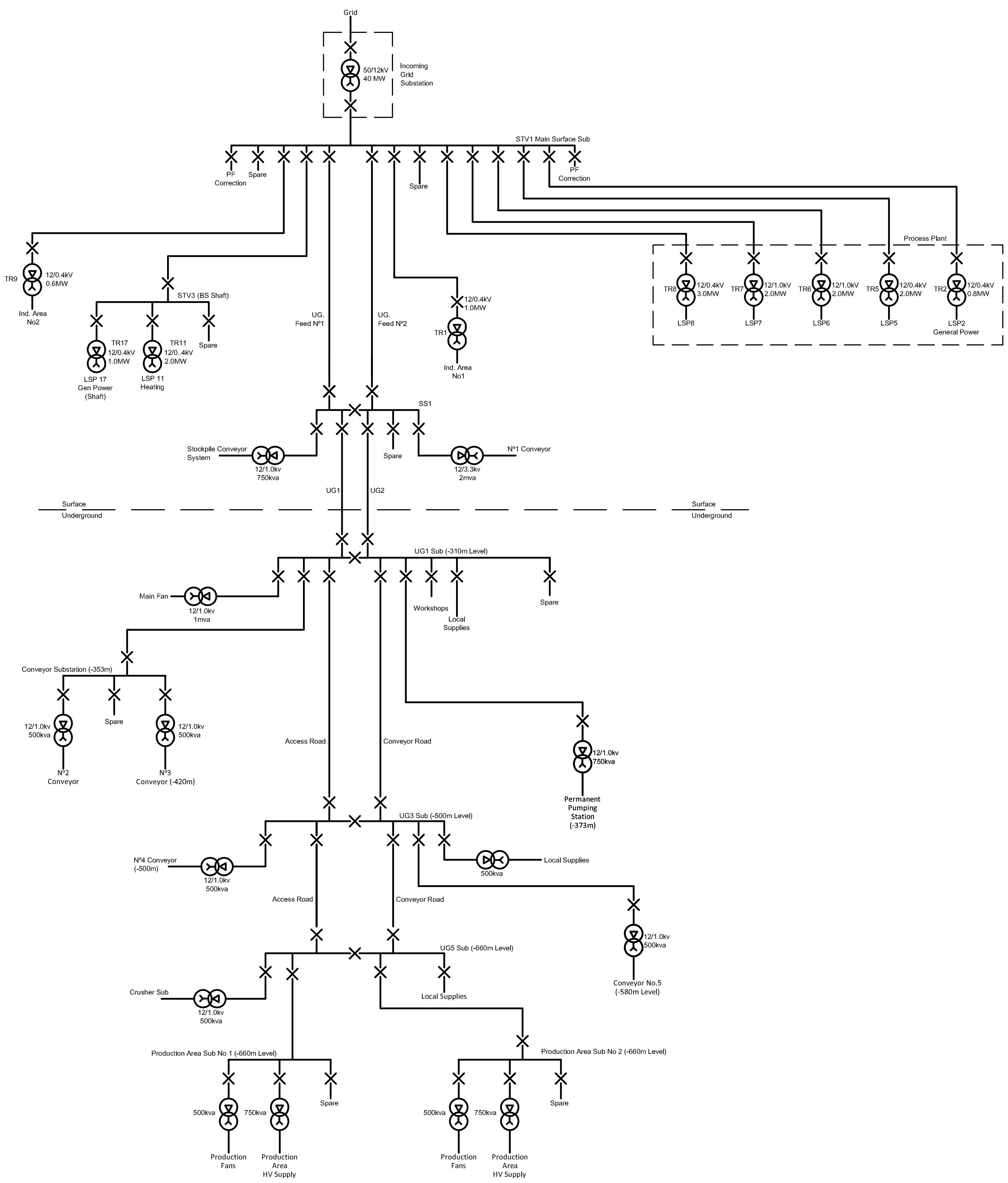
Oxelösunds Hamn

STORLEK	FSCM NR	DWG NR	
A1		1-60672	J
SKALA 1:2000		VIKT	ARK

KONTRAKT NR				Oxelösunds Hamn AB
RITAD	Leif Lidevad/SJn			
GODKÄND				
GODKÄND AV		Huvudkarta med lossn.station		
SKAPAD				
	2010-02-05	STORLEK A0	FSCM NR	DWG NR 5-00656
		SKALA 1:2500	VIKT	ARK

Appendix C

UNDERGROUND DRAWINGS



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Earth. Insight. Values

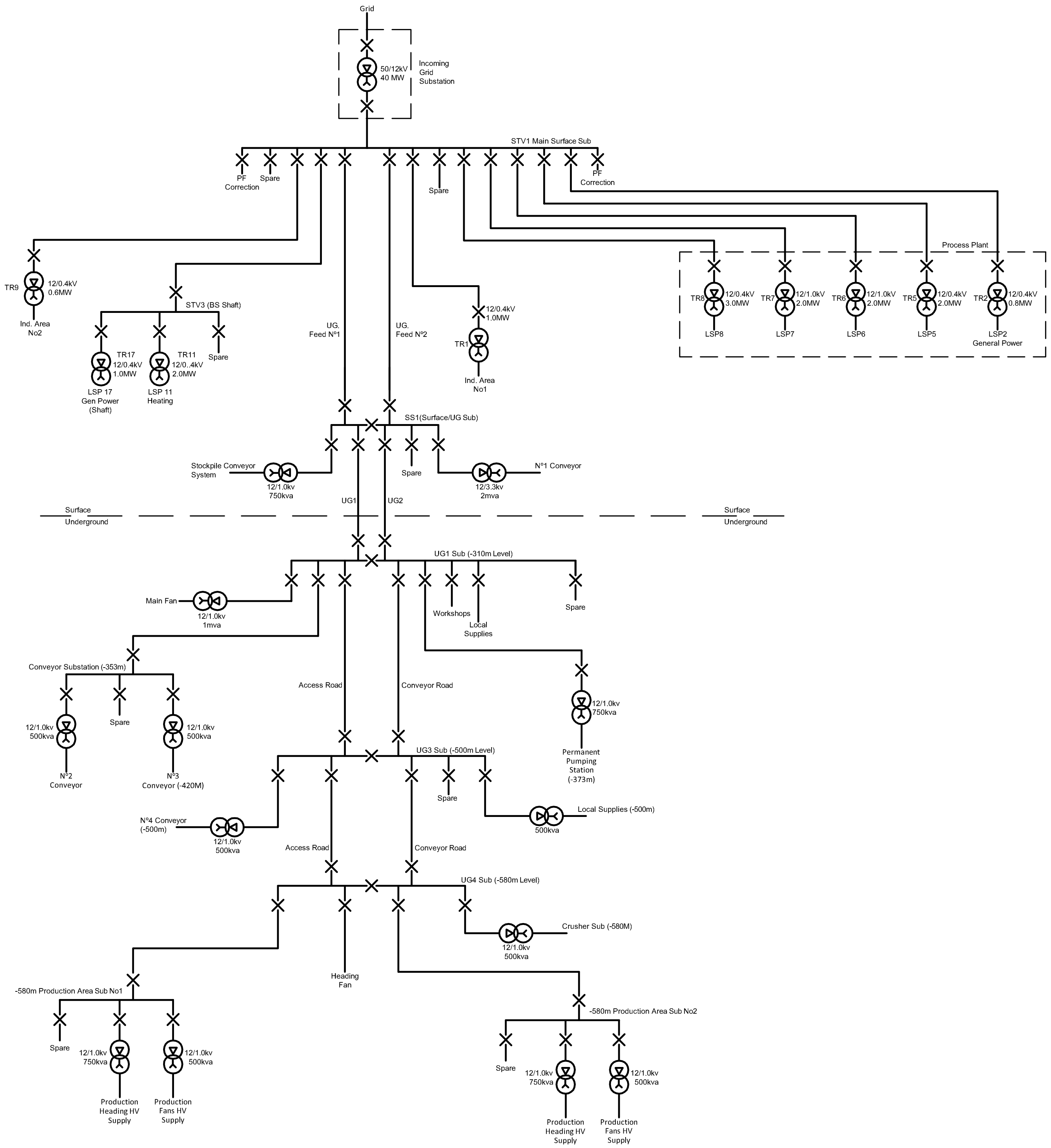
Project:
Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Stage Seven
Development Complete to
-660M level

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	Aprill 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	027	



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Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

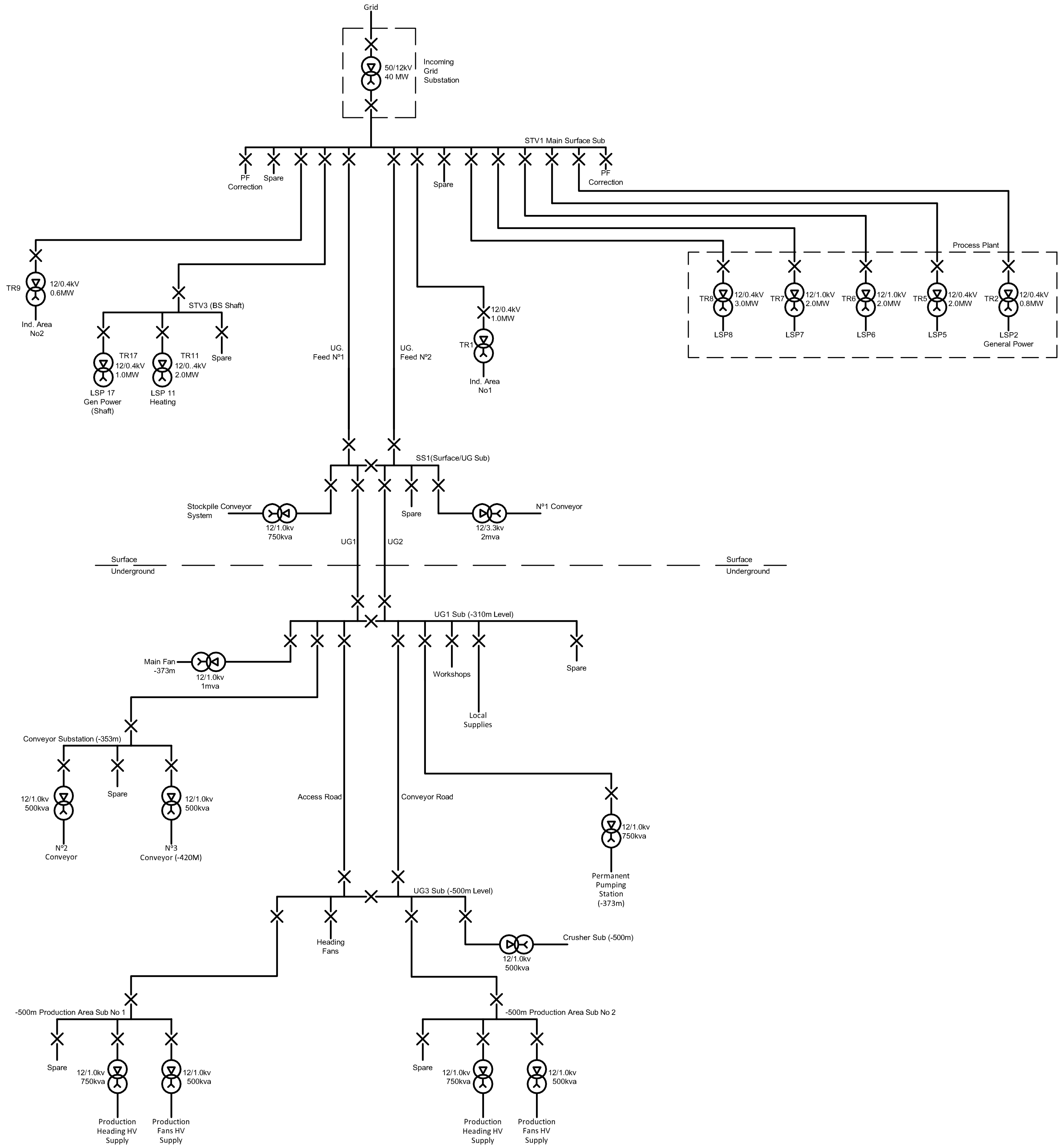
Drawing Title:

Stage Six
Development Complete to
-580M level

Rev	Description	By	Chk	App	Date
	Saved location path				

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	026	



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Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

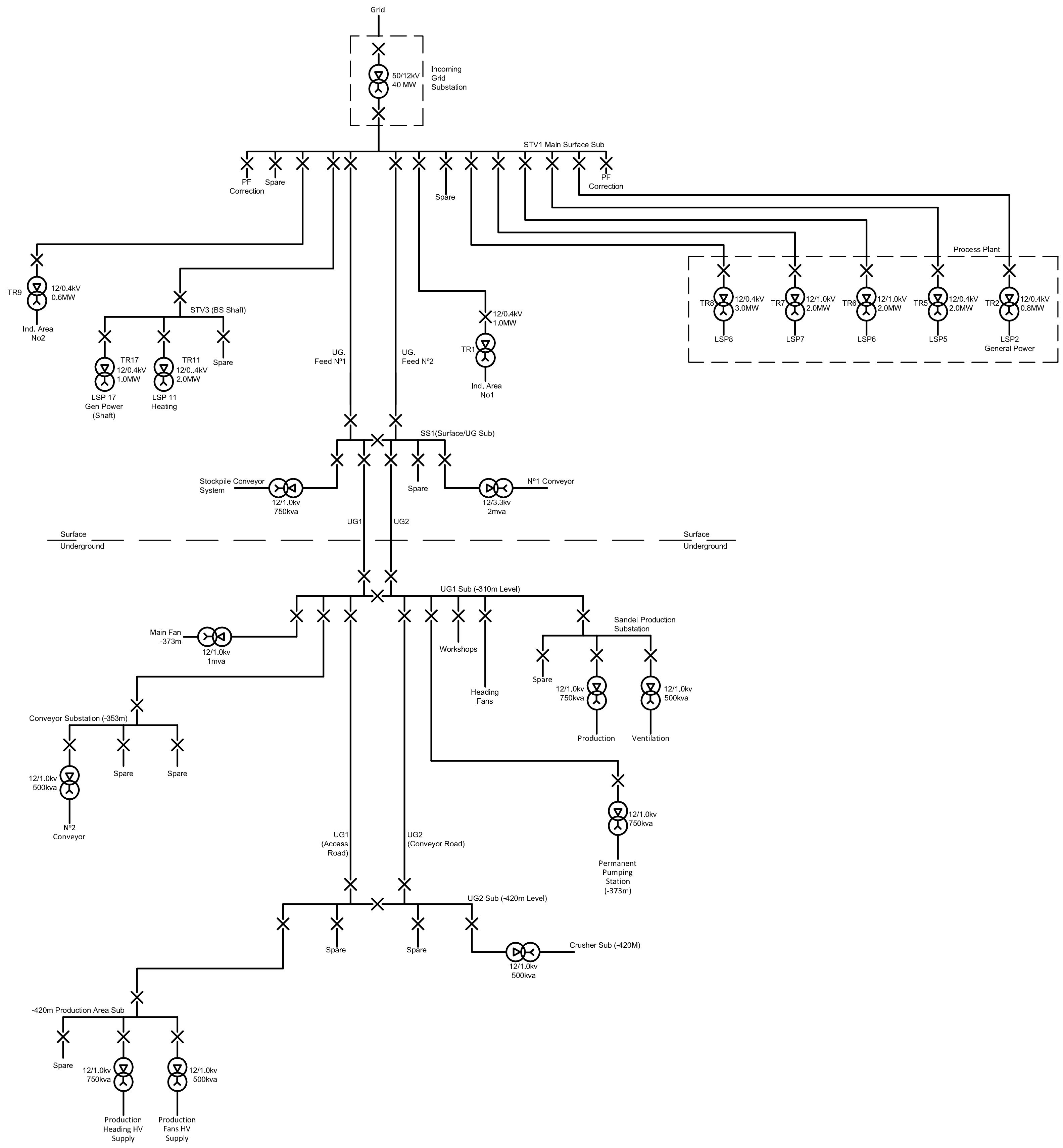
Drawing Title:

Stage Five
Development Complete to
-500M level

Rev	Description	By	Chk	App	Date
	Saved location path				

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	025	



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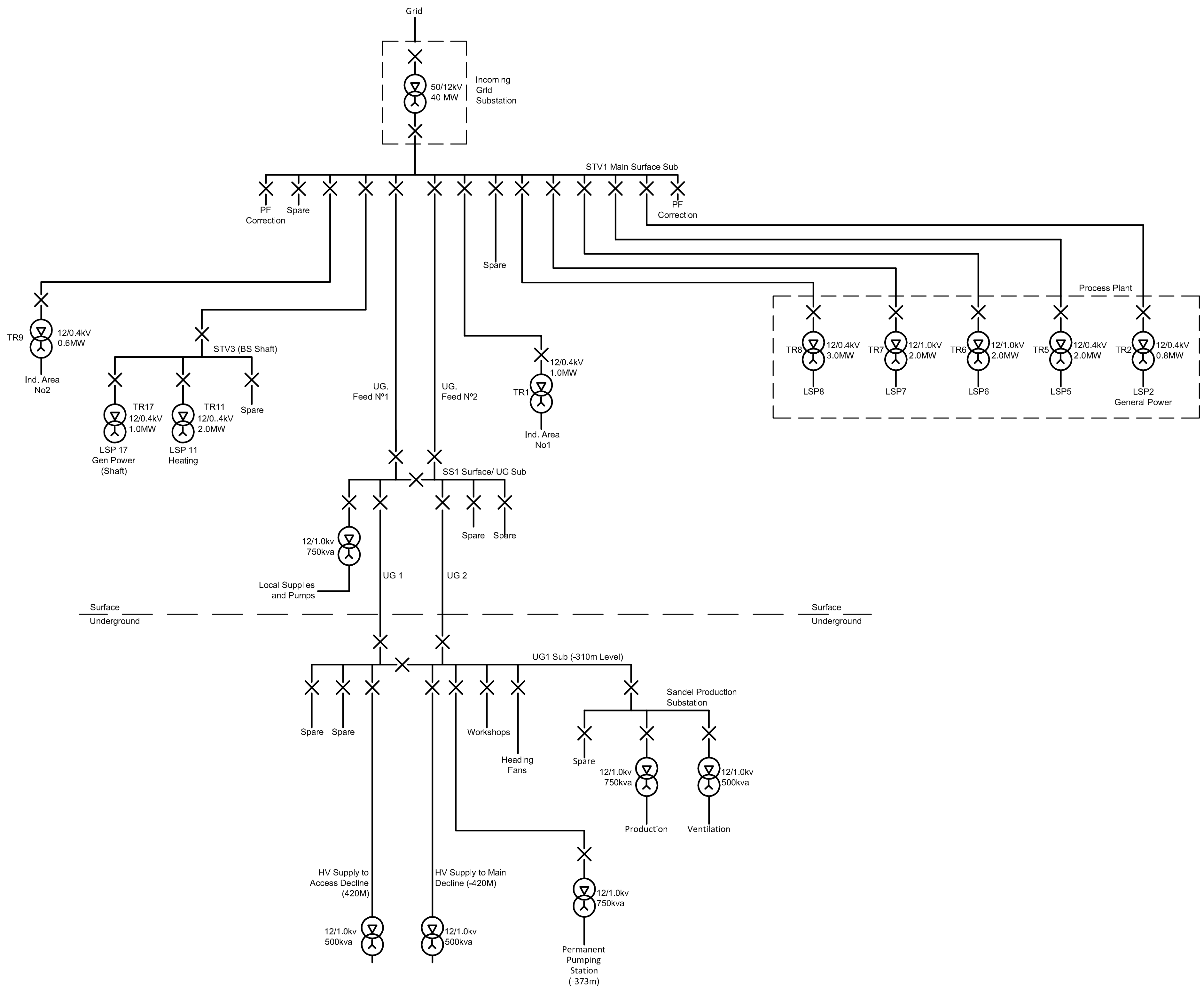
Project:
Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Stage Four
Development Complete to
-420M level

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	024	



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United Kingdom
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Fax: +44 (0)1623 729359
e-mail: uk@dmt-group.com

Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:

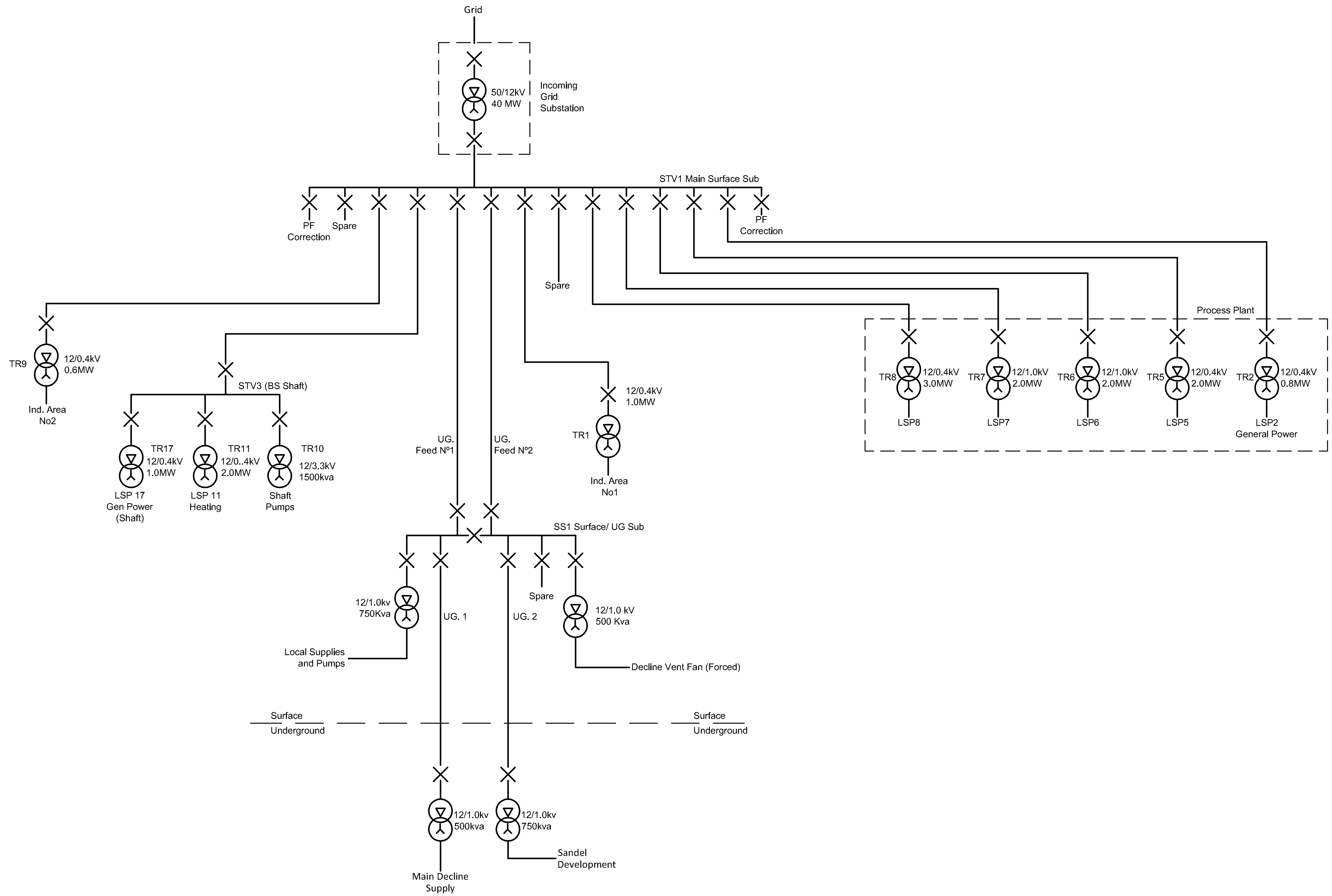
Stage Three
Development Complete to
-310M level

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	023	



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Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

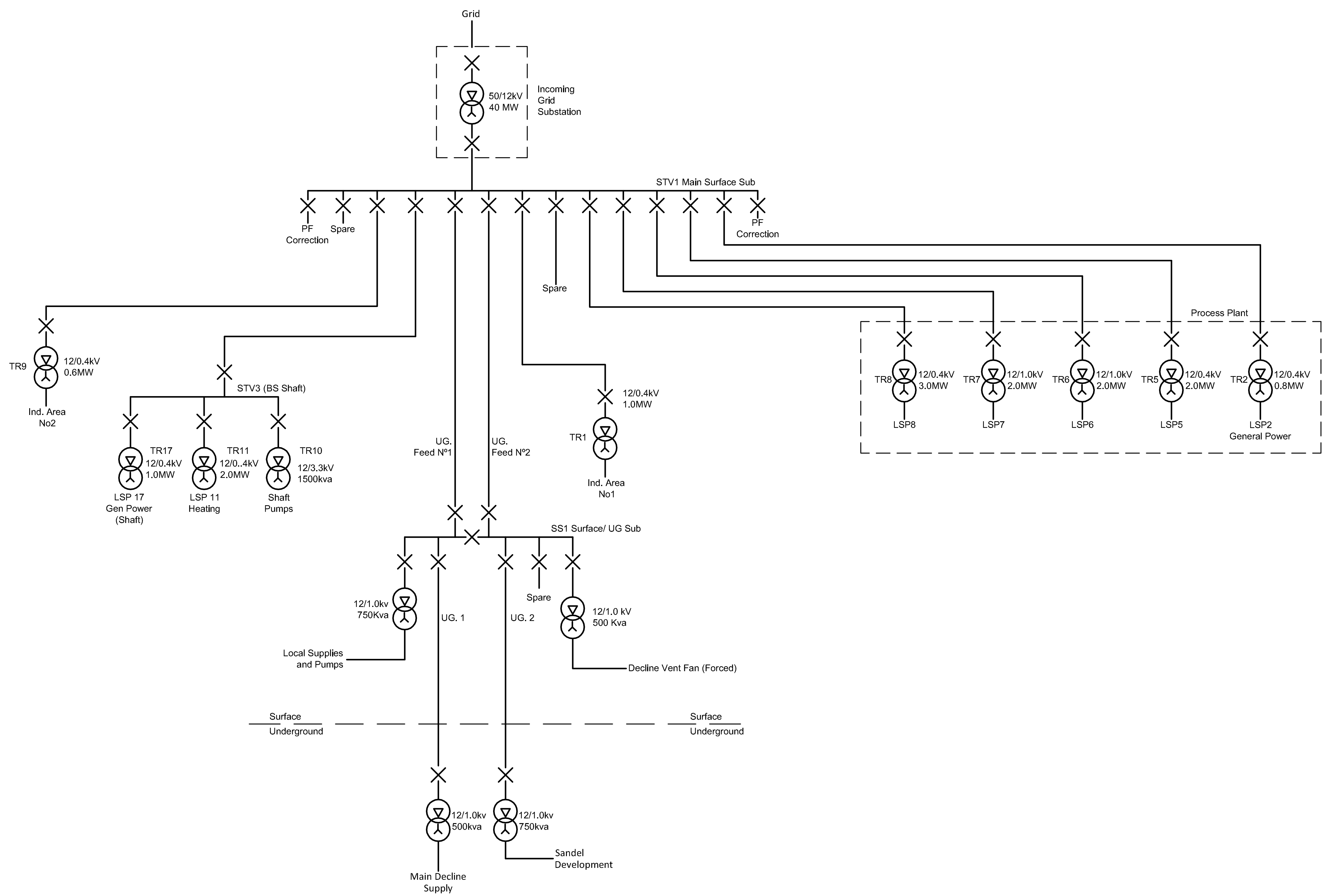
Drawing Title:

Stage Two
Initial Development

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	022	



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Project:

Nordic Iron Ore TM

Blötberget Iron Ore Project
Interim Technical Study

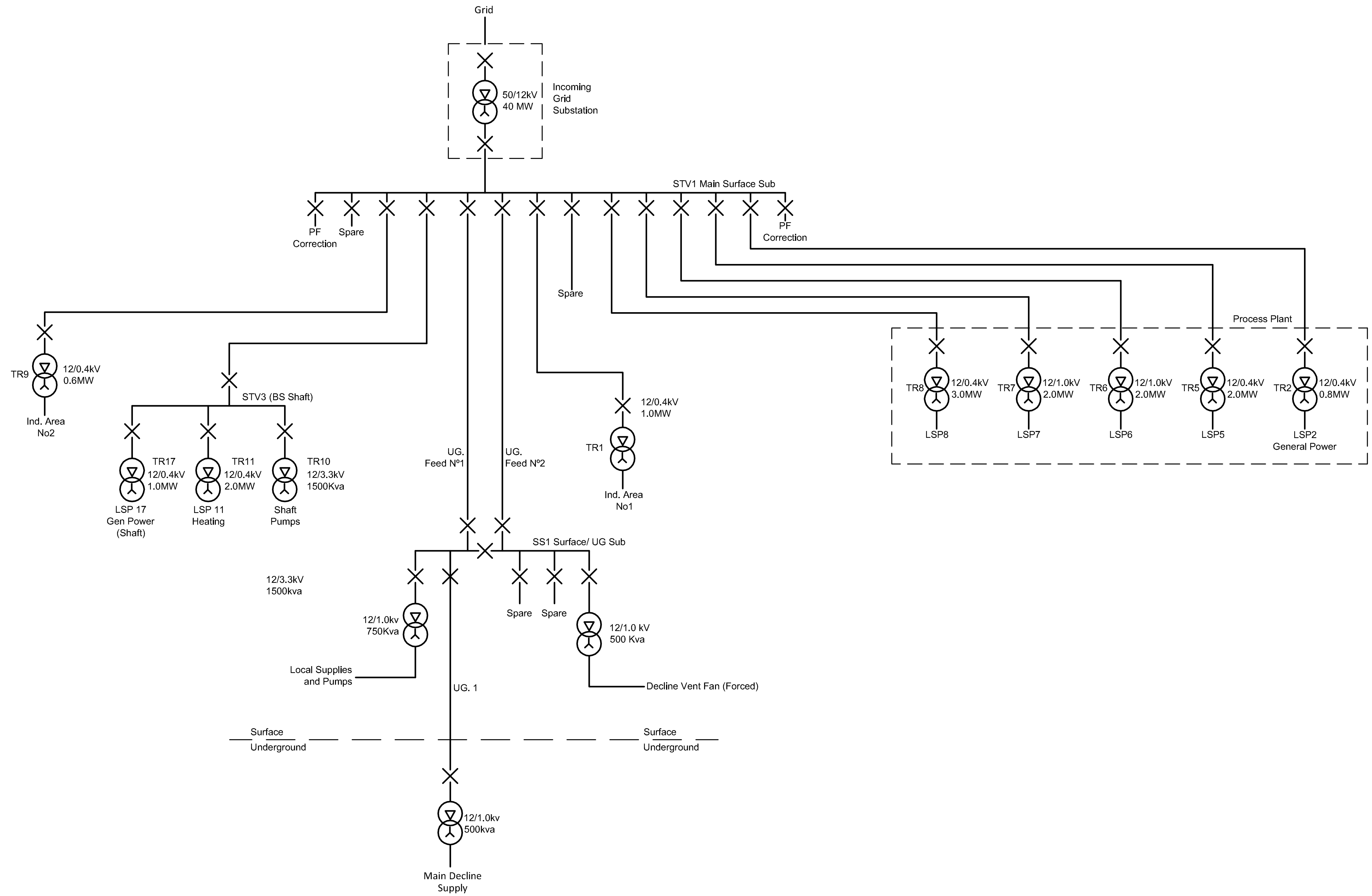
Drawing Title:

Stage Two
Initial Development

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	022	



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Project:
Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

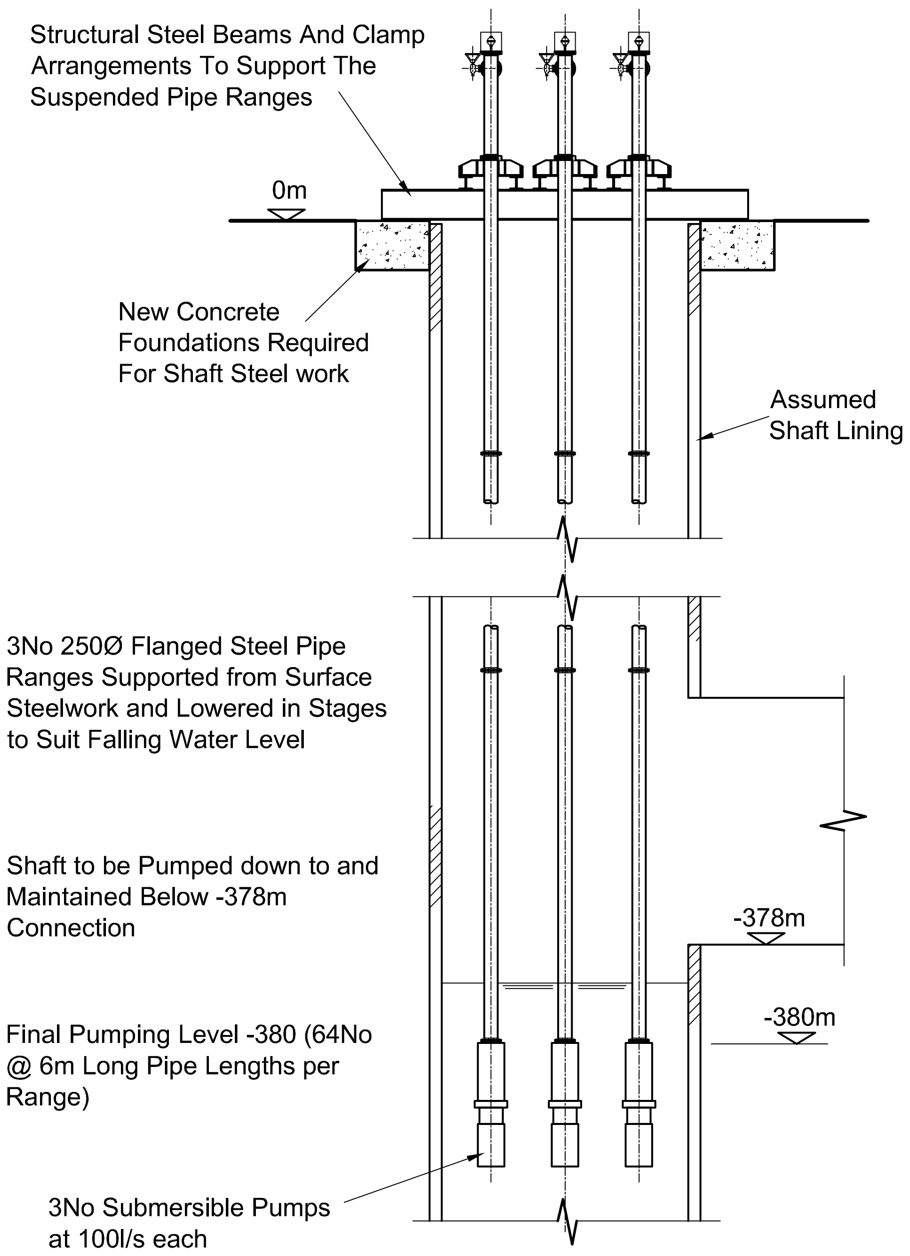
Drawing Title:
Stage One
Initial Development

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	SJ	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	021	



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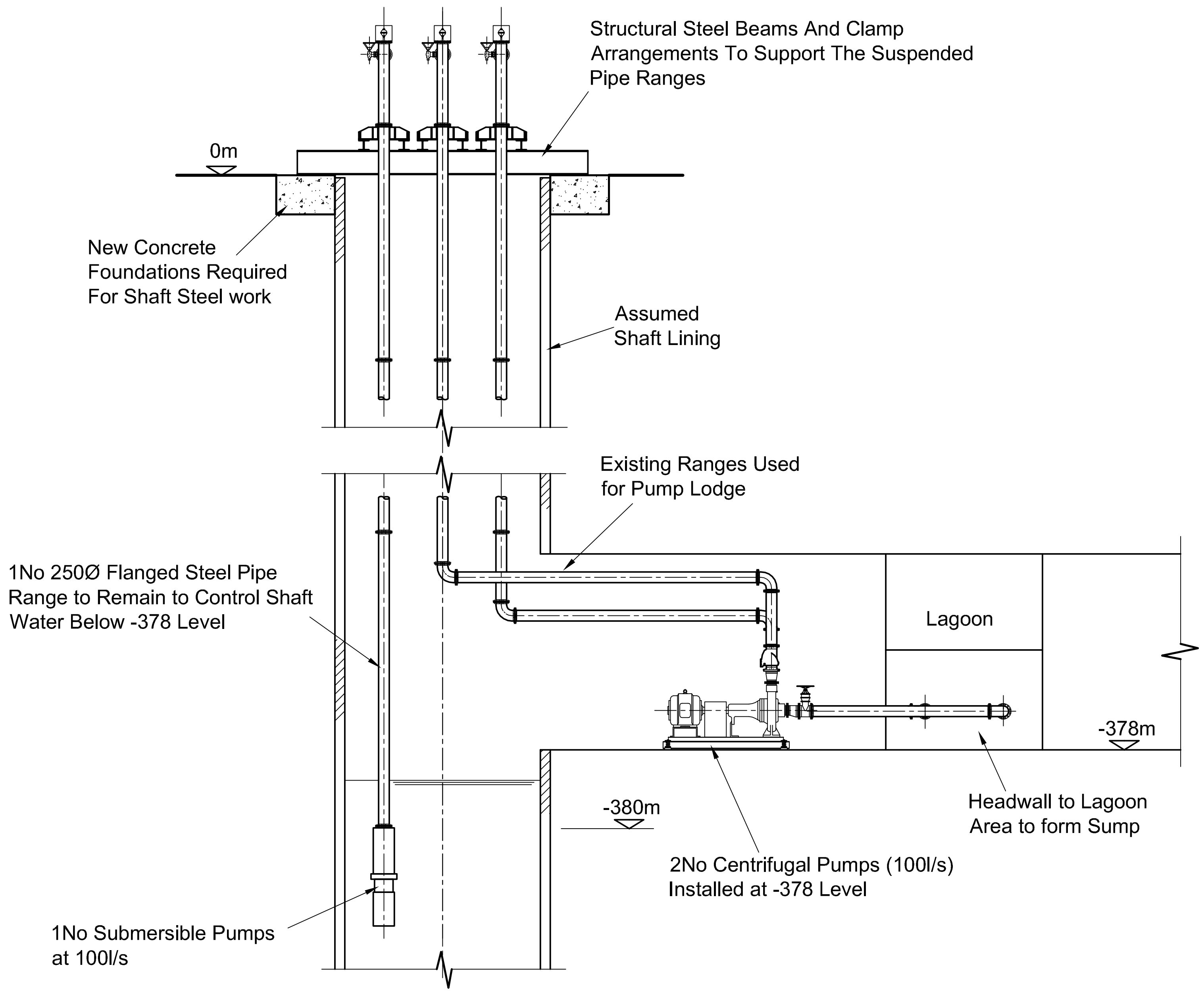

Phase 2 - Operational Pumping

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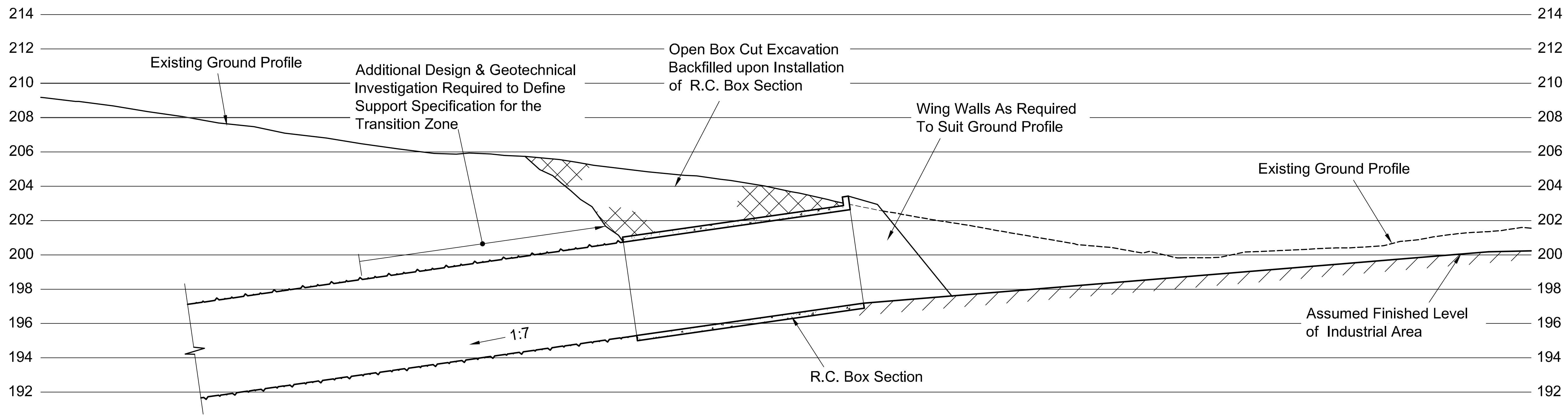
Project: Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
G.A. of Pumping Ranges to Existing Shaft

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	March 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	019	



Section

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Earth. Insight. Values

Project: Nordic Iron Ore™

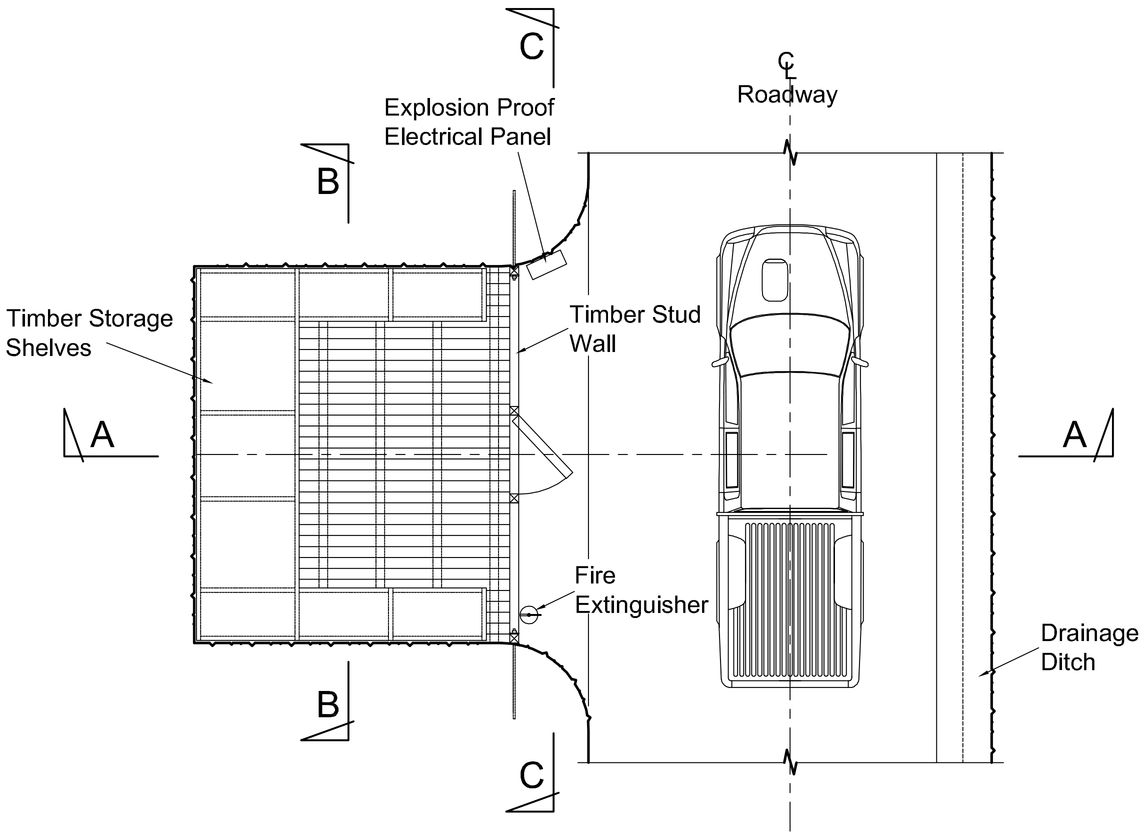
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Conceptual Design of Portal Entrance of Combined Ramp From Industrial Area 1

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
1:200	RS	April 14	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	018	



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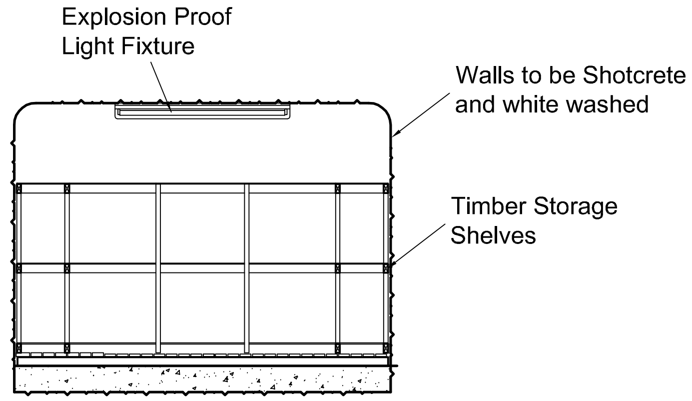

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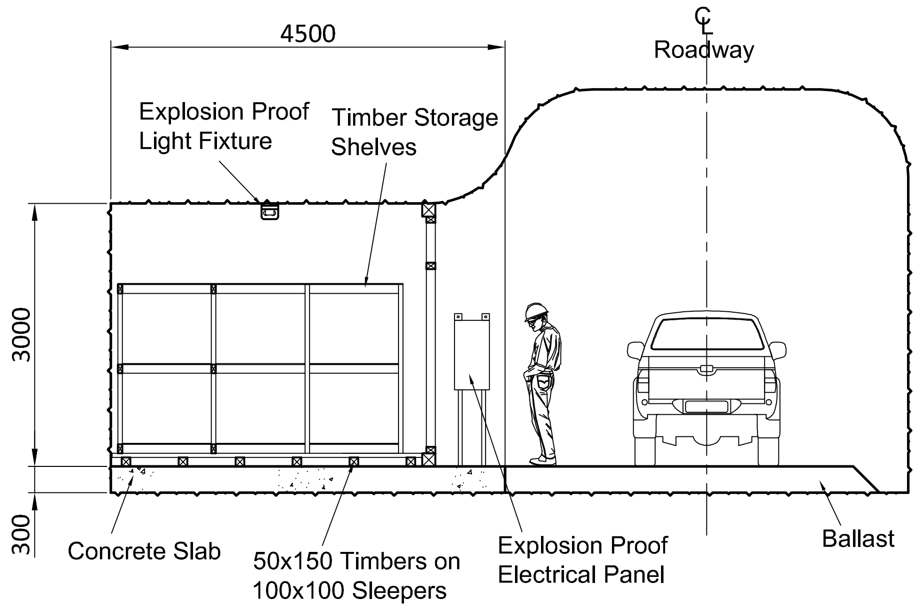

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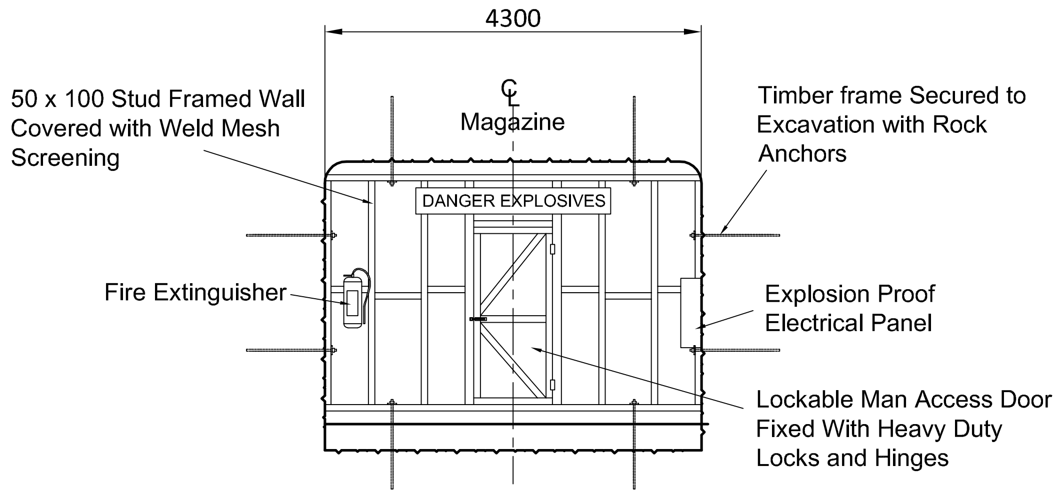

Section C-C

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Earth. Insight. Values

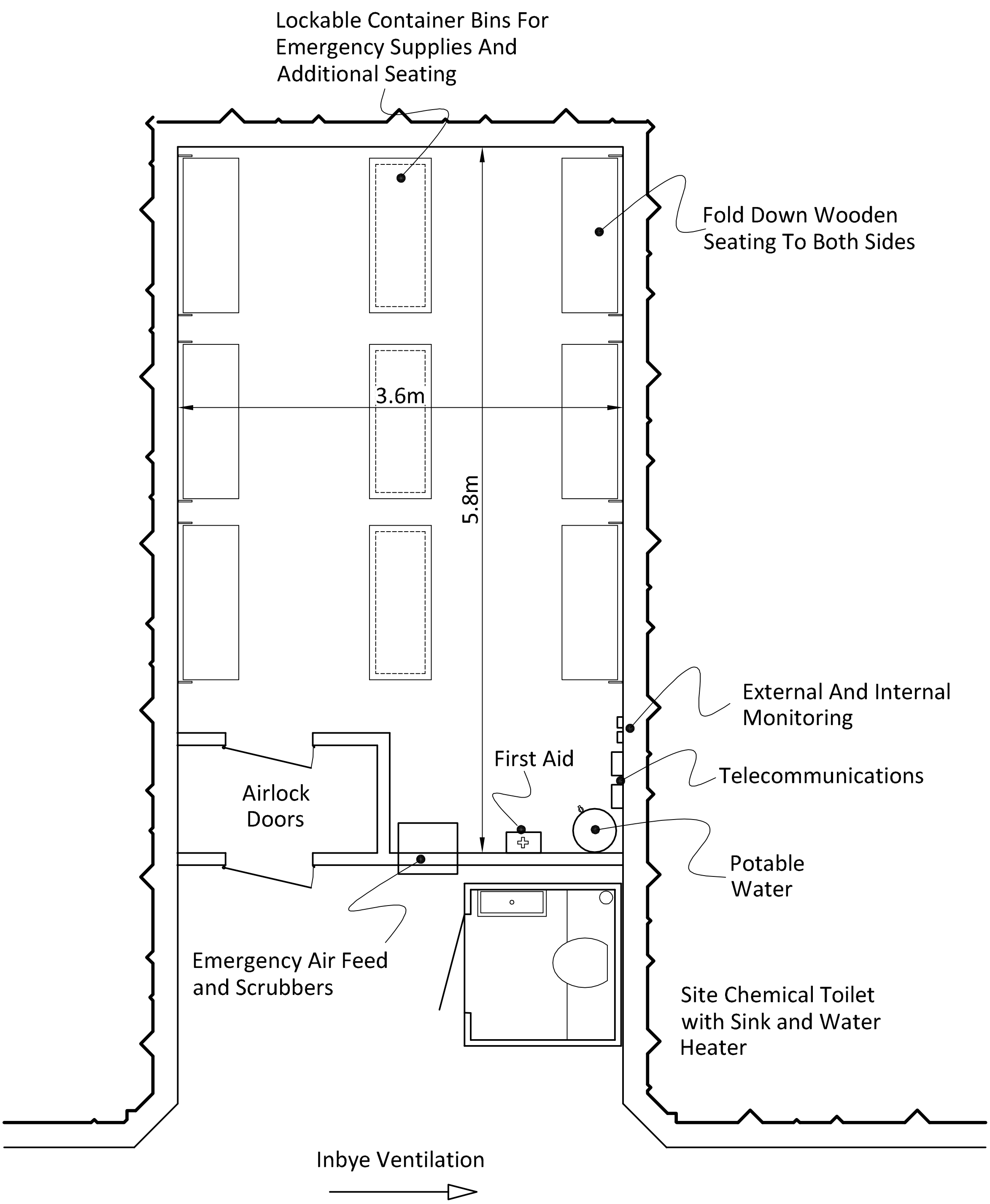
Project:
Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Typical General Arrangement of Detonater Store

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	017	



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Project:
Nordic Iron Ore,
Blotterget Mine

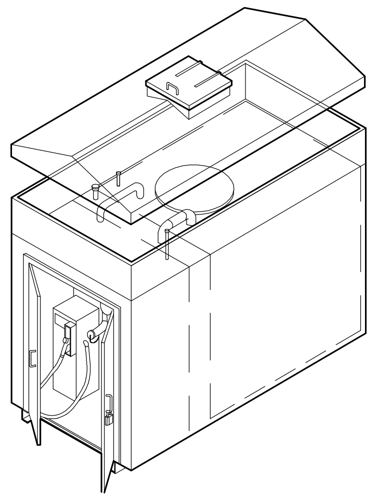
Drawing Title:
Typical Emergency
Muster Station

Rev	Description	By	Chk	App	Date

Saved location path

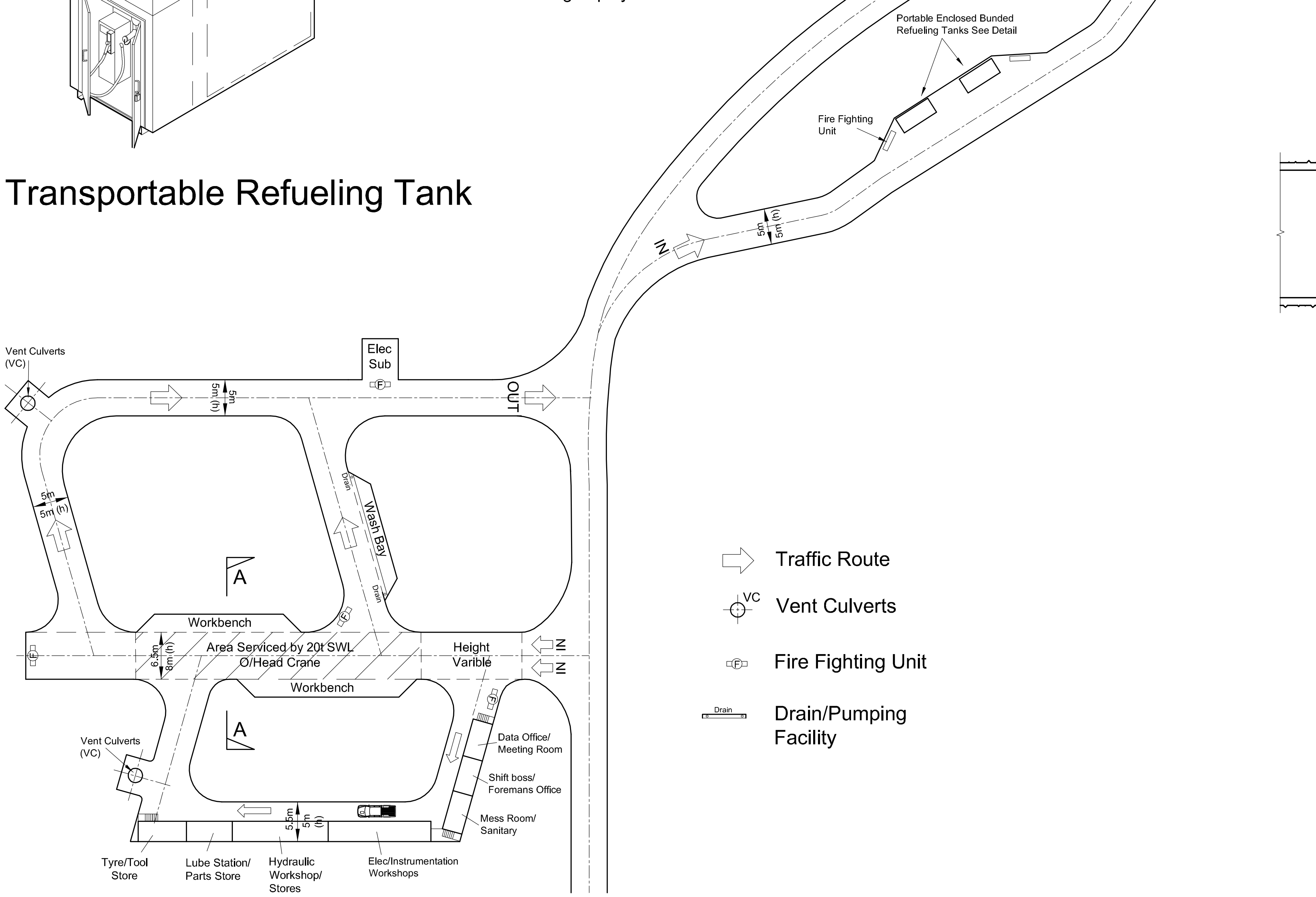
Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Feb 15	BR	April 15	DJFS	April 15

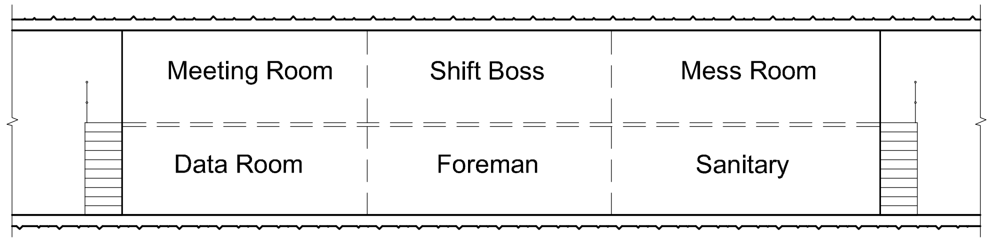
Project No.	Drawing No.	Revision
C22	016	



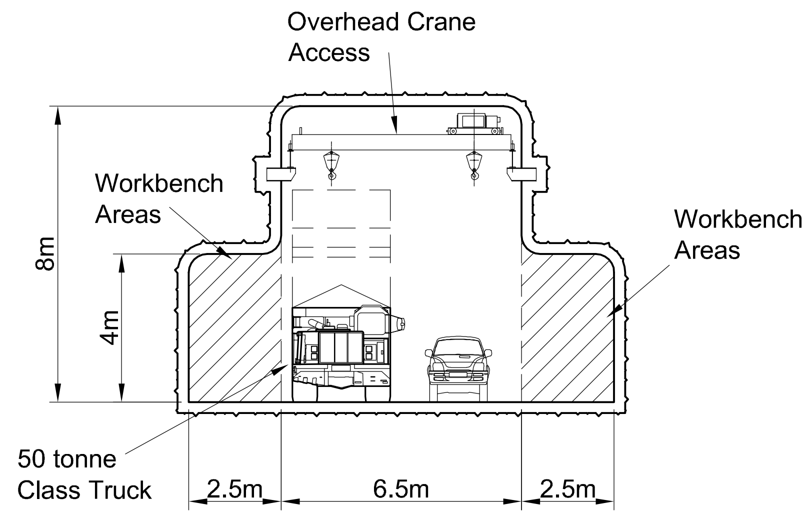
- Self Contained Unit with enclosed bunded tank, bund capable of containing 110% of tank volume
- Tank either mounted on sledge or trailer for removal to surface for refilling or in the event of a emergency
- Two tanks per station, one in use underground, other at surface filled and waiting deployment

Transportable Refueling Tank





Typical Elevation on Workshop/Stores Arrangement



Section A-A

Typical Maintenance Complex Layout

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Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:

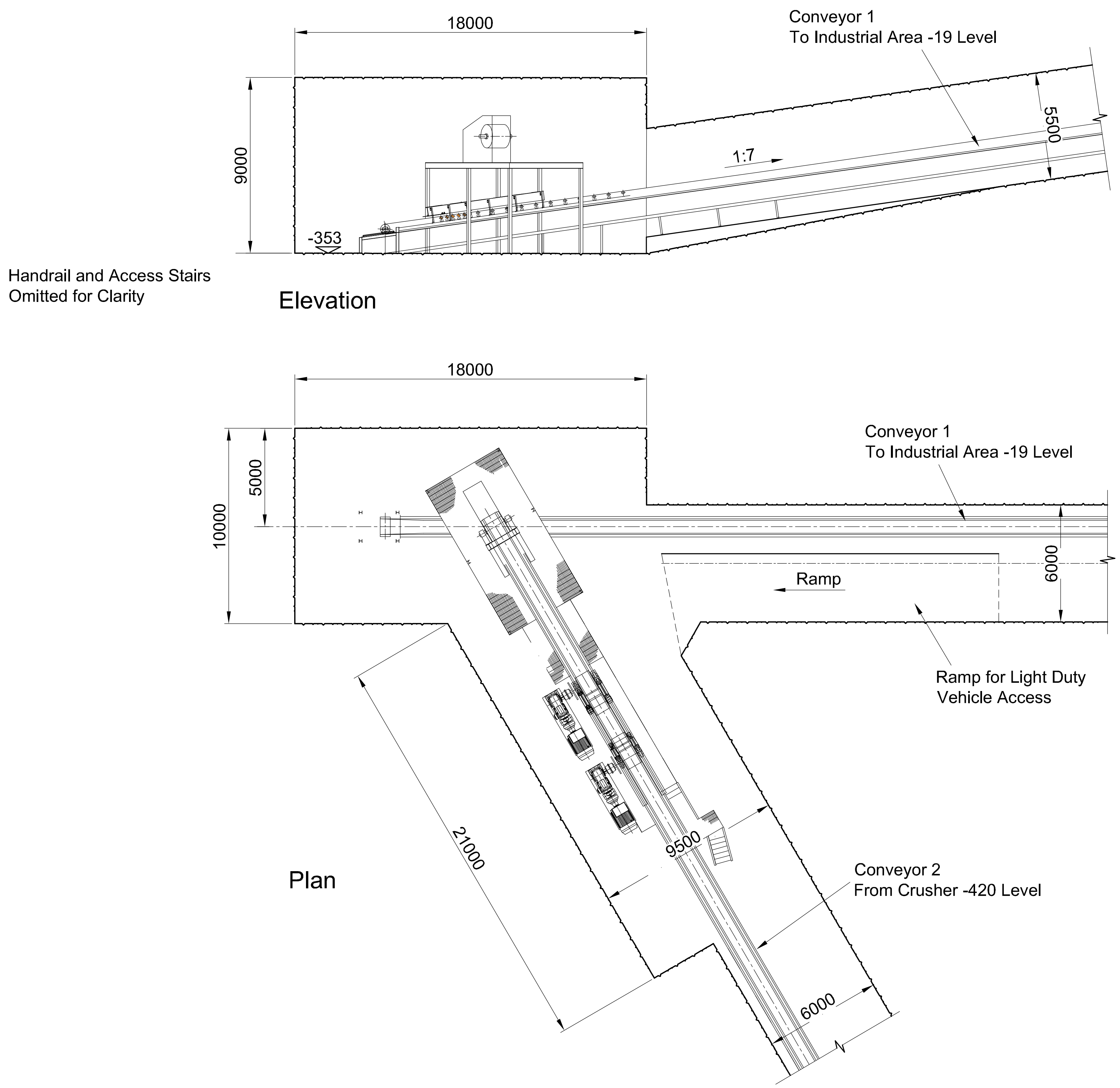
Typical Underground Maintenance Complex

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Nov 14	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	015	



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Earth. Insight. Values

Project:
Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

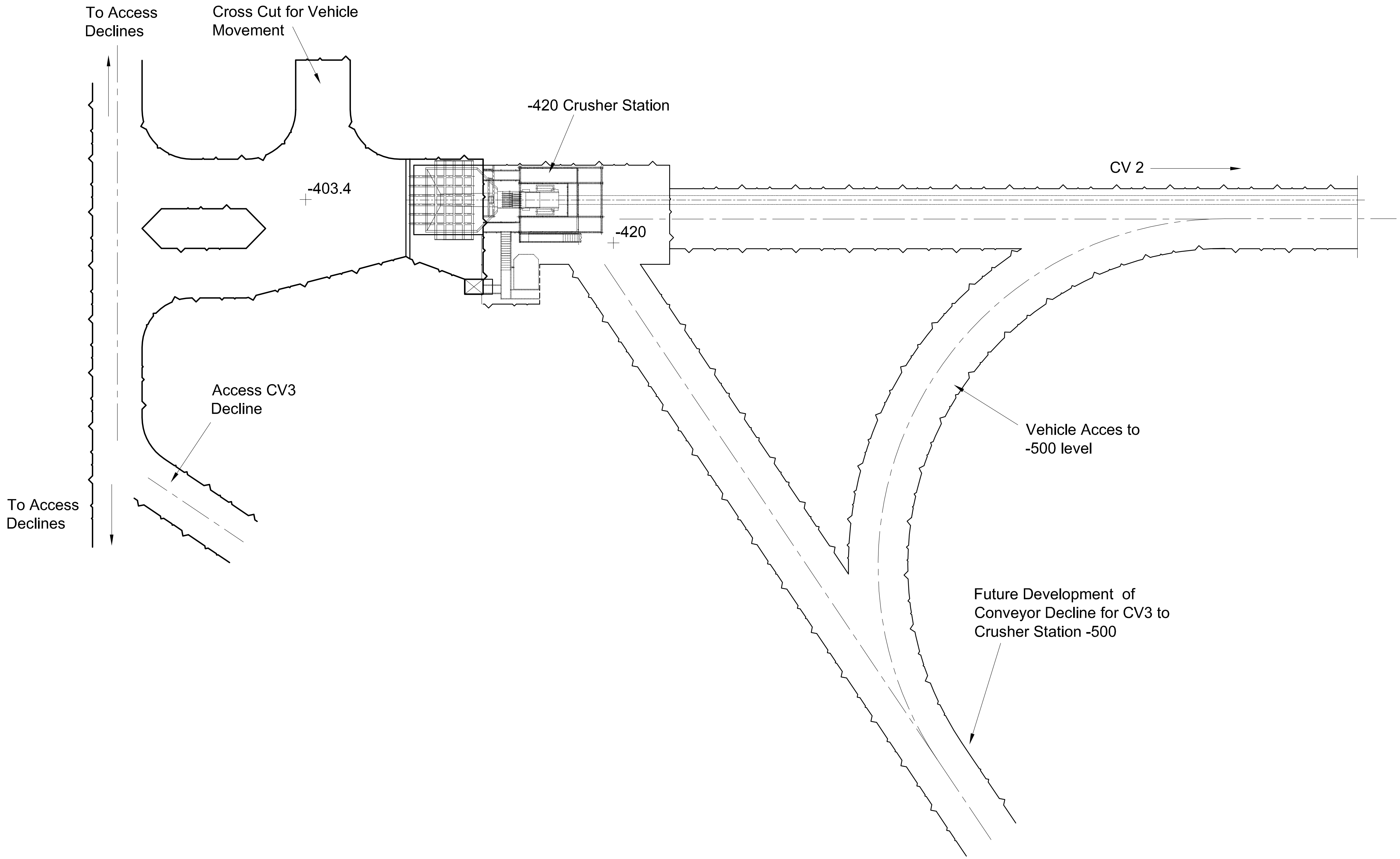
Drawing Title:
Schematic Arrangement of Belt Transfer Station CV2 to CV1 (-353 Level)

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
1:250	RS	March 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	014	



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Earth. Insight. Values

Project: Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

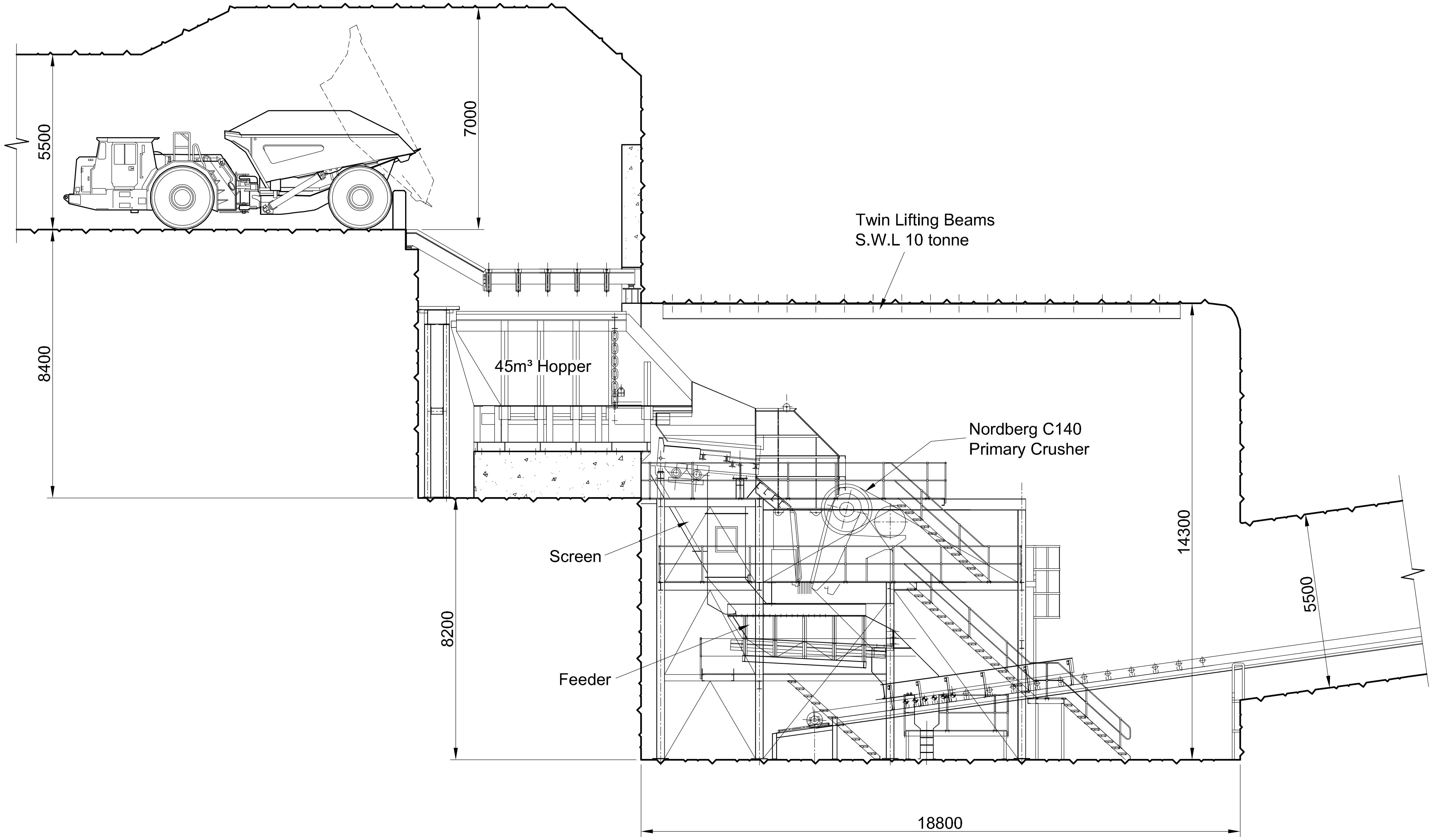
Drawing Title:

Crushing Station Showing Plan Levels

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
1:200	RS	Nov 14	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	013	



Elevation

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Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

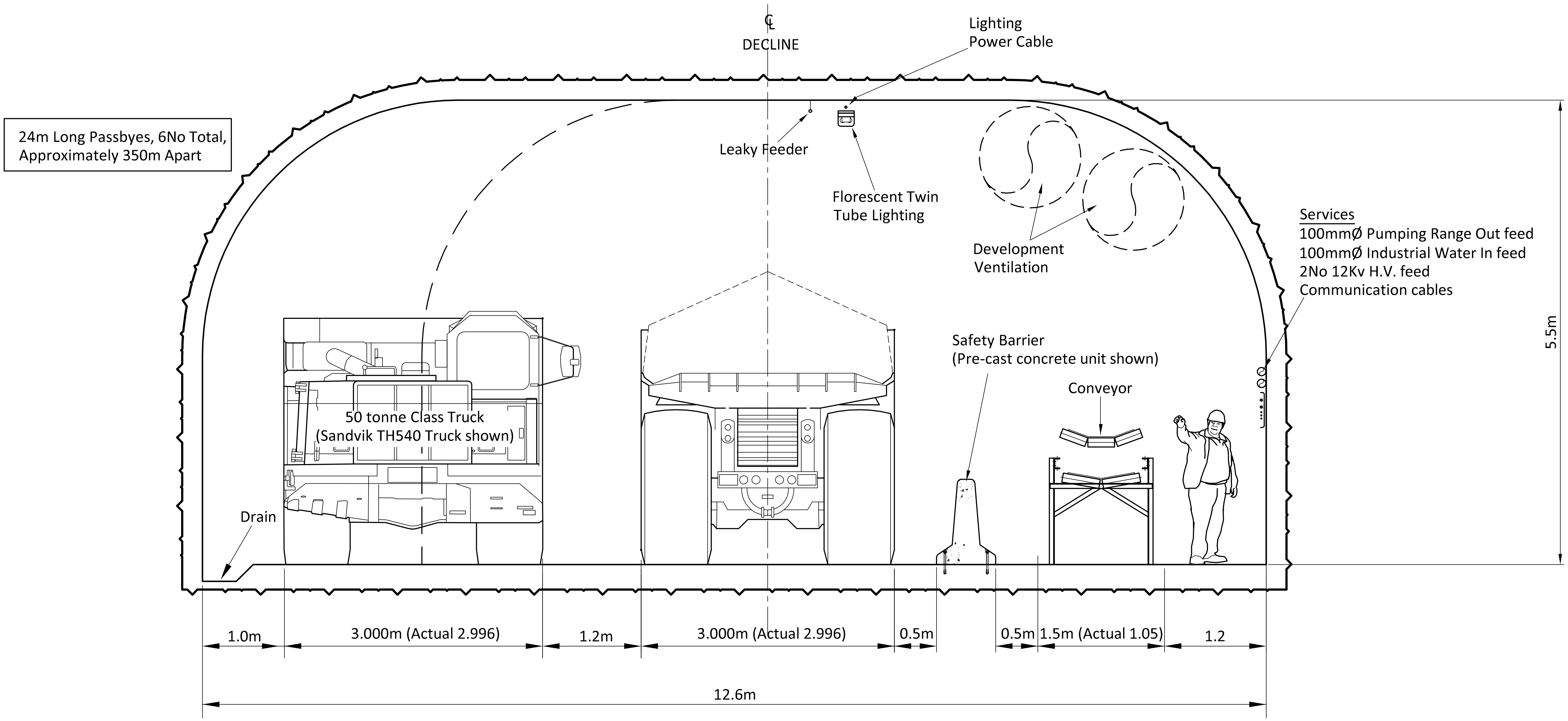
Drawing Title:

Crushing Station

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
1:200	RS	Nov 14	BR	April 15	DFJS	April 15

Project No.	Drawing No.	Revision
C22	012	



SECTION SHOWING COMBINED RAMP TO INDUSTRIAL AREA AT PASSING POINT

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Earth. Insight. Values

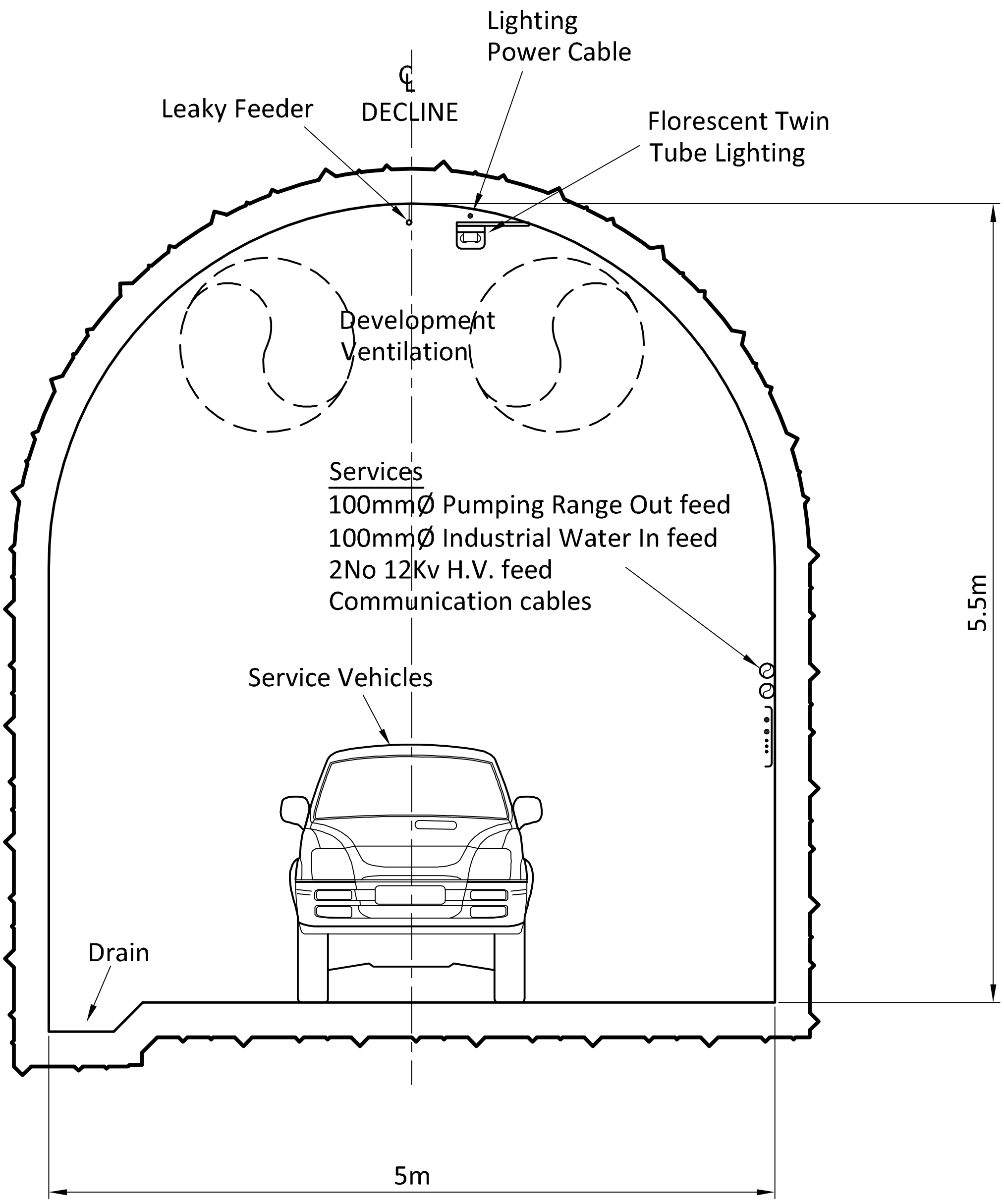
Project: Nordic Iron Ore™ Blötberget Iron Ore Project Interim Technical Study

Drawing Title: Typical Section of Combined Ramp to Industrial Area (-19 to -320)

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	March 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	011	



SECTION SHOWING ACCESS DECLINE

DMT

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TUV NORD GROUP

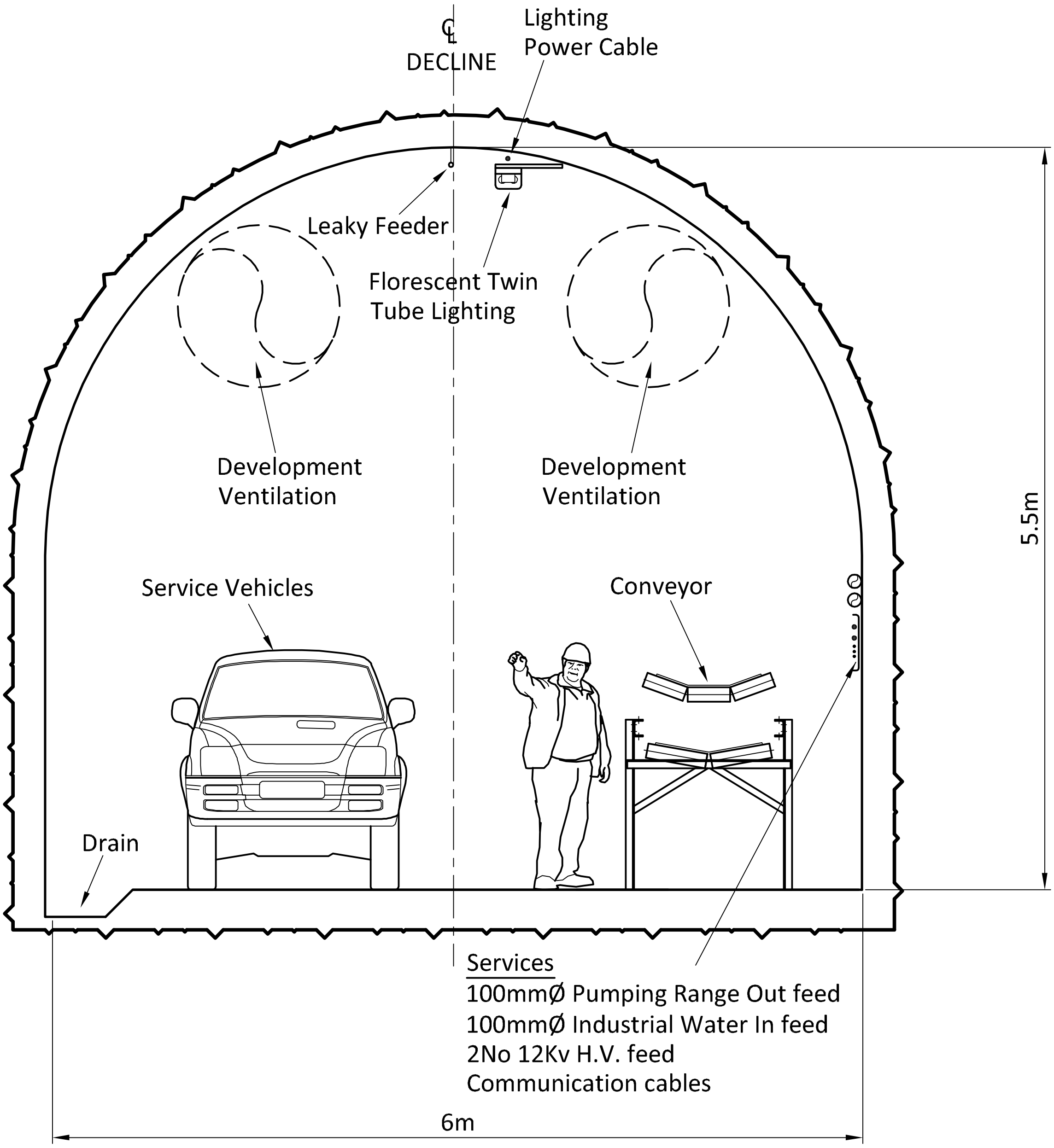
Project:
Nordic Iron Ore,
Blotterget Mine,

Drawing Title:
Typical Access Decline
Section 5.0m x 5.5m
-320 to -660

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S	RS	March 15	BR	April 15	DJFS	Aprl 15

Project No.	Drawing No.	Revision
C22	010	



SECTION SHOWING CONVEYOR DECLINE

DMT
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TUV NORD GROUP

Project:
Nordic Iron Ore,
Blotterget Mine,

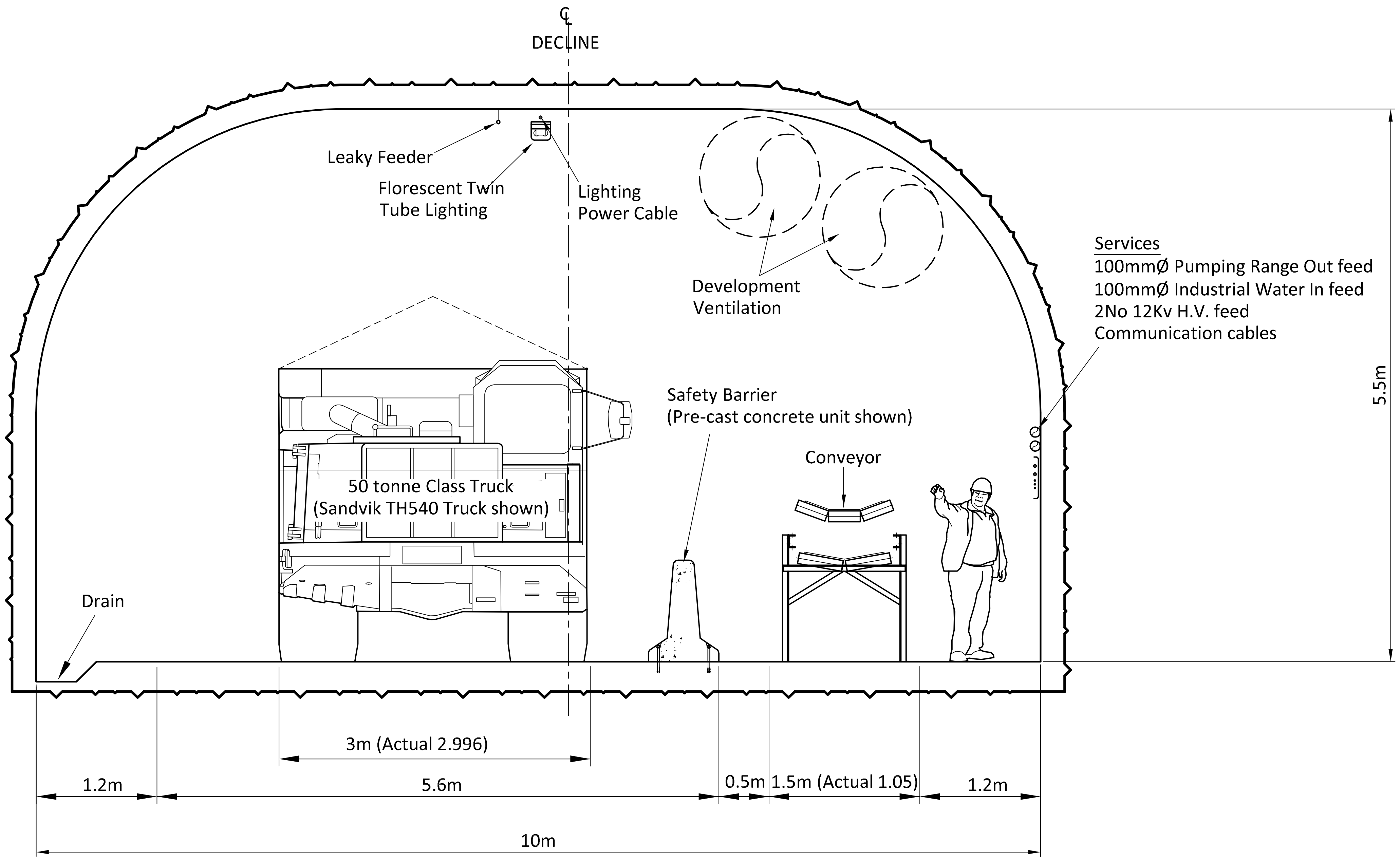
Drawing Title:
Typical Conveyor Decline
Section 6.0m x 5.5m
-320 to -660

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By / Date	Checked By / Date	Approved By / Date
N.T.S	RS March 15	BR April 15	DJFS April 15

Project No.	Drawing No.	Revision
C22	009	



SECTION SHOWING COMBINED RAMP TO INDUSTRIAL AREA

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Earth. Insight. Values

Project:
Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Typical Section of Combined Ramp to Industrial Area
(-19 to -320)

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	March 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
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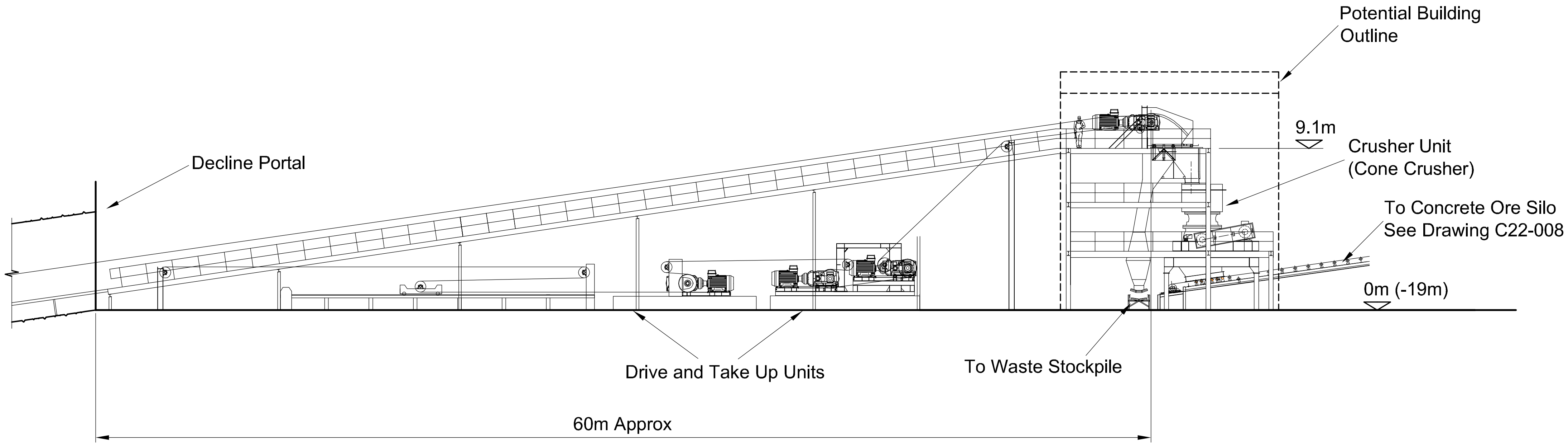
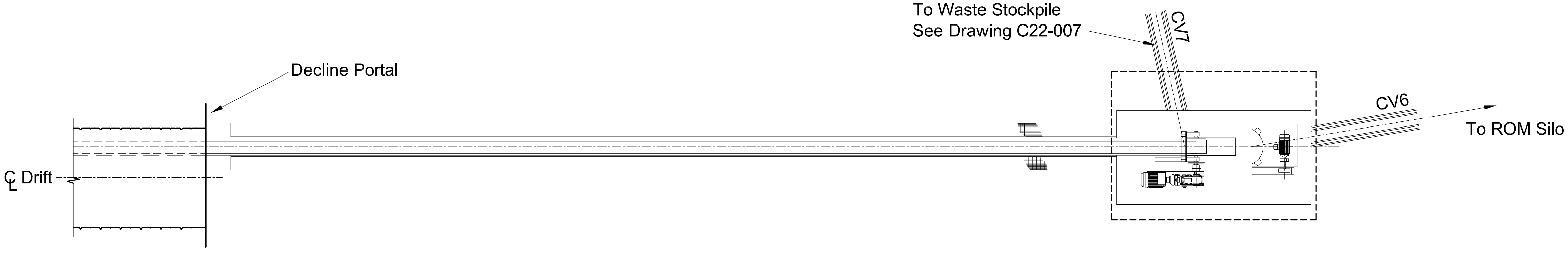


Elevation



Plan

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e-mail: uk@dmt-group.com

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Project: Nordic Iron Ore™
Blötberget Iron Ore Project
Interim Technical Study

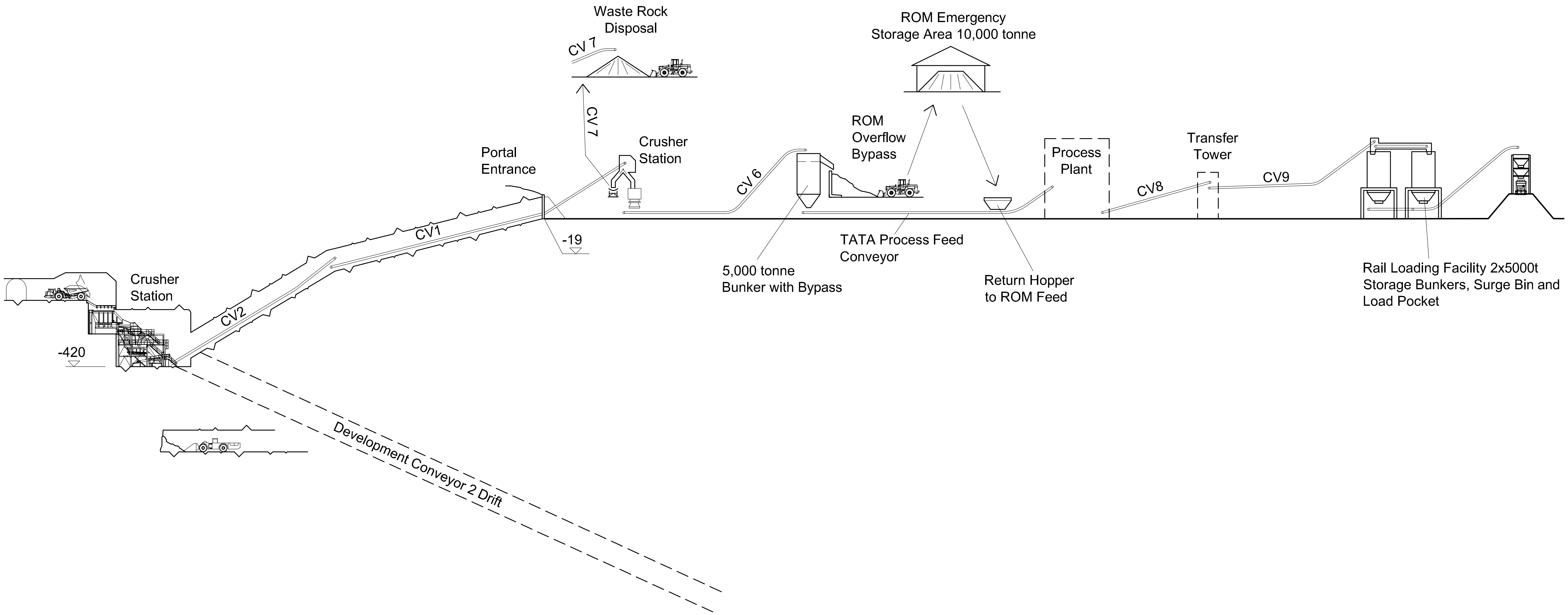
Drawing Title:
Plan & Elevation of Conveyor (CV 6) & Crusher Unit

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	March 15	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	004	



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Project:
Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:
Schematic of Materials Handling

Rev	Description	By	Chk	App	Date
	Saved location path				

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
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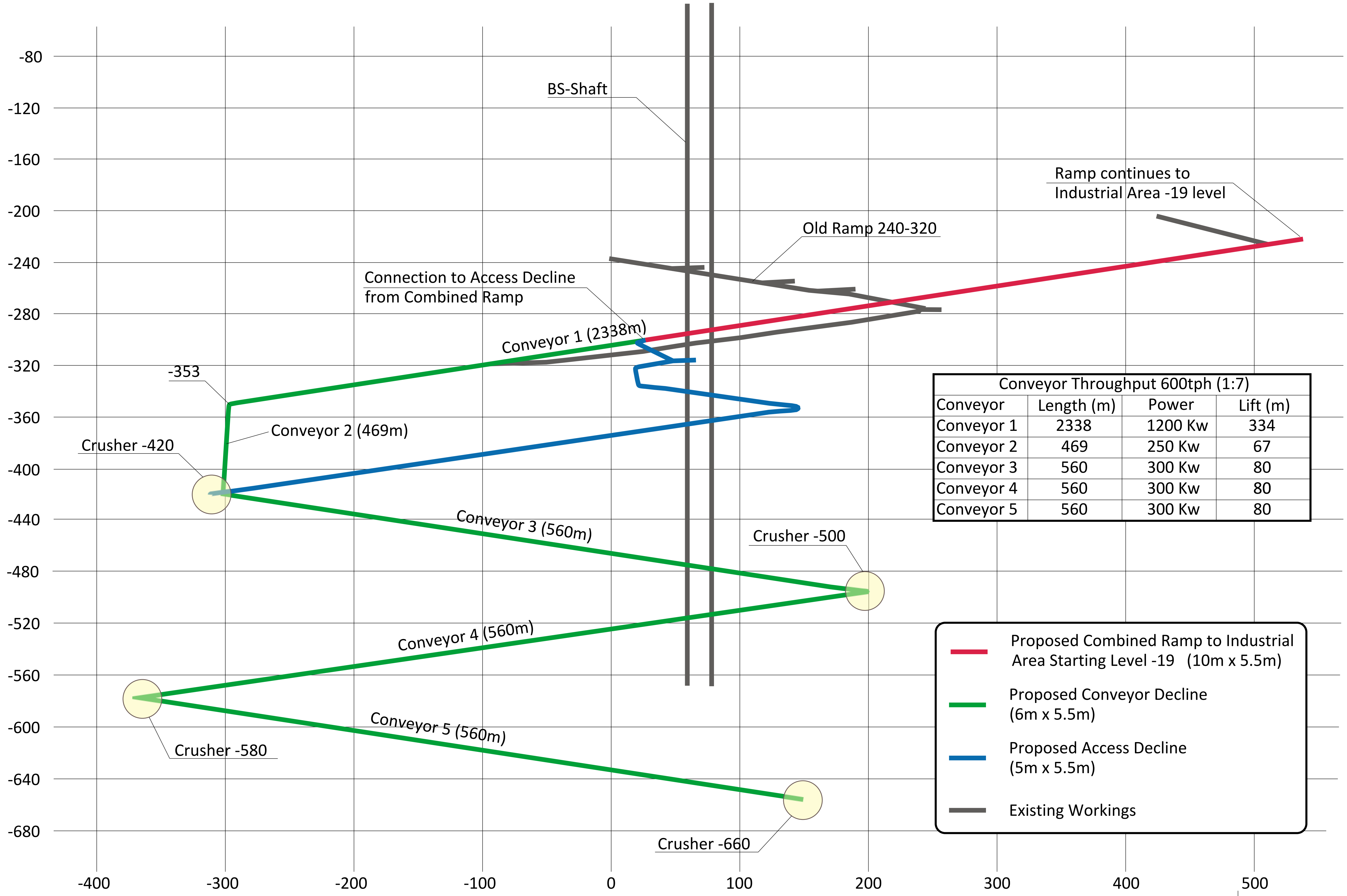
Project No.	Drawing No.	Revision
C22	003	



Conveyor Throughput 600tph (1:7)

Conveyor	Length (m)	Power	Lift (m)
Conveyor 1	2338	1200 Kw	334
Conveyor 2	469	250 Kw	67
Conveyor 3	560	300 Kw	80
Conveyor 4	560	300 Kw	80
Conveyor 5	560	300 Kw	80

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Project: Blötberget Iron Ore Project
Interim Technical Study

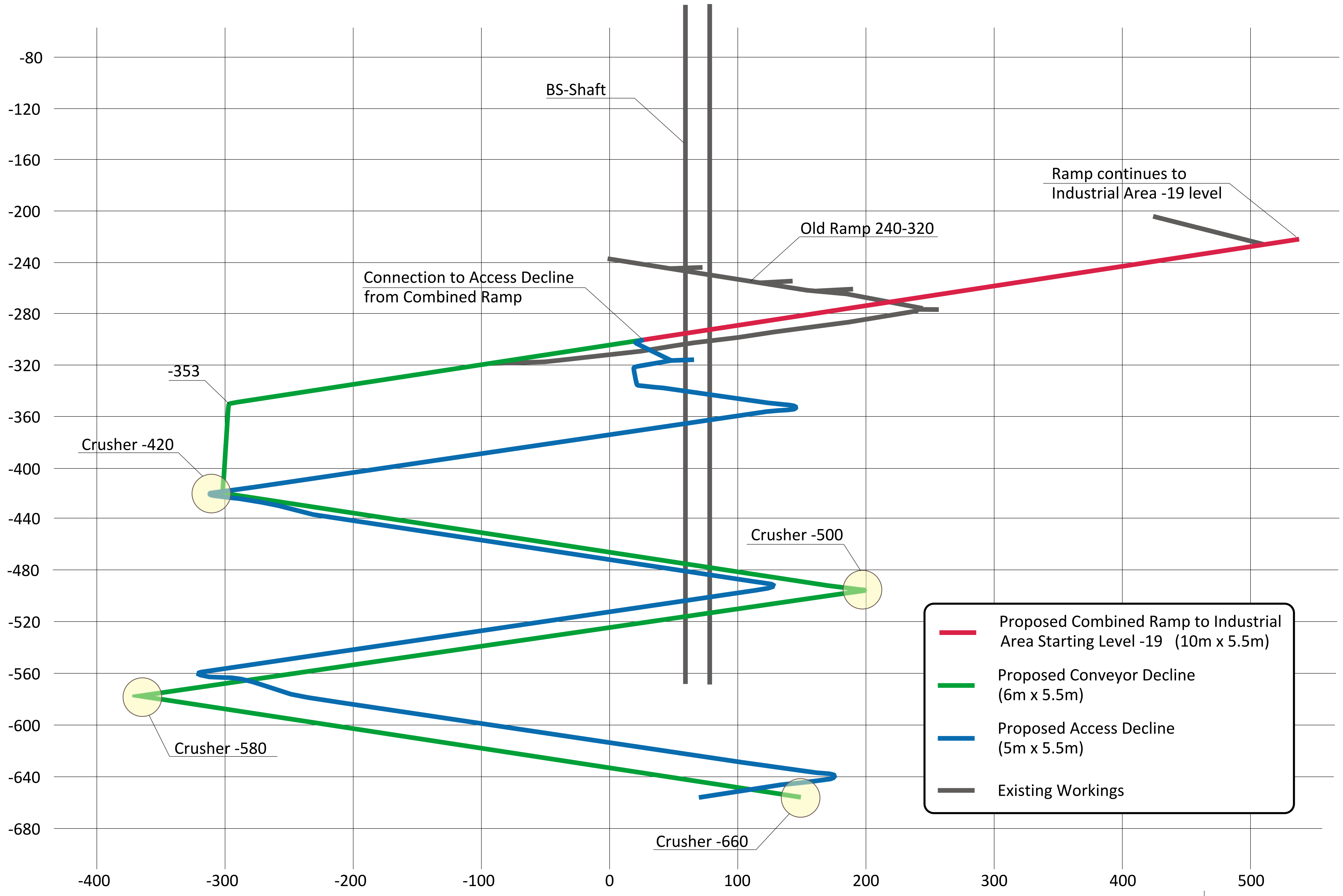
Drawing Title:
Schematic Layout of Mine Conveyors Phase 1

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Nov 14	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	002	



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Earth. Insight. Values

Project:

Nordic Iron Ore™

Blötberget Iron Ore Project
Interim Technical Study

Drawing Title:

Schematic Layout of Mine Infrastructure

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Nov 14	BR	April 15	DJFS	April 15

Project No.	Drawing No.	Revision
C22	001	

Appendix D

BASIS OF DESIGN

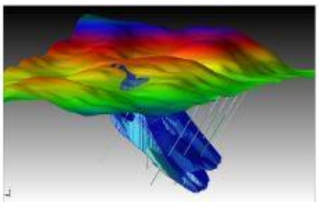
DMT Consulting Ltd
International Mining Consultants

Blötberget Iron Ore Project

Interim Technical Study

Preliminary Basis of Design









March 2015

Prepared for

Nordic Iron Ore™

Document Ref.: C22-R-123

Document Details

Project Details			
Document Title:	Preliminary Basis of Design		
Document Number:	C22-R-123	**Revision:**	R4
Client:	Nordic Iron Ore		
Project Director:	D.J.F. Smith	**Project Manager:**	

Signatories		
Production:		Bryan Richards **Senior Engineer**
Verification:		Dr. V. Roubos **Senior Mining Engineer**
Approval:		D.J.F. Smith **Mining Director**
Date:	March 2015	

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Web: www.DMT-Group..com

Table of Contents

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1 INTRODUCTION

DMT Consulting Ltd. (DMT) has been appointed by NIO to act as Lead Consultant to prepare a Technical Study (the Study) on the Blötberget Iron Ore Project (the Project). Much of the engineering work is to be carried out by a number of other consultants appointed directly by NIO.

This document provides the study basic parameters and basis for the engineering, operating and capital cost estimates and financial modelling.

1.1 Project Description

The Blötberget mine field extends 1.2 km, striking east-northeast at approximately 0600 and comprises several independent units named (from southwest to northeast) Kalvgruvan, Flygruvan, Hugget, Carlsvärdsgruvan, Sandell, Guldkannan and Fremundsberget.

The Blötberget mine closed in 1979 but was in operation since the early part of the 1900s producing lump, sinter fines and concentrates. The Kalvgurvan, Flygruvan and Hugget zones are mined down to the 240/350 m level. The units dip towards the southeast at between 600 - 550 in the mined-out areas, near-surface and flatten at depth to ~250.

A Preliminary Economic Assessment ("PEA") was undertaken by NIO, in November 2011. This was based on a project that combined the two existing Ludvika mines, Blötberget and Håksberget.

The mining method is planned to be a version of the previously used sublevel caving and open stoping methods.

The Ludvika orebodies contain both magnetite and hematite (martite) ores which may require differential processing.

Since the PEA, NIO changed the company strategy in order to focus initially on restarting operations at Blötberget as a standalone project, utilising as far as possible, the existing mine infrastructure from the previous mining operations.

Ore from Blötberget mine would be fed directly to the new concentration plant to be built at Skeppmora.

The run of the mine (ROM) ore from Blötberget is planned to be 3.0Mtpa when fully operational with an expected production of about 1.4 Mtpa of high quality finished iron ore products with an average iron content of 69%.

In January 2014, a new and independent resource estimate was prepared for NIO by Geovista AB.

The Mineral Resources will be updated as part of the Study but are anticipated to be in the region of 48Mt with an assumed head grade of approximately 37% Fe. Run-of-mine (ROM) ore production rate is to be 3 Mtpa.

1.2 Purpose of Document

This document provides the study basic parameters and basis for the engineering and cost estimation for the Study.

It presents a list of assumptions as to how the Study will be developed, the envisaged mining and processing operations, how concentrate products will be transported and delivered and financial modelling criteria.

These are to be used by the various members and consultants of the project team for the development of engineering required for the Study.

The following sections provide the agreed key criteria that will be used for the Study. Additional information and data will be generated for use in the Study by the various contributors.

As appropriate, these will be agreed with NIO and DMT and presented in the final report.

2 PROJECT DESIGN PARAMETERS

This section provides the design criteria and parameters that will be used as the basis for the technical study for the Blötberget Iron Ore Project.

Table 2-1: Project Design Parameters

Mine Design Basic Parameters			
Description	Units	Amount	Comments
Tonnes ROM	tonnes	3.0Mt/Annum	ROM Includes dilution excludes development rock
Tonnes Waste	tonnes	300,000t/Annum	Variable – depends on development schedule
Mine Life	14 Years	Variable	Determined by mineable reserve
Operating days per year	Days	350	
Days per week	Days	7	
No. of shifts per day	Shifts/day	3	Applied to all areas
Working Hrs per shift for mine	Hours/shift	8	Applied to all cases
Available hours per shift for mine	Hours/shift	7	Applies to mine
Total Available Hours	Hours/year	7,350	Applied to all cases (21hrs/day)
Nominal Tonnes per day ROM Ore	Tonnes/day	8,600 t ore	(14.3 hours/day at 600tph)
Nominal Tonnes per day waste	Tonnes/day	860 t rock	Waste rock depends on mine development and production schedule
Average Sp. Gravity Ore		3.8	(SG varies from 3,4 to 3,8 "to be determined from resource or reserve model")
Average Sp. Gravity Waste		2.7	
Mining			

Mine Design Basic Parameters			
Description	Units	Amount	Comments
Mining Method	Owner operated, Sub Level Open Stoping		
Crown Pillar	Nominal 10 vertical metres		
Mining Recovery (SLC)	%	95	
Mining Recovery (SLOS)	%	80	
SLC External Dilution	%	20	10% Fe content
SLC Internal Dilution	%	5	0% Fe content
SLOS External Dilution	%	10	15% Fe content
SLOS Internal Dilution	%	5	0% Fe content
Ramp gradient	%	14.3	(1:7) Typical
Surface Vehicle/Conveyor Ramp Width	Meters	10	Changes Section @ 304m Level
Surface Vehicle/Conveyor Ramp Height	Meters	5.5	Changes Section @ 304m Level
Internal Ramp (Vehicle)/Conveyor Width	Meters	6	
Internal Ramp (Vehicle)/Conveyor Height	Meters	5.0	
Internal Ramp Vehicle Width	Meters	5	
Internal Ramp Vehicle Height	Meters	5.5	
Main levels Width	Meters	6	
Main levels Height	Meters	5.5	
Cross cuts/production drives Width	Meters	6	
Cross cuts/production drives Width	Meters	5,5	
Mine Dewatering			
Requirements for mine process water supply	l/s	25 l/s	All operational machines
Mine water inflow	l/s	40	PEA
Permanent pump stations		1	Located @ 370mL
Estimated initial water volume to be pumped	Mm^3	5.0	PEA
Peak Mine Dewatering discharge	l/s	300	Environmental permit
Average Mine Dewatering discharge rate	l/s	150	Environmental permit
Primary Crushing and Conveying			
Annual Operating Hours	Hours	7,350	DMT
Ore Conveying Shifts	Shift	2	
Average nominal ore conveying capacity	Tonnes/hour	535	DMT
Additional peak flow	%	10	DMT
Crusher and conveyor Availability	%	95	DMT
Crusher and conveyor Utilisation	%	90	DMT

Mine Design Basic Parameters			
Description	Units	Amount	Comments
Peak Design Capacity	Tonnes/hour	690	DMT
Maximum feed size	mm	1000	
Maximum product size	mm	250	
Processing Plant			
Annual Hours	Hours/Year	8,400	
Plant Availability	%	92%	
Plant Utilisation	%	90%	
Plant Available Hours	Hours/year	6,955	TSC
Average Nominal Throughput	Tonnes/hour	431	TSC
Peak Allowance	%	10	TSC
Design Throughput	Tonnes/hour	475	TSC
Product production rate	Tonnes/hour	230	TSC (Variable)
Final Product Storage (ground & silo)	tonnes	10,000	Concentrate
Crushed Ore Stockpile Silo	tonnes	5,000	24 hours operation
Emergency ROM Ore Stockpile	tonnes	10,000	Mobile Plant Reclaim
Water Requirement	M^3/t ore	0.4	To be determined depending on option
Tailings disposal			
Capacity	Mt	33.0	(48Mtpa x .63 = 30Mt +10% design Factor)
Tailings Pumping(45% w/w)	tph	271	Nominal
Slurry Flow (45% w/w)	m^3/hr	440	Nominal (Max 585)
Water (45% w/w)	m^3/hr	331	Nominal (Max 450)
Rail Transport			
Nett train load	tonnes/train	2,500	Assumed Maximum for current rail specification
Conveyor Product Rate	tph	250	Variable
Load Out storage capacity	tonnes	5,000	Track side loading silo
Ground product storage	tonnes	5,000	Covered ground coverage with reclaim
Maximum waggons per train	No	40	
Maximum Axle Load	tonnes	25	
Maximum tonnes per metre rail	Tonne per metre	8.0	
Length of Passing track	m	650	
Permitted Operating Speed	Km/hr	40	NIO
Rail head to port	Km	270	
Port			
Size of Vessels	Kt	Variable	Panamax/Barge/Cape

Mine Design Basic Parameters			
Description	Units	Amount	Comments
Vessel Loading Rate	t/day	25,000	
Port Stockpile	kt	250	Ship loading stockpile 15k capacity.
Water			
Option 1 – Lake Väsman	litres/s	100	Any 4 months per year
Option 2 – Recycled process water	m3/hr	250	To be determined from water balance
Power			
Supplied power – HV overhead lines	kV	50	Power supplied by local utility
Maximum Capacity	MVA	40	
Underground Medium voltage	kV	12	
Underground utilisation voltage	V	3300/1000/415	
Cost Estimation			
Currency	US$		
Base Date of Estimate	31st March		
Accuracy of Estimate		+/-30%	
Structural costs	Based on US$/m2		
Major Process Plant & Equipment items	Based on database information and budget quotations available by Base Date of Estimate.		
Minor Process Plant items & Equipment	Costs for ancillary equipment, erection, piping, cabling, Civils and structural works determined by applying a percentage increment to relevant Major Process Plant Equipment costs.		
Local Costs	All in SEK		
EP related Costs	SEK	TBC	NIO
Land acquisition	SEK	TBC	NIO
Currency X/Rate SEK/USD	SEK/USD	TBC	Fixed, based on quoted rate at base date
Currency X/Rate Euro/USD	Euro/USD	TBC	Fixed, based on quoted rate at base date
Contingency	%	10	Applied to Mine & Process Operating costs
Contingency	%	10	Applied to Mine and Process Capital costs
Contingency	%	15	Applied to Infrastructure Capital costs
Energy	SEK/kWh	0.57	
Fuel	SEK/l	TBC	1st March
Water/Sewerage consumption cost	SEK/m^3	TBC	Municipal cost
EPCM cost	%	15	Applied to total installed capital cost
Financial			
Currency	US$		

Mine Design Basic Parameters			
Description	Units	Amount	Comments
Base Date of Model	31st March		
Currency X/Rate SEK/USD	SEK/USD	8.6232	Fixed, based on quoted rate at base date
Currency X/Rate SEK/Euro	SEK/Euro	9.2869	Fixed, based on quoted rate at base date
Currency X/Rate Euro/USD	USD/Euro	1.0770	Fixed, based on quoted rate at base date
Currency X/Rate SEK/GBP	SEK/GBP	12.7441	Fixed, based on quoted rate at base date
Product Price	$/t	TBC	NIO/DMT
Real Discount Rate	%	8	NIO/DMT
Royalty	%	0.2	FOB
VAT Rate	%	25	NIO
Corporate Tax	%	22	NIO

3 SURFACE INFRASTRUCTURE & BUILDINGS

The surface infrastructure data listed below outlines the details of the proposed mine site to support the planned mining operations;

- Mine Complex access,
- Mine Infrastructure Support operations,
- Mine buildings, utilities and services,
- Product Storage and Mine Waste,
- Power and communications and
- Mine Water Management.

The surface infrastructure components required to sustain these operations are summarised in the table below and should be identified on the surface plan including all logistical services and access routes. The information requirement is indicated in the respective column or if no information is required.

Table 3-1 Surface Infrastructure Requirements

Blötberget Surface Infrastructure		
Description/Component	Requirement	Comment/Requirement
Mine Complex Access		
Access by Air	X	N/A
Access by Road	√	Axle Load
Access by Rail	√	New Switch facility
Port Facilities	√	TBC

Blötberget Surface Infrastructure		
Description/Component	Requirement	Comment/Requirement
Transport Facilities	√	Public Transport/Parking
Mine Infrastructure Support Operations		
Site & Location (Plan & General Arrangement)	√	Industrial site 1 & 2
Surface or Underground	√	Surface
Open Pit	X	N/A
Haul Roads	√	Industrial site 1 & 2
Bunkerage/Storage	√	Industrial site 1 & 2
Mine Buildings, Utilities and Services		
Accommodation	X	No onsite accommodation
Welfare Facilities	√	Mess & Changing
Storage Facilities	√	Process & mining Equipment spares/process consumables and reagents/fuel, oils and greases
Administration Facilities	√	Management
Maintenance Facilities	√	Workshops
Materials Handling	√	Stores/Supplies
Plant & Equipment	√	Mobile Plant
Compressed Air	√	Mobile Compressors
Explosives & Storage	√	No surface storage
Stockyard	√	Underground Supplies
Workshops	√	Mechanical/Electrical
Fuel Storage Distribution	√	Surface & Underground
Product Storage and Mine Waste		
Product Storage Capacity	√	Storage/Loading facility
Waste Storage Capacity	√	Industrial site 2
Transport Plan	√	Industrial site 1 & 2
Security/boundary fencing	√	Industrial site 1 & 2
Power and Communications		
Final Power Supply	√	HV Supply
Plant Electrical Distribution	√	Surface Sub-Station
Lighting	√	Industrial site 1 & 2
Communication Systems	√	Surface
Mine Water Management		
Hydrological Data	√	Water Table
Surface Drainage	√	Discharge Licence

Blötberget Surface Infrastructure		
Description/Component	Requirement	Comment/Requirement
Water Supply (Potable & Process)	√	Extraction Licence
Water Course, Diversions & Drainage	√	EIA
Quality	√	Discharge consent
Pumping and Settlement Lagoons	√	EIA

4 COST ESTIMATE AND FINANCIAL MODELLING

The capital costs to bring the mine into production and the operating costs of production will be compiled by DMT with input from others responsible for their sections of work.

All expenditure up to commencement of commercial production will be capitalised.

Sustaining capital for maintaining production and periodic replacement for all major items of equipment will be estimated. Sustaining capital will include mine capital development such as main ramps and ore and air raises.

Costs of stope development will be included in mine operating costs.

Cash Operating costs will include mine operating costs, processing costs, mine site G & A costs (but excluding all off site costs, royalties and taxes) and will be expressed as US$/tonne ore and US$/tonne product

Total Cash Costs will include Cash Operating Costs plus all off site costs, royalties and taxes

A pre-tax discounted cash flow model will be prepared by DMT based on the mineable inventory and the life of mine plan for the Project. The DCF will be prepared on 100% equity basis and will not include costs of debt or financing arrangements. No working capital will be included in the DCF.

Revenue from sales will be recognised at the time of production and it is assumed that the model will be on be on a VAT-neutral basis; that is, any VAT paid in a particular year is returned to the company within that year.

Appendix E

MINERAL RESOURCE ESTIMATE REPORT

DMT Consulting Ltd
International Mining Consultants

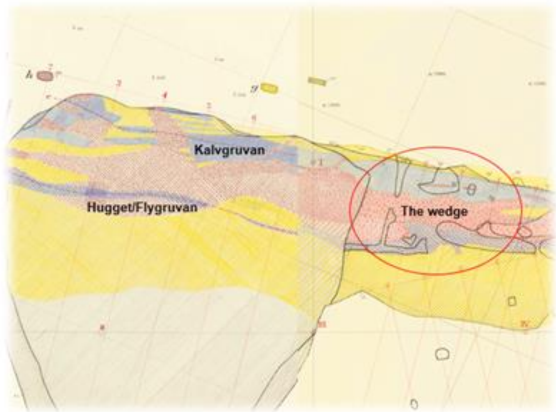
Mineral Resource Estimate

For the

Blötberget Iron Ore Project, Ludvika, Sweden

On behalf of

Nordic Iron Ore™



Prepared:	**Florian Lowicki** *Pr.Sci.Nat Geol. (SACNASP)* *Resource Geologist*	
Reviewed:	**Tim Horner** *CGeol CEng P.Geo. (APGO)* *Principal Geologist*	
Approved:	**David JF Smith** *CEng* *Director of Mining*	

Effective Date of the Report: 10th April 2015

Date of Signing: 10th April 2015

Document Ref.: C22-R-126

Earth. Insight. Values.

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1 EXECUTIVE SUMMARY

1.1 Introduction

DMT Consulting Ltd. (“**DMT**” or “**the Consultant**”) was retained by Nordic Iron Ore (“**NIO**” or ”**the Client**”), to prepare an independent Mineral Resource Estimate (“**MRE**” or “**the Study**”) for the Blötberget Iron Ore Project (“**the Project**”), located near Ludvika, Sweden.

The purpose of this report is to update the Mineral Resource Estimate for the Project.

The estimation of Mineral Resources has been prepared in compliance with the guidelines set out in the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012.

NIO is a mining company aiming to reopen the main Ludvika mines - Blötberget and Håksberg, and resume iron ore production.

1.2 Technical Summary

1.2.1 Property Description & Location

The Blötberget Project is situated in Dalarnas County in central Sweden, approximately 500 m south east of the village of Blötberget, and near to the town of Ludvika.

The Project region is known as the Bergslagen District, famous for its very long mining and steelmaking history, with notable former and current production areas within this region.

1.2.2 Land Tenure

NIO currently holds 12 exploration permits, which together cover an area of 3,044.36 hectares. NIO also holds two mining concessions - Blötbergsgruva K nr 1 and Håksbergsgruva K nr 1, covering an area of 262.7 ha. All areas, besides those covered by the Väsman concession and parts of the Håksberg concessions are so called “brownfield” sites and have previously been worked and contain abandoned mines.

NIO applied for a mining concession within the Blötberget area in October 2010 and it was granted in August 2011. The mining concession, which runs for 25 years with possibility of extension, implies the right of exploitation and utilisation of iron, rare earths, and apatite. The environmental permit for this Project was granted in June 2014.

1.2.3 Local Infrastructure

Blötberget is located 2.5 km west of Route 50 and is directly accessible along well maintained asphalt roads.

Blötberget does not have its own railway station, however the main Gotenborg to Gävle line lies 2.5 km directly to the east of the Project.

The closest large town to Blötberget is Ludvika, which is located 12 km north east and has a population of approximately 14,500.

1.2.4 Climate & Physiography

The climate at the Project is classified as cold and temperate (sub-arctic or boreal), characterised by short, cool summers and long cold winters. Average annual precipitation is 713 mm and the average annual temperature is 4.6 °C.

The Project is located in an area dominated by arboreal forest. Generally, the local terrain consists of gently undulating hills, except for the area around Blötberget, which is predominantly flat and marshy. Elevation in the Project area varies between 150 and 250 m above sea level.

1.2.5 History

Mining and exploration in the Ludvika area has been active since the 1600's. The majority of this small scale mining was focused on iron production.

Blötberget originally operated as two separate mines from the early 1900s, the German owned Vulcanus "original" mine and the Swedish owned Blötberget "new" mine.

Blötberget started operations in 1944. After the Second World War, in 1949, the Vulcanus mine regained Swedish ownership and continued production until the mine closed in 1979. Since 1979, the deposit has been controlled by several companies through exploration leases, until NIO was formed in 2008.

Airborne and ground-based geophysical surveys over the Project area were carried out in the 1960s and 1950s respectively.

From 1942 until 1977, the deposits were systematically diamond drilled for definition and extension.

Most of the drillholes (~80 %) were collared from underground. In areas where historic mining was on-going or planned, regular 'fan' drilling was carried out. Longer holes were drilled from underground positions in cross cuts from the hanging wall, with only the drillholes probing the deeper, down dip parts of the deposit, drilled from surface.

A total of 391 drillholes have been drilled historically at Blötberget, totalling 32,751 m.

The mining company, Stora Kopparbergs Bergslags AB submitted a closure report to the Inspector of Mines at the cessation of mining activities in 1979. The 'reserves' (non-compliant) at that time were estimated to be 25 Mt at an average grade of 43 % Fe.

The final production achieved in 1979 at Blötberget was 400 Ktpa (thousand tonnes per annum).

The process plant handled up to a maximum of 415,000 tonnes of feed material per annum. Large changes in the proportion of magnetite/hematite concentrate are noted in the production over the five year period ahead of closure in 1979, and reflect the variability of the ore composition with respect to hematite and magnetite. Historically the recovery has varied between 76.3% and 85.6%. Mineral recovery appears to be greater when magnetite percentages are higher.

1.2.6 Geology

The Blötberget apatite-iron oxide deposit is located in the western part of the intensely mineralised Paleoproterozoic Bergslagen Province in south central Sweden.

The Province is dominated by several generations of intrusive rocks, which enclose inliers of metasedimentary and metavolcanic rocks. The metasedimentary and metavolcanic inliers are of great importance as they host an overwhelming majority of the more than 6,000 known metallic mineral deposits and prospects in the Province. These rocks have been subjected to multiple-phase deformation and metamorphism under mainly greenschist to amphibolites facies conditions.

The host rocks to the Blötberget iron mineralisation have traditionally been classified as belonging to the ”leptite formation”, i.e. mainly felsic to, more rarely, intermediate, regionally metamorphosed (c. 1.90–1.87 Ga) volcanic rocks.

The mineralisation at Blötberget is a so-called "apatite lake ore" which, besides the iron minerals magnetite and hematite, also contains the phosphorus mineral apatite.

The mineralised zone at Blötberget appears as a set of vertically narrow, elongated lenses dipping 50°–70° to the SE. Airborne geophysical surveys and historical drillholes indicate that mineralisation extends to a depth of at least 900 m below surface.

The Blötberget field consists of five mineralised bodies, from west to east, these are:

- Kalvgruvan;
- Flygruvan;
- Hugget & Betstamalmen; and
- Sandellmalmen.

The Blötberget deposit is referred to as a Kiruna type deposit although the exact origin is still disputed.

1.2.7 Exploration & Drilling

2009

At the start of NIO’s ownership of the Project in 2009, Kopparberg Mineral AB carried out a detailed magnetometry survey over a limited part of the Blötberget area to assist with defining the geometry of the mineralisation and planning further exploration works.

2011

During 2011/12 Berg och Gruvundersökningar AB ("**BGU**") was engaged by NIO to log and sample historical archive cores that were stored at the SGU repository in Malå.

13 cores, totalling 5077.21 m were logged for geological and geotechnical data (for RQD), then photographed (dry and wet).

2012

In 2012, a 16 hole drill programme which included twinned drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling was completed by NIO.

Drilling for this programme totalled 7,430 m.

One hole, BB12015-MET, was drilled for the purposes of generating material for metallurgical sampling. To date, this has been the only hole drilled using oriented core.

No drillholes were water pressure tested during this drilling campaign.

2014

The 2014 NIO drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget and to infill the intermediate depth extension of Hugget in order to improve the confidence of the geological model.

13 drillholes, totalling 7,093 m, were drilled and one of the drillholes (BB_14-011) was drilled down-dip for geotechnical purposes. All holes were drilled with orientation information, and eight holes were subject to pump testing to provide information on the potential for water bearing fracture zones.

1.2.7.1 Logging

The core from the 2012 and 2014 drill programmes was logged and sampled at NIOs preparation facilities in Grängesberg. The core storage, logging and sampling facilities were inspected by DMT and found to be clean, well-organised and provide suitable conditions for logging and sampling.

The core is inspected for mislabelling or depth inaccuracies, checked using a magnetic pen and subject to a UV lamp to detect presence of Scheelite (tungsten).

The cores orientation is then marked and the geologist logs the core in accordance with NIOs geological logging template and in accordance with industry standard procedures.

Core photography, point load testing, drill core structure orientation, geotechnical logging and classification complete the logging procedure.

1.2.8 Sampling & Analysis

The core from the 2012 and 2014 drill programmes was collected in the field by NIO's technicians or geologists and transported directly to the NIO core logging and sample preparation facilities in Grängesberg.

NIO compiled its own Sampling Quality Manual, which sets out procedures in accordance with industry best practice, relating to core handling, sampling, analysis and QA/QC procedures

Analyses for the 2012 and 2014 samples was carried out by ALS Global in Vancouver.

As part of the verification program, NIO has re-logged and re-assayed many of the located historical cores. In total, 45 drillholes from Blötberget were found in Malå and 15 at the former mine storage facility in Håksberg.

There has been re-logging of 31 of these cores (6036 m), 950 m of mineralisation has been re-sampled and re-assayed according to current industry practice and standards. Recent re-analysis of the historical holes indicate that Fe has been under represented but is a consistent <15 Fe % error which means there is reasonable confidence levels in the historical analysis

1.3 Mineral Resources

DMT applied only geological constraints and a grade of ~+15 % Fe to establish the mineralised wireframes and solids. Based on analysis of the available data, the internal waste and country rock was assigned a grade of 8 % Fe (total).

The total 'global' (geological) Measured and Indicated Resources for the Blötberget Project are estimated at 61.3 Mt at a grade of 36.6 % Fe (Total) and 0.5 % P.

DMT subsequently applied preliminary mining and economic parameters and assumptions to the geological wireframe model to estimate a preliminary cut-off grade ("**COG**").

The total Measured and Indicated Resources estimated for the Blötberget Project are 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P at a COG of 25 % Fe (Table 1-1). Of these Measured and Indicated Resources, 62 % is magnetite and 38 % is hematite.

Table 1-1 Measured and Indicated Resources for the Blötberget Iron Project - January 2015

Fe Cut-off % Fe	Resource Category	Volume Mm^3	Tonnage Mt	Density t/m^3	Fe %	Magnetite %	Hematite %	Magnetite proportion %	Hematite proportion %	Phos. %
25	Measured	11.1	42.5	3.8	41.9	36.8	21.9	0.63	0.37	0.51
	Indicated	1.4	5.3	3.7	38.2	30.5	23.2	0.57	0.43	0.5
	Measured + Indicated	**12.5**	**47.8**	**3.8**	**41.5**	**36.1**	**22.0**	**0.62**	**0.38**	**0.51**
	Inferred	1.5	5.4	3.5	33.5	23.5	23.5	0.50	0.50	0.52

Notes:

1) JORC 2012 definitions were followed for estimating Mineral Resources;
2) Mineral Resources are estimated at a cut-off grade of 25 % Fe;
3) Mineral Resources are estimated using a five year historical average price of US$ 100 per tonne (Source: IndexMundi.com); and
4) Figures may not total due to rounding errors.

1.4 Conclusions

The Blötberget apatite-iron oxide deposit is located in the western part of the intensely mineralised Paleoproterozoic Bergslagen Province in south central Sweden.

The deposits in the neighbouring area occur along a ~40 km long, broad zone. This zone of mineralisation is the third largest iron ore deposit in Sweden by production, only outnumbered by the giant Kirunavaara and Malmberget iron ores in Norrbotten, northern Sweden.

Airborne geophysical surveys and historical drillholes indicate that mineralisation extends to a depth of at least 900 m below surface.

The Blötberget field consists of five mineralised bodies, from west to east, these are: Kalvgruvan ("**KALV**"); Flygruvan ("**FLY**"); Hugget ("**HUG**") and Betsta ("**The Wedge**"); Sandell ("**SAND**").

NIO applied for a mining concession within the Blötberget area in October 2010 and it was granted by the Mining Inspectorate of Sweden in August 2011. The mining concession, which runs for 25 years with possibility of extension, implies the right of exploitation and utilisation of iron, rare earths, and apatite. The environmental permit for this Project was granted in late June 2014.

The Hugget and Kalvgruvan/ Flygruvan zones had previously been mined out from near-surface to the 200 m and 240 m levels respectively. The units dip towards the southeast at between 50° and 55° in the near-surface mined-out areas, and flatten at depth to ~25°.

A drilling programme was undertaken by NIO during the summer and winter of 2012 and was completed in November 2012. This 16 hole programme included drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling. NIO completed 16 drillholes totalling 7,430 m of drilling.

The 2014 drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget (formally known as "the Wedge" or Betsta area) and to infill the intermediate depth extension of Hugget in order to improve the confidence of the geological model. 13 drillholes, totalling 7,093 m, were drilled.

The Wedge was successfully explored during the 2014 drilling programme and, as a result, Kalvgruvan and Hugget/Flygruvan ("**HUG/FLY**") have now been shown to form continuous zones of mineralisation.

Mine maps and historical drilling data have been collected from various sources and digitised, where possible. Drill core from historical exploration drilling in the Blötberget project area has been recovered, re-logged and re-analysed.

DMT was provided with a comprehensive set of historical reports and data which have been collated and used in conjunction with data collected more recently by NIO in order to estimate and report Mineral Resources for the Blötberget Project in accordance with JORC standards.

In the resource development programme of 2012 and 2014 NIO completed industry standard QA/QC programmes to ensure the data is reliable and suitable for resource estimation. The drill density of the resource is adequate for the purpose and is reflected in the JORC compliant resource category classifications of Measured, Indicated and Inferred Mineral Resources.

DMT has relied heavily upon the information provided by NIO, however DMT has, where possible, verified data provided independently during the site visits.

DMT was able to overlay license information on the Mineral Resource estimate area to confirm that the deposit is within NIO's license. DMT has not undertaken a legal review of the licences and assume that all the required licences are in place.

The geology of the deposit is fairly well understood and DMT has constructed a wireframe geological model for the Blötberget deposit based upon a combination of logged lithologies, analytical and Saturation Magnetisation Analyser ("**SATMAGAN**") magnetite results, which has allowed the splitting of the deposit into geological domains comprising of magnetite-rich material of KALV and hematite-rich material of HUGFLY and SAND.

DMT has undertaken a statistical study of the data, which demonstrates adequate splitting of the data into single iron population domains, and undertaken a geostatistical study to investigate the grade continuity and to provide grade estimation parameters for Ordinary Kriging.

Geovia Surpac solid and block models were created using all of the available geological and sample analytical test data has defined an iron ore resource. At this stage of the investigation most of the mineral resources of Blötberget have been classified into the Measured and Indicated categories.

DMT has estimated the total Measured and Indicated Resources for the Blötberget Project to be 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P at a preliminary COG of 25 % Fe. The magnetite-hematite ratio of the total resource is 62:38.

1.5 Recommendations

For the current MRE it is considered that there is only limited additional geological information that can be gained from further, expensive, surface based drilling programmes. The bulk of the upper levels of the Blötberget deposit that have been identified as part of the proposed mine plan are within the Measured Resource category, therefore the confidence in the model overall would not benefit from more drilling.

However, surface drilling for rock mechanical / structural and or metallurgical information for detailed mine planning should be considered. There is an indication in the current drillhole information and geological level mapping that there may be structural (fissures, joint and /or faults) that exist in the 'Wedge' area and vicinity that potential may impact on the rock quality and hydrogeology locally.

Further hydrogeological investigations on existing drill holes should be undertaken, it is considered that insufficient data exists on the hydrological and hydrogeological conditions for underground mining.

Definition and grade control drilling should commence as soon as there is access to the underground areas after dewatering. This close spaced drilling is required to support the transfer of Measured Resources into (Proven) Ore Reserves. The underground based drilling should follow a similar approach to that used historically, with a fan pattern of close spaced drilling into the mine blocks. Wider spaced and deeper, down – dip, drilling collared from hanging wall positions to provide increased confidence in the areas containing Indicated Resources.

2 INTRODUCTION

DMT Consulting Ltd. ("**DMT**" or "**the Consultant**") was retained by Nordic Iron Ore ("**NIO**" or "**the Client**"), to prepare an independent Mineral Resource Estimate ("**MRE**" or "**the Study**") for the Blötberget Iron Ore Project ("**the Project**"), located near Ludvika, Sweden.

The purpose of this report is to update the Mineral Resource Estimate for the Project.

The estimation of Mineral Resources has been prepared in compliance with the guidelines set out in the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012 ("**the JORC Code**").

2.1 Nordic Iron Ore

NIO is a mining company aiming to reopen the main Ludvika mines - Blötberget and Håksberg, and resume iron ore production.

Deposits are located along an approximately 25 km long vein field of iron-rich deposits that run from Blötberget in the south to the north section of the Håksberg field. For the first time in history, this mineralised field is controlled by a single stakeholder, NIO, through a total of nine exploration permits and two mining concessions.

2.2 Terms of Reference

DMT has been retained by NIO to prepare a full technical report on the current status and development of the Blötberget Iron Ore Project. In order to fulfil the requirements of the technical report, an updated Mineral Resource Estimate is required in accordance with an internationally accepted Mineral Resource reporting code. This MRE will form a standalone report, extracts of which will be made available for inclusion in the final technical report.

The previous MRE for the Project was prepared by Thomas Lindholm (GeoVista AB) in January 2014, entitled 'Technical Report: Blötberget – Mineral resource estimate'.

Since the issuance of this report, NIO has carried out a new drilling campaign at Blötberget, re-analysed a number of samples from historic drill core and carried out additional mineralogical and metallurgical analyses, thus marking a 'material change' in the status of the project. This Study will update the Mineral Resources to take account of the new data available.

2.3 Sources of Information

Initial site visits were carried out by DMT Principal Geologist , Mr Tim Horner CGeol P.Geo. on 14/08/2014 and 15/08/2015 in which the core storage and logging facilities in Grängesberg were inspected. Several drill sites were observed and some surface (historic open pit) locations were viewed where it was proposed to recover bulk sample material.

Discussions were held with senior technical personnel from NIO.

Mr Florian Lowicki Pr.Sci.Nat Geol. (400425/13; SACNASP), DMT's Resource Geologist, visited the NIO Ludvika site offices on five separate occasions between September 2014 and January 2015 to review the data acquisition procedures applied to the drilling programme and the database. Technical discussions relating to the on-site mineralogical testing and the geological model were held with Thomas Lindholm (GeoVista), Michael Setter and Emma Bäckström (NIO Geologists).

The individuals responsible for this report have extensive experience in estimating and evaluating mineral resources and are members in good standing of appropriate professional institutions and hence are Qualified Persons ("**QPs**") under the terms of JORC.

Neither DMT nor any of its employees and associates employed in the preparation of this report have any beneficial interest in NIO or in the assets of NIO. DMT will be paid a fee for this work in accordance with normal professional consulting practice.

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 14 - References.

2.4 Units & List of Abbreviations

Units of measurement used in this report conform to the metric system. All currency in this report is stated in United States Dollars ("**USD**") unless otherwise noted.

Table 2-1 List of abbreviations

Abbrv.	Description	Abbrv.	Description
°	degrees	m^2	square metre
°c	degrees Celsius	m^3	cubic metre
%	percent	m^3/hr	cubic metres per hour
<	less than	m^3/t	cubic metres per tonne
>	greater than	Ma	million years
attrib.	attribute	mag	magnetite
BCM	bank cubic metres	magsus	magnetic susceptibility
BGU	Berg och Gruvundersökningar AB	masl	metres above sea level
CAPEX	capital expenditure	mm	millimetre
cm	centimetre	MOP	mine operation period
COG	cut-off grade	MRE	mineral resource estimate
Conc	concentrate	Mt	million metric tonnes
DGRF	definitive geomagnetic reference field	Mtpa	million metric tonnes per annum
DMT	DMT Consulting Limited	m/min	metres per minute
DMT	dry metric tonnes	m/s	metres per second
DSCO	drill core structure orientation	N	north
DTM	digital terrain model	NIO	Nordic Iron Ore
E	east	OK	ordinary kriging

Abbrv.	Description	Abbrv.	Description
EIA	environmental impact assessment	OPEX	operating expenditure
ESIA	environmental & social impact assessment	OREAS	Ore Research and Analysis Australia
EMP	environmental management plan	P	phosphorous
Fe	iron	QA/QC	quality assurance / quality control
FLY	Flygruvan	REE	rare earth element
g	gram	ROM	run of mine
Ga	billion years	RQD	rock quality designation
GPS	global positioning system	RTK	real time kinetic
ha	hectare	S	south
HCl	hydrochloric acid	S	sulphur
HUGFLY	Hugget-Flygruvan	SAND	Sandell
ICP-AES	inductively coupled plasma–atomic emission spectroscopy	SATMAGAN	saturation magnetisation analyser
IEC	International Electrotechnical Commission	SEK	Swedish Krona
IOCG	iron oxide copper gold (deposit)	SG	specific gravity
ISO	International Organisation for Standardisation	SGU	Swedish Geological Survey
JORC	Joint Ore Reserves Committee	SiO_2	silica dioxide
KALV	Kalvgruvan	SOP	standard operating procedure(s)
kg	kilogram	TGA	thermo gravimetric analyser
km	kilometre	t/m^3	tonnes per cubic metre
km^2	square kilometres	tonnes	metric tonnes
ktpa	kilo (1,000) metric tonnes per annum	ToR	terms of reference
kV	kilovolt	US$	United States Dollar
LIMS	low intensity magnetic separation	UV	ultra violet
LOI	loss on ignition	W	west
LOM	life of mine	XRF	x-ray fluorescence
m	metre		

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by DMT, for NIO.

The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to DMT at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by NIO and other third party sources.

For the purpose of this report, DMT has relied on ownership information provided by NIO. DMT has not researched property title or mineral rights for the Project and expresses no opinion as to the ownership status of the property.

4 PROPERTY DESCRIPTION & LOCATION

4.1 Location

The Blötberget Project is situated in Dalarnas County in central Sweden, approximately 500 m south east of the village of Blötberget, and near to the town of Ludvika. Sweden's largest city, Stockholm, is located to the South East, within driving distance along Route 66 and E18 (228 km). The country's second largest city, Göteborg, to the South West is within driving distance along Route 50 and E20 (400 km).

The Project region is known as the Bergslagen District, famous for its very long mining and steelmaking history, with notable former and current production areas within this region including the Grängesberg iron ore mine, Zinkgruvan sulphide mine, Garpenberg sulphide mine and Falun sulphide mine. Ludvika is located at the southern shore of Väsman lake at an elevation of around 157 m above sea level ("**masl**").

Figure 4-1 **Location Map**

Source: Google Maps

4.2 Land Tenure

The Blötberget area was historically divided into six fields based on the old mining concessions, namely:

- Kalvgruvan;
- Flygruvan;

- Betsta;
- Hugget;
- Sandell; and
- Guldkannan.

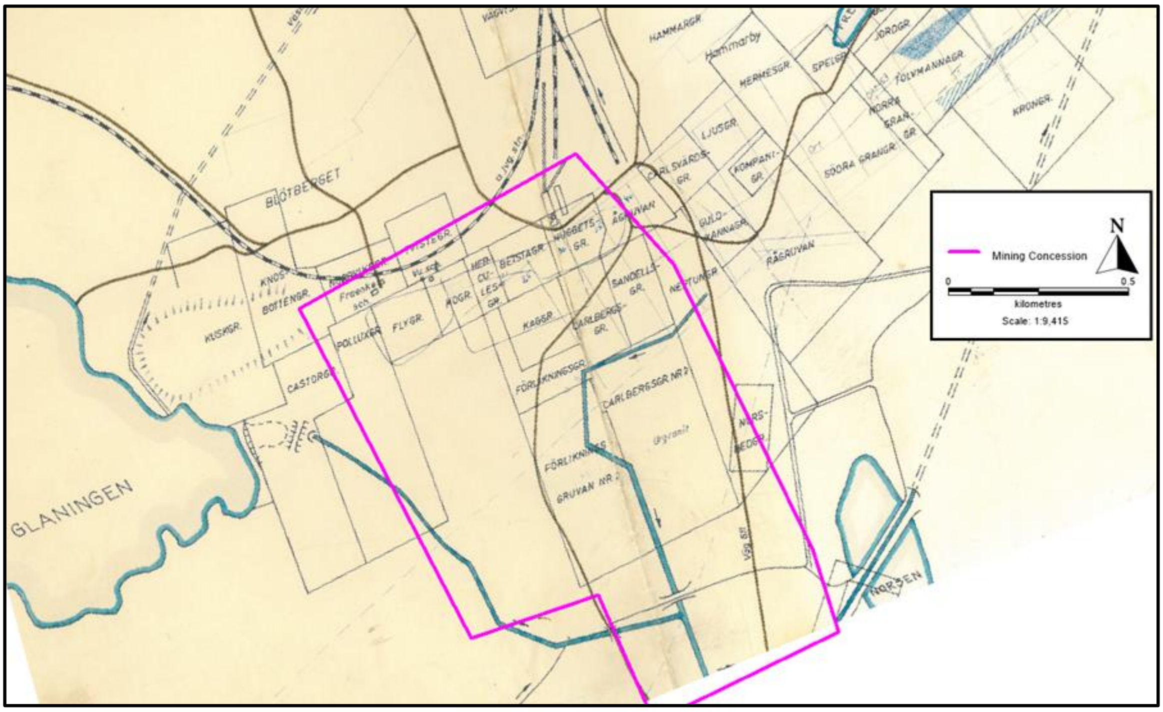


Figure 4-2 **Historical and current mining concessions - Blötberget**

Source: NIO

NIO currently holds 12 exploration permits, which together cover an area of 3,044.36 hectares (“**ha**”). NIO also holds two mining concessions - Blötbergsgruva K nr 1 and Håksbergsgruva K nr 1, covering an area of 262.7 ha. All areas, besides those covered by the Väsman concession and parts of the Håksberg concessions are so called “brownfield” sites and have previously been worked and contain abandoned mines.

NIO applied for a mining license within the Blötberget area in October 2010 and was granted an application by the Mining Inspectorate of Sweden in August 2011. The mining concession, which runs for 25 years with possibility of extension, implies the right of exploitation and utilisation of iron, rare earths, and apatite. The environmental permit for this concession was granted in late March 2014. The licence locations and descriptions are shown in Table 4-1 and Figure 4-3.

In addition to the one exploitation concession, NIO has four exploration licenses within the Blötberget area.

Table 4-1 Mining and Surface Licences

Deposit / Area	Concession / Licence Number	Expiration Date	Area ha	Concession / Licence Points	Northing SWEREF99 TM	Easting SWEREF99 TM
Blötbergsgruva K nr 1	2010001141	30/08/2036	126.4	Mining concession for Iron, Apatite and Lanthanum	6664955.321	503339.990
Blötberget nr 1	2007000066	29/05/2015	255.85	Exploration concession for Iron	503218.911	6665117.675
Blötberget nr 2	2007000114	07/06/2015	421.25	Exploration concession for Iron	502770.639	6664721.268
Blötberget nr 3	2010000564	16/06/2015	215.7	Exploration concession for Iron	501545.841	6664697.950
Främundsberget nr 1	2008000970	25/09/2015	156.03	Exploration concession for Iron	504658.801	6665047.721

Source: NIO

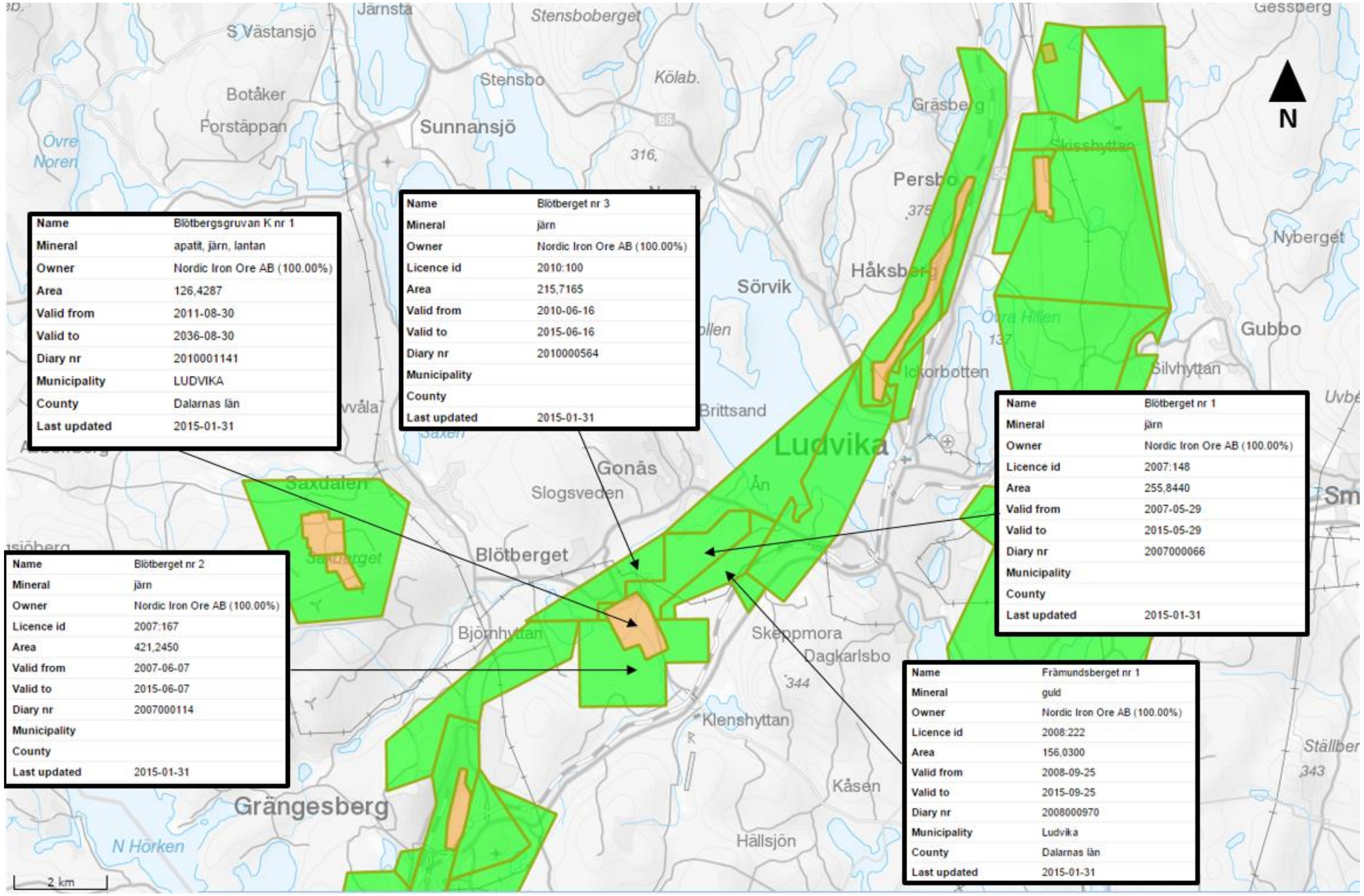


Figure 4-3 **Licences Map**

Source: NIO

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Accessibility

Blötberget is located 2.5 km west of Route 50 and is directly accessible along well maintained asphalt roads.

5.2 Climate

The climate at the Project is classified as cold and temperate (sub-arctic or boreal), characterised by short, cool summers and long cold winters.

There is significant precipitation throughout the year, with an average annual precipitation of 713 mm. The driest month is March with an average of 35 mm of precipitation. Most precipitation falls in August, with an average of 87 mm for the month.

The average annual temperature is around 4.6 °C. The warmest month of the year is July with an average temperature of 16.1 °C. The coldest month is February, when the average temperature is -5.6 °C. The average temperature fluctuation throughout the year is 21.7 °C.

The cold climate at the Project site has the potential, but is unlikely to, affect surface operations during the winter months, should the mine become operational.

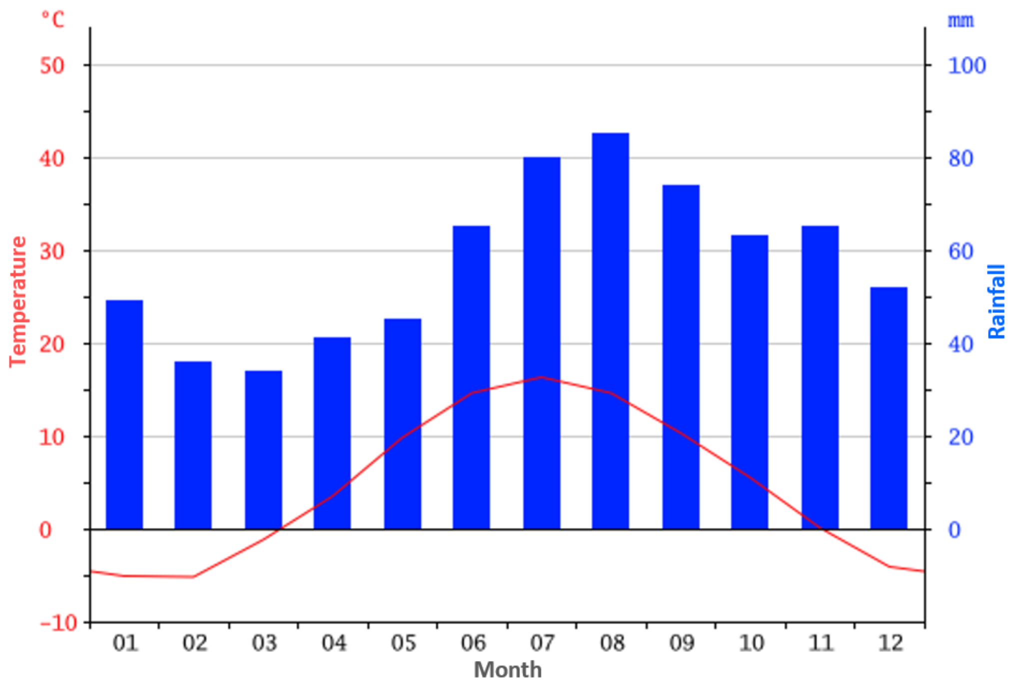


Figure 5-1 **Climate data (Ludvika)**

Source: http://en.climate-data.org

5.3 Local Resources

The closest large town to Blötberget is Ludvika, which is located 6 km north east along Route 50. Ludvika has a population of approximately 14,500 and population density of 15.5/km^2 as of 2012 (*Urbistat.it, 2015*). Ludvika offers general services including medical care, telecommunications, banking, housing, hotels, vehicle repair and schooling.

The local community has a 7.8 % unemployment rate, lower than the Swedish average of 8.0 % (*Ekonomifakta, 2014*). The multinational engineering company ABB has manufacturing facilities in Ludvika for power transformers, capacitors and equipment for power transmission. ABB, as well as Sandvik and Atlas Copco are major employers in the area.

5.4 Infrastructure

5.4.1 Road

The national highway, Route 50, which runs north-south, passes close to the Project.

5.4.2 Rail

Blötberget does not have its own railway station, however the main line lies adjacent to the Project. This line is the main Northern Swedish railway from Gotenborg to Gävle which primarily runs parallel to route 50. There is also a railway station in Grängesberg, 9.7 km to the South West of Blötberget.

The railway that passes through Ludvika and close by the Project, offers connections to three port towns/cities namely; Gävle (180 km) and Oxelösund (270 km) at the Baltic Sea and Lysekil (410 km) on the Swedish west coast.

5.4.3 Air

The nearest airport with domestic flights to and from Arlanda International Airport is Dala Airport, located in the neighbouring town Borlänge, approximately 55 km northeast of Ludvika.

5.4.4 Power

The electrical power required for mining and milling operations will be sourced from the main power line (50 kV), operated by VB-Kraft, which lies approximately 1 km from Blötberget, but passes into the planned industrial area.

5.4.5 Water

Water for the industrial areas and process plant can be sourced from nearby lakes.

5.5 Physiography

The Project is located in an area dominated by arboreal forest.

In general, the local terrain consists of gently undulating hills, except for the area around Blötberget, which is predominantly flat and marshy. The elevation in the Project area varies between 150 and 250 masl.

5.6 Surface Rights

The agreements fully in place are those with the landowner of the water rights for the mine (for dewatering), necessary before the mining concession rights are granted.

Access has been agreed with the landowner to allow the building of the sedimentation ponds, required for dewatering.

Currently, NIO does not have final agreement for all the industrial areas or the tailings dams. However, discussions and agreements are pending commitments for further investment and advancement of the Project.

Importantly, all the arrangements for the development of the project and the surface rights are now covered by the Mining Laws. In the event of disputes, the Government representatives of the mining law can either arbitrate an agreement or, in extreme cases, expropriation can be enforced if agreement cannot be achieved through negotiations.

6 HISTORY

6.1 Previous Ownership

Mining and exploration in the Ludvika area has been carried out in different periods since the 1600's. The majority of this small scale mining was focused on iron production.

Blötberget originally operated as two separate mines from the early 1900s, the German owned Vulcanus "original" mine and the Swedish owned Blötberget "new" mine. Each operated with separate hoisting shafts between 1950 and 1966.

Blötberget started operations in 1944 by sinking the new shaft to the 300 m level and building the new Central Plant. After the Second World War, in 1949, the Vulcanus mine regained Swedish ownership under Stora Kopparberg and continued production until the mine closed in 1979. Since the mine closed in 1979, the deposit has been controlled by several companies through exploration concessions, until NIO was formed in 2008.

1900	Bergverks AB Vulcanus starts large-scale mining operations
1944	Stora Kopparberg Bergslags AB begins preparations for mining in the nearby mining district. The Bergslag shaft (BS) is sunk and the modern industrial site is established
1949	SStora Kopparberg Bergslags AB buys the Vulcanus mine from Flyktkapitalbyrån (Flight Capital Agency)
1950–1966	The mining area is integrated, both the Vulcanus mine and Bergslag shafts are utilised. Annual production reaches about 400 kilo tonnes of crude ore and 220 kilo tonnes of dressed ore products
1968–1975	The BS shaft is sunk to the 570 metre level, the BS ore skip is upgraded to an annual capacity of 600 kilo tonnes and the new plant comes into operation in December 1975
1978	SSAB is formed and Stora Kopparberg Bergslags AB hands over the Blötberget mine
1979	Mining operations cease
1980	Permits and mining rights are returned to the state
2007	New exploration permits are applied for and awarded
2008	**The permits are transferred to Nordic Iron Ore**

Figure 6-1 History of Blötberget

Source: www.nordicironore.se

Figure 6-2 **Historic workings (for Section A-B see Figure 6-6)**

Source: NIO

6.2 Exploration

6.2.1 Airborne Geophysical Surveys

The Geological Survey of Sweden (“**SGU**”) performed regional airborne geophysical surveys over the area. In the 1960’s, an airborne magnetometry and gamma spectrometry survey was completed. This was carried out with 250 m line spacing at a height of 30 to 60 m. The resultant map shows measured variations in the magnetic total field after the Earth's magnetic reference field (DGRF 1965.0) was subtracted. . The map provides information on lithological variations and structures in the bedrock at the surface and at depth. Shifts in anomaly pattern can detect faults and their relative movements. The information has been used for geological mapping and prospecting, and is particularly useful in areas where large parts of the bedrock is covered by soft soils and water, a common occurrence in this part of southern Sweden. The information is stored in the SGU database.

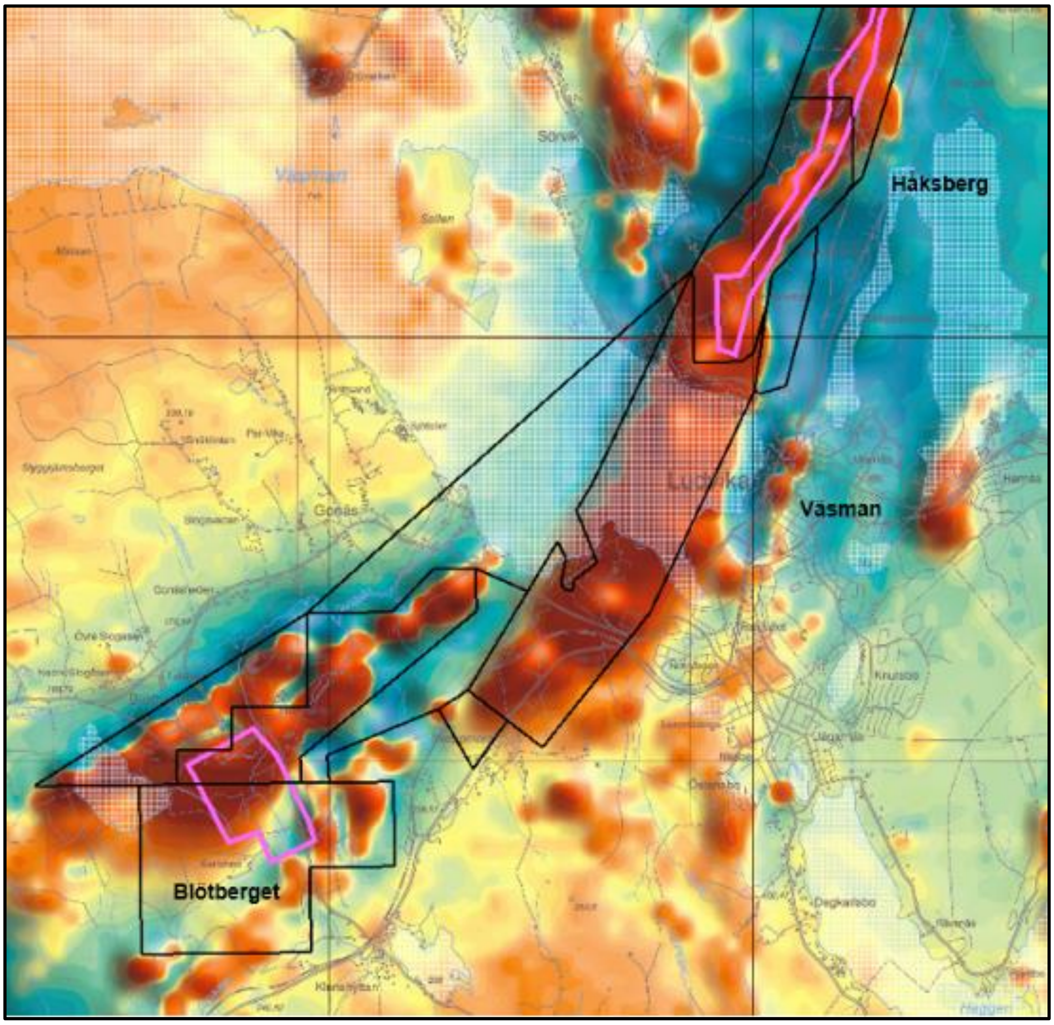


Figure 6-3 **SGU airborne geophysical survey**

Source: NIO

6.2.2 Ground Geophysical Surveys

A ground magnetic anomaly survey of the Vulcanus and Blötberget project areas was conducted in 1950 by ABEM geophysics, on behalf of Stora Kopparberg. The results of this survey assisted in focussing historical drilling campaigns.

In 1967 Terratest (now owned by ABEM) reinterpreted the existing data, as illustrated in Figure 6-4.

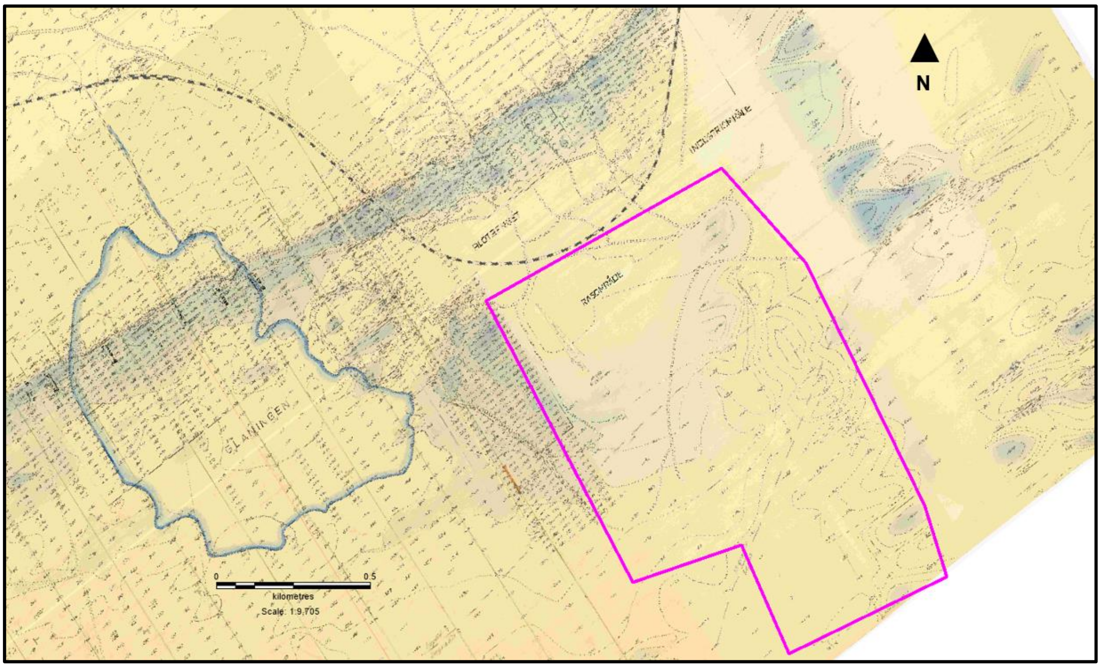


Figure 6-4 Terratest magnetic anomaly map showing current mining concession

Source: NIO

6.2.3 Historical Drilling

From 1942 until 1977, the deposits were systematically diamond drilled for definition and extension.

In areas where mining was on-going or planned, regular drillhole fans, spaced 30 m apart, were drilled from underground positions in drifts. Deeper parts of the deposits were investigated with wider spaced drilling (~100 m). Most of the drillholes (~80 %) are collared underground in both the hanging and footwalls, and in some cases, mineralised zones. All of these drillholes had varying dips and azimuths. Only the drillholes probing the deeper, down dip parts of the deposit, were drilled from the surface.

The deeper drillholes, drilled in the late 1960's and early 1970's, were initially drilled with 52 mm core with step down to 32mm core and then 22mm core in the deeper parts of the hole. Drilling has been carried out in the past by contractors as well as by the mining companies themselves.

A total of 391 drillholes have been drilled historically at Blötberget, totalling 32,751 m.

All digitised historical drillholes either have locations and surveys in mine maps, or in supplementary documentation. Where possible, collars have been located in the field and verified by NIO.

Average recoveries for these drillholes have not been recorded. In the majority of cases when re-logging of the available historical cores using current standard procedures was carried out, it was noted that most core loss was not recorded. In cases where it had been recorded, it was often not mentioned in the accompanying geological log and only on the actual core box. NIO does not have access to many of the historical cores as they have either been destroyed or not yet located. Daily drilling reports and hole status reports were historically not used or they have been mislaid or destroyed.

Mine maps and historical drilling data have been collected from various sources (including the Mining Inspectorate, a division of SGU) and digitised where possible. Drill core from historical exploration drilling in the Blötberget project area has been recovered at the core storage facility at the SGU in Malå, along with additional drill core found in buildings on the former mine sites.

In total, 45 drillholes from Blötberget were found in Malå and 15 at the former mine storage facility in Håksberg. 41 of these holes (6,884.83 m) have been re-logged, 1,047.75 m of mineralised material has been re-sampled and re-assayed according to current best industry practice and standards. This included mineralised core that had not been sampled historically as it was considered too low grade when assessed visually. Approximately 5-10 m mineralised core was sampled adjoining the historical sampled sections.

Figure 6-5 **Typical condition of recovered historical core and core boxes**

Source: NIO

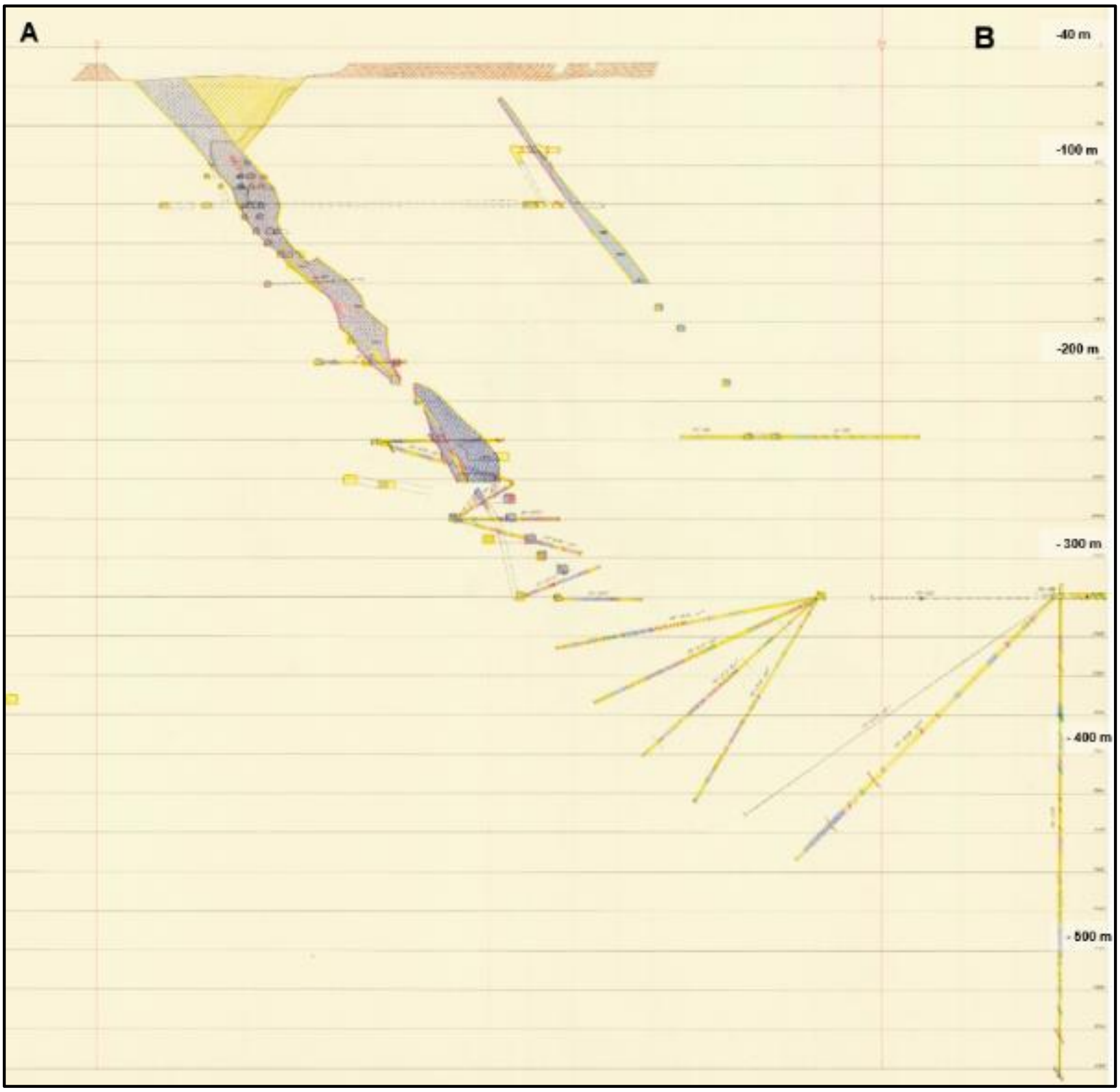


Figure 6-6 **Cross section showing old workings and delineation drilling**

Source: NIO

6.2.3.1 Downhole Geophysics & Survey

Most of the deeper drillholes, drilled in the late 1960's and early 1970's, to investigate the depth extension of the iron ore zone, were logged with a magnetometer as well as deviation surveyed by Terratest AB.

Historically, only holes longer than 150-200 m were surveyed for deviation on a regular basis. To date, NIO has only been able to locate downhole deviation survey records for holes drilled from surface. These records have been entered into the database and some have been reconstructed from trace plots of the respective holes.

6.3 Historical Estimates

The mining company, Stora Kopparbergs Bergslags AB submitted a closure report to the Inspector of Mines at the cessation of mining activities in 1979. The 'reserves' (non-compliant) were estimated to be 25 Mt at an average grade of 43.5 % Fe.

6.4 Historical Production

The final production capacities achieved in 1979 at Blötberget was 400 Ktpa (thousand tonnes per annum).

Operations ceased in June 1979. A total of 19 Mt of material, averaging 37 % Fe Total, 0.55 – 0.8 % P and <0.01 % S, was reportedly extracted.

The information in the tables below was extracted from reports given to "Bergmästarämbetet", Falun between the years 1973 and 1979. Figures for 1979 are up until the end of June (six months), as this is when the mine closed.

6.5 Adjacent Properties

Within a couple of kilometres northeast of Blötberget there are three smaller abandoned underground mines, namely:

- Frädmundsberg (mined up to 1944);
- Gonäs (mined up to 1919)
- Våghalsen/Finnäset (mined up to 1919)

The old workings at Finnäset became the investigation centre for the Väsman deposit, with a shaft sunk to 280 m.

Adjacent to the Våghalsen/Finnäset area, and located under Väsman Lake in the direction of Håksberg, lies the Väsman exploration target. This has previously been investigated via an exploration drift at the 300 m level, driven from a separate shaft at Finnäset.

Table 6-1 Historic feed and processed grades

Year	Feed % Fe	Magnetite Concentrate				Hematite Concentrate				Fe-Recovery %
		w %	Assay			W %	Assay			
			% Fe	% SiO_2	% P		% Fe	% SiO_2	% P	
1973	37.2	23.2	68.2	2.9	0.1	25.9	60.9	6.8	0.48	85.6
1974	37.3	22.9	68.3	2.66	0.11	25.6	60.2	6.85	0.56	85.4
1975	35.7	20.4	67.5	2.4	0.11	24.4	59.9	7.52	0.57	80.4
1976	37.1	14.8	67.4	3.87	0.07	28.9	61.1	6.71	0,50	76.3
1977	37.1	22,4	67.2	3.06	0.09	25.7	61.3	5.91	0,54	83
1978	34.5	18.2	68	2.93	0.1	25.5	61.7	5.93	0,46	83
1979	34.5	18,5	68.5	2.93	0.06	27,4	61.7	5.93	0,35	83
Average (excl. 1979)	**36.5**	**20.3**	**67.8**	**2.97**	**0.1**	**26**	**60.9**	**6.62**	**0.52**	**82.3**

Source: NIO

Table 6-2 Combined historic processed grades

Year	Combined Magnetite-Hematite Conc. (calculated)				
	W %	% Fe	% P	Proportion magnetite	Proportion hematite
1973	49,1	64,3	0,30	47,3	52,7
1974	48,5	64,0	0,35	47,2	52,8
1975	44,8	63,4	0,36	45,5	54,5
1976	43,7	63,2	0,35	33,9	66,1
1977	48,1	64,0	0,33	46,6	53,4
1978	43,7	64,3	0,31	41,6	58,4
1979	45,9	64,4	0,23	40,3	59,7
Average (excl. 1979)	**46,3**	**63,9**	**0,33**	**43,7**	**56,3**

Source: NIO

Note: Production between 1973 and 1979 was from the Hugget and Betsta deposits

The process plant handled up to a maximum of 415,000 tonnes of feed material per annum. This maximum was achieved in 1976 when an additional shift was added, increasing the operational time from 5,058 to 5,824 hours (66% utilisation grade).

To summarise the figures in the table (excl. 1979):

- Feed Fe varied between 34.5 and 37.3 %
- Fe grade in magnetite concentrate is quite consistent: 67.2 and 68.3 % (Δ 1.1)
- Fe grade in hematite concentrate varies between 59.9 and 61.7 % (Δ 1.8)
- P-grade variation in magnetite concentrate from 0.07 to 0.11 %P
- P-grade variation in hematite concentrate from 0.46 to 0.57 %P
- Proportion of magnetite and hematite concentrate has varied from 35:65 to 50:50 (rounded figures) over the six year period 1973 to 1978. (Avr.= 44:56)
- Fe-recovery has varied between 76.3 and 85.6 %

Tests were carried out with the SALA HGMS in 1976 which showed that P-grade in hematite concentrate could be decreased to 0.22-0.27 % P without grinding and to 0.12-0.14 % P with grinding. No further details are known.

The magnetite primary concentrate was reground and passed to a second stage of LIMS (to reduce P).

In 1978, a decision was taken to stop mining of the Sandell magnetite ore body due to high content of phosphorus combined with the need for fine grinding

Large changes in the proportion of magnetite/hematite concentrate are noted in the production over the five year period ahead of closure in 1979, and reflect the variability of the ore composition with respect to hematite and magnetite.

The recovery has varied between 76.3 to 85.6%. Mineral recovery appears to be greater when magnetite percentages are higher.

7 GEOLOGICAL SETTING & MINERALISATION

7.1 Regional Geology

Regional geological maps over the area have been published by the SGU. Publications include a colour version of map sheet Ludvika AF158, 1:50 000 (1986), a more detailed map in scale 1:50 000 (2005).

The Blötberget apatite-iron oxide deposit is located in the western part of the intensely mineralised Paleoproterozoic Bergslagen Province (“**the Province**”) in south central Sweden.

The Province is volumetrically dominated by several generations of intrusive rocks, which enclose inliers of metasedimentary and metavolcanic rocks. The Province has been described as being an extensional, continental back-arc, magmatic example. The metasedimentary and metavolcanic inliers are of great importance as they host an overwhelming majority of the more than 6,000 known metallic mineral deposits and prospects in the Province.

These rocks have been subjected to multiple-phase deformation and metamorphism under mainly greenschist to amphibolites facies conditions. Pre-Svecofennian rocks are not exposed but various isotopic, petrogenetic, and trace element studies of supracrustal rocks imply that an older Proterozoic, possibly in part Archaean, felsic basement underlies the western part of the Province.

The deposits in the neighbouring area occur along a ~40 km long, broad zone This zone of mineralisation is the third largest iron ore deposit in Sweden by production, only outnumbered by the giant Kirunavaara and Malmberget iron ores in Norrbotten, northern Sweden.

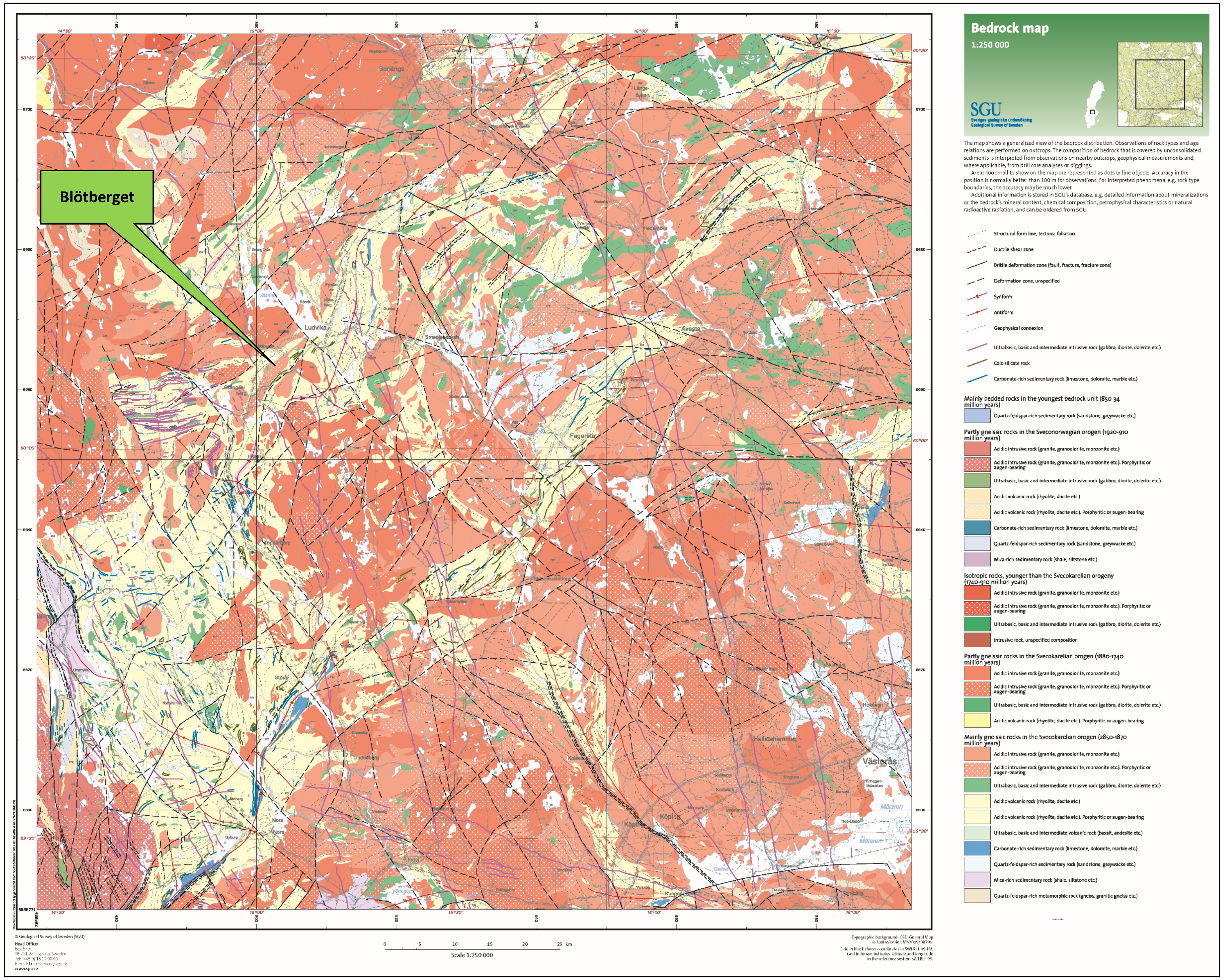


Figure 7-1 **Regional geology map**

Source: Swedish Geological Survey

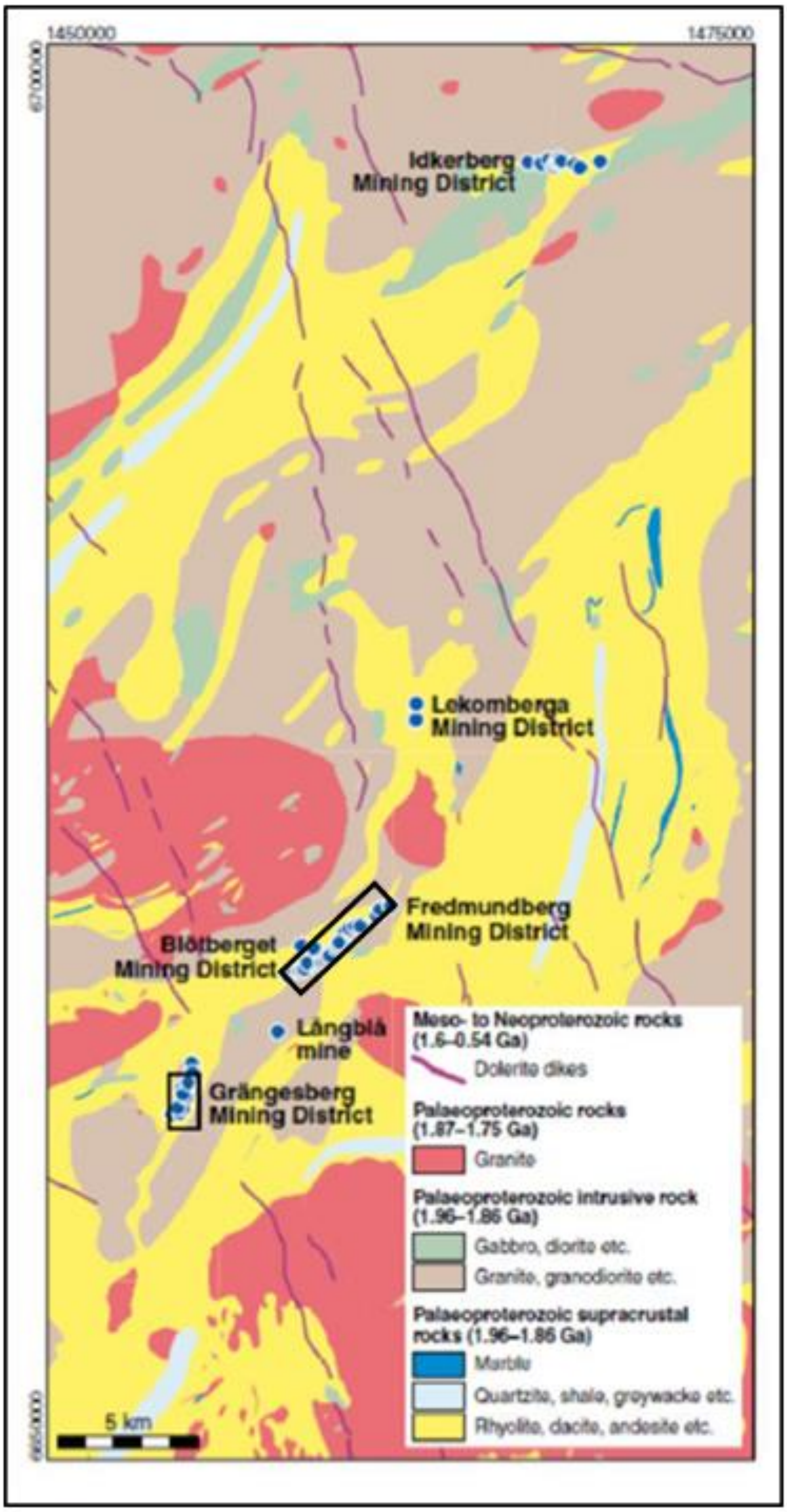


Figure 7-2 **Geological map of part of Western Bergslagen**

Source: Stephens et al (2007)

7.2 Property (Local) Geology

The mineralised zone at Blötberget appears as a set of vertically narrow, elongated lenses dipping 50°–70° to the SE. Airborne geophysical surveys and historical drillholes indicate that mineralisation extends to a depth of at least 900 m below surface.

The host rocks to the Blötberget iron mineralisation have traditionally been classified as belonging to the ”leptite formation”, i.e. mainly felsic to, more rarely, intermediate, regionally metamorphosed (c. 1.90–1.87 Ga) volcanic rocks. In most parts of the Bergslagen ore province these leptites are predominantly SiO_2-rich and have mainly rhyolitic to dacitic compositions, yet, the immediate host rocks to Blötberget ores exhibit significantly more of intermediate to basic compositions. The metavolcanic rocks are locally feldspar-porphyritic, fine-grained and generally range between rhyolitic-dacitic to basaltic/andesitic in composition. A number of the observed leptites within Blötberget area, particularly in the mining concession, also exhibit crosscutting relations to various rock units and have been interpreted as subvolcanic in origin.

Alteration is evident in these host rocks, both in the form of regional-style sodic or potassic alteration and locally, as disseminated as well as discrete phyllosilicate (mainly biotite + chlorite) and amphibole-rich zones. These alteration assemblages systematically occur in and around the main ore zone.

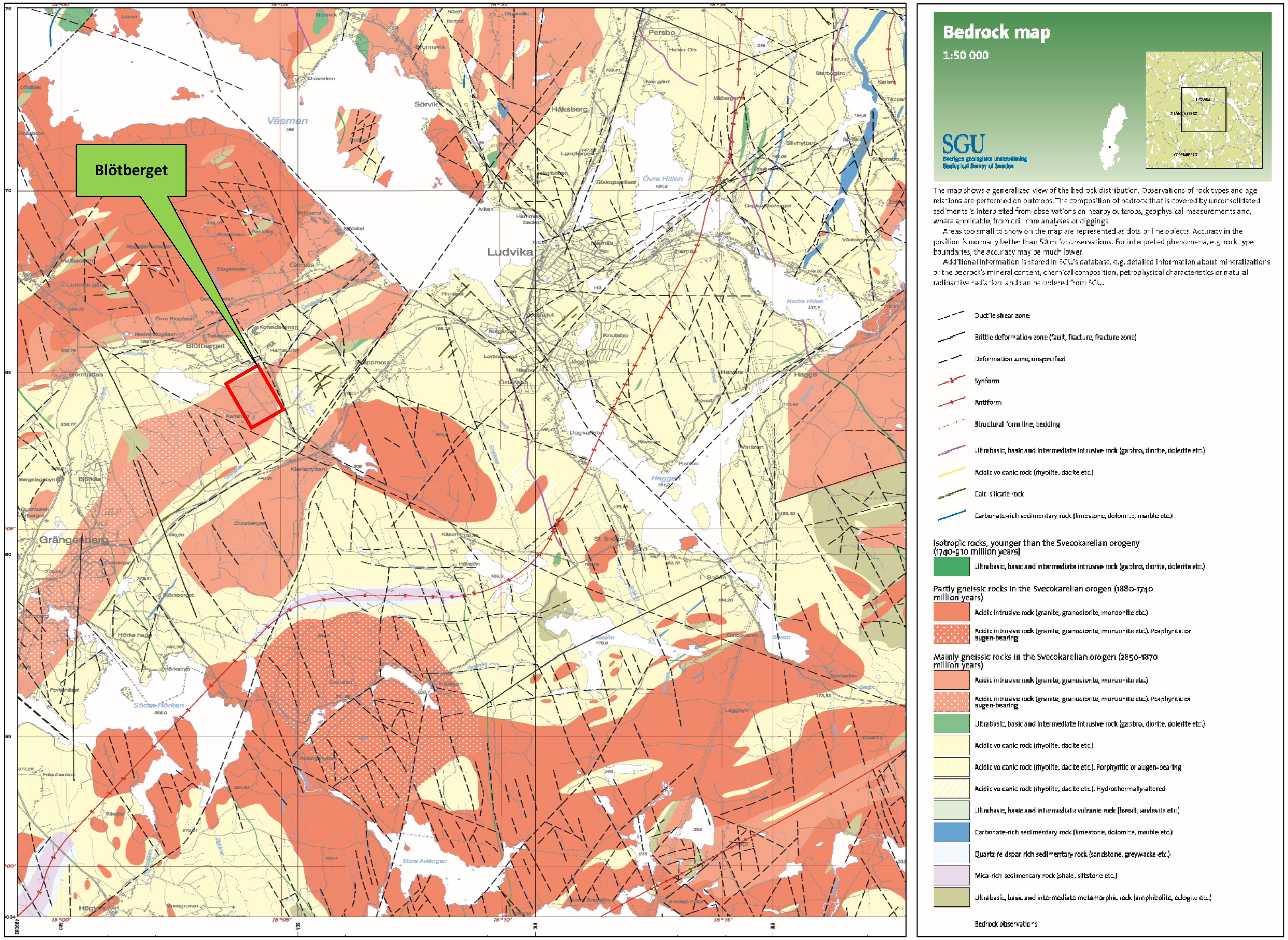


Figure 7-3 **Local geology map**

Source: Swedish Geological Survey

7.3 Mineralisation

The mineralisation at Blötberget is a so-called "apatite lake ore" which, besides the iron mineral magnetite and hematite, also contains the phosphorus mineral apatite, which previously caused problems in the production of iron. With the technological developments that occurred when the so-called Thomas process was invented in 1879, it became possible to also take advantage of ore that was rich in phosphorus.

The Blötberget field consists of five mineralised bodies, from west to east, these are identified in Table 7-1.

Table 7-1 Mineralisation zones

Mineralised Body	Short Form Name	Mineralisation
Kalvgruvan	KALV	apatite-rich magnetite
Flygruvan	FLY	apatite-rich, hematite-dominated, minor magnetite
Hugget & Betstamalmen	HUG & 'Betsta' or 'the Wedge'	Apatite-rich magnetite-hematite
Sandellmalmen	'Sandell'- SAND	Apatite-rich magnetite

Source: NIO

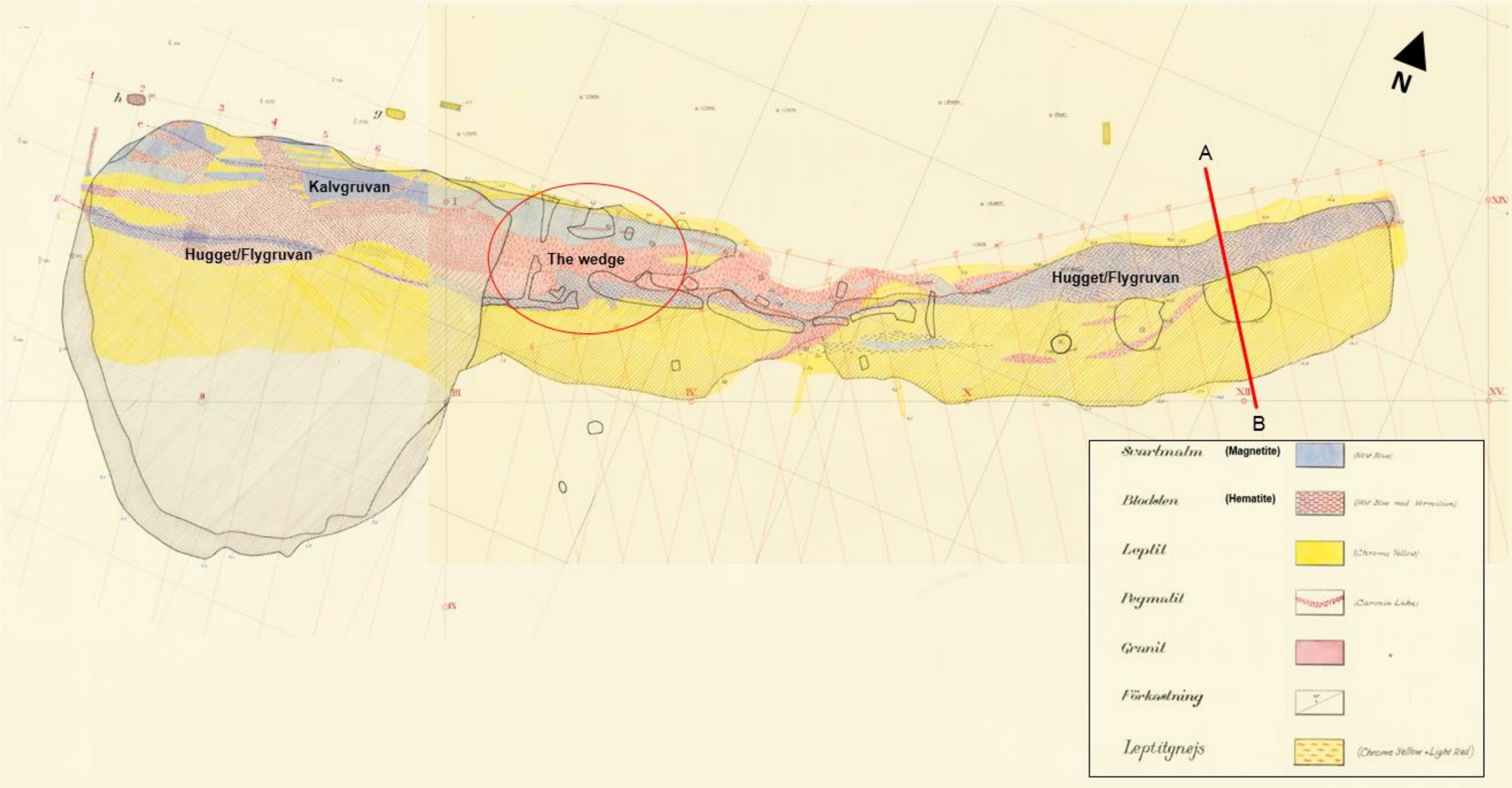


Figure 7-4 **Plan Location of mineralised zones**

Source: NIO

The Hugget/Flygruvan and Kalvgruvan zones had previously been mined down from near-surface to the 250 m and 350 m levels respectively. The units dip towards the southeast at between 50° and 55° in the near-surface mined-out areas, and flatten at depth to ~25°.

The area, previously known as ‘the Wedge’ or Betsta, was an unknown area between the two former mining concessions, Vulcanus and Blötberget. The Wedge was successfully explored during the 2014 drilling programme.

Kalvgruvan and Hugget/Flygruvan have now been shown to be continuous zones of mineralisation Figure 7-4.

8 DEPOSIT TYPE

The Blötberget deposit and its NNE continuation to Idkerberget constitute an anomaly in the Bergslagen Province. Of the >6000 deposits in the Province, registered in the SGU mineral deposits database, 5500 are iron oxide deposits. Of these, most are either banded iron formations or skarn-type deposits, except those in the spatially restricted Grängesberg-Blötberget-Idkerberget area.

The Blötberget deposit and its northern extension to Idkerberget thus represent a significant ore genesis and geological anomaly in the Province. Based on the mineralogy, deposit geometry, host rock relations and geochemical character, it is evident that the Blötberget apatite-iron oxide deposits represent Paleoproterozoic Kiruna-type deposits that have been deformed and metamorphosed to amphibolite-facies grade.

The similarities with the Kiruna deposits were acknowledged early, but hypotheses concerning the origin of these ores have varied over time, from direct magmatic, exhalative sedimentary to hydrothermal and metasomatic. Two main hypotheses on the origin of apatite-iron oxide ores have dominated the discussions during recent years, namely:

- Hydrothermal or orthomagmatic origin i.e. formed directly from a melt.
- Direct-magmatic origin, noted through several textural similarities between the Kiruna deposit and the much younger apatite-iron oxide ore at El Laco in Chile.

The magmatic model was challenged by Sillitoe & Burrows (2002), and a theory for a hydrothermal replacement process was proposed in its place. This was subsequently rejected based on the magmatic textures and relationships between apatite iron boulders and the host-rock. However, a hydrothermal origin for other apatite-iron oxide ore deposits was proposed.

A comparative study of different apatite-iron oxide ore deposits in North America was completed and suggested that these deposits, characterised with respect to age, tectonic setting, mineralogy and alteration, ought to be referred to as Iron oxide-copper-uranium-gold-REE deposits. These were later to be called Iron Oxide Copper Gold (“**IOCG**”) deposits. The deposits of Kiruna-type were considered a sub-set within this IOCG concept, and a primary, shallow-level, hydrothermal origin has been suggested.

The IOCG concept marked the onset of an exploration frenzy for these deposits and subsequently more research has been conducted.

It has been shown that there is great variation between different possible IOCG occurrences, tentatively related to different ore forming processes. The trace element composition in apatite from the Tjårrojåkka deposit in Norrbotten for instance, is very different compared to other IOCG deposits and the question remains whether Kirunavaara should be considered an IOCG type deposit at all. This statement is also valid for the Blötberget deposit, as it does not contain any significant concentrations of either gold or copper, which perhaps emphasizes its similarity with the Kirunavaara ore. Indeed, the concept of an orthomagmatic origin for the Kiruna-type deposits suggest they are “non-IOCG” (*Nilsson et al. 2013*)

Besides iron and apatite being present in these rocks, there are significant accumulations of rare earth elements (“**REEs**”) and phosphorous (*Nilsson et al. 2013*).

9 EXPLORATION

Since the formation of NIO, several surface sampling campaigns have taken place. The majority of these have been within the mining concession areas but some have extended to include the surrounding exploration licence in order to allow a better understanding of the geochemical relationship between the satellite deposits and the main Blötberget mineralised zone. This work assisted with the realisation of potential sites for a bulk sample/test mining site. A handheld magnetic susceptibility (KT-10) device and a Thermo Niton x-ray fluorescence ("**XRF**") XL3 were used to ascertain iron and magnetite percentages of outcrop samples. Rock samples were then sent to ALS for chemical assay.

During 2009, Kopparberg Mineral AB carried out a more detailed magnetometry survey over a limited part of the Blötberget area on behalf of NIO (Figure 9-1).

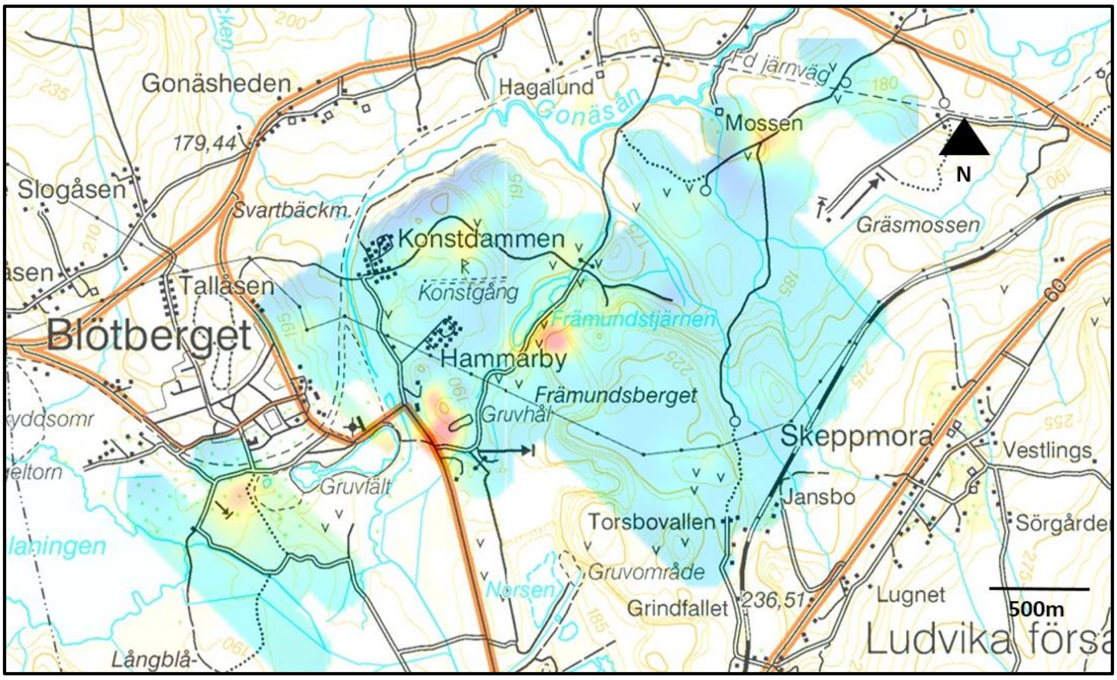


Figure 9-1 **Ground magnetic anomaly map (November 2009)**

Source: NIO

10 DRILLING

10.1 2012 Drill Programme

A drilling programme was undertaken during the summer and winter of 2012 and was completed in November 2012. This 16 hole programme included twinned drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling.

NIO completed 16 drillholes totalling 7,430 m of drilling. The NIO drilling in 2012 was carried out by the Swedish contractor, Drillcon Core AB, or by their Finnish subsidiary Suomen Malmi Oy ("**SMOY**") using Onram 1000 and Onram 1500 drill rigs and wireline 56 methodology, the programme recovered 39 mm diameter drill core.

One hole, BB12015-MET, was drilled for the purposes of generating material for metallurgical sampling, using HQ-size equipment to recover 63.5 mm diameter core. To date, this has been the only hole drilled using oriented core.

No drillholes were water pressure tested during this drilling campaign.

Three holes were left uncompleted as they hit highly fractured and clay altered rock which collapsed the drillhole. These holes were BB12003 (@ 400 m), BB12012B (@ 810 m) - BB12012 was a re-drill of this hole - and BB12014-MET (@ 30 m). No mineralisation was encountered in these holes.

Several deviations from the planned targets occurred during this drilling campaign, due largely to the small drill equipment and small drill diameter used.

Figure 10-1 SMOY drilling rig (2012 drill programme)

Source: NIO

Table 10-1 2012 NIO drill programme summary

Hole ID	Collar X	Collar Y	Collar Z	Depth m	Azimuth °	Dip °	Completed	Orientation	Water pressure
BB12001	504478.89	6664648.45	-46.85	680.60	330	-77	Y	N	N
BB12002	504054.36	6664849.61	-49.97	300.25	317	-77	Y	N	N
BB12003	503754.84	6664541.22	-42.16	404.20	325	-75	Y	N	N
BB12004	503866.26	6664556.24	-47.43	495.20	320	-74	Y	N	N
BB12005	503938.02	6664523.59	-48.22	530.75	350	-78	Y	N	N
BB12006	503850.94	6664388.32	-45.81	611.50	352	-76	Y	N	N
BB12007	503741.82	6664364.83	-43.08	599.30	353	-75	Y	N	N
BB12008	503922.63	6664262.80	-45.82	706.50	350	-76	Y	N	N
BB12009	503967.26	6664148.12	-47.03	769.75	356	-77	Y	N	N
BB12010	503982.65	6664354.08	-46.48	649.40	357	-78	Y	N	N
BB12011	504559.18	6665269.31	-38.02	101.80	322	-54	N	N	N
BB12012	504496.76	6665214.94	-36.60	287.50	395	-56	Y	N	N
BB12012B	504491.60	6665241.50	-42.75	8.10	322	-50	N	N	N
BB12013	503870.56	6664162.06	-45.99	782.90	326	-78	Y	Y	N
BB12014-MET	503518.32	6664743.35	-44.12	30.00	89	-70	N	N	N
BB12015-MET	503795.81	6664758.28	-45.58	468.30	146	-80	Y	N	N
			Total	**7426.05**					

Source: NIO

10.2 2014 Drill Programme

The 2014 drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget (formally known as "**the Wedge**" or Betsta area) and to infill the intermediate depth extension of Hugget, (-320 m to -660 m; measured from surface depth and relating to mining blocks) in order to improve the confidence of the geological model.

At the onset of planning for this drill programme, it was deemed necessary that a larger diameter drilling method, NQ2 (50.6 mm core diameter), and larger, more powerful drilling rigs were to be used to alleviate deviation; hole losses due to fractured/clay strata and to improve core recovery.

13 drillholes, totalling 7,093 m, were drilled by the Finnish contractor Kati using a Sandvik DE 140 and Onram 1000 and NQ2 drilling methodology, recovering 50.6 mm diameter drill core and producing 75.5 mm diameter drillholes. Kati used a hexagonal reamer which helped ensure that the drillholes had minimum deviation.

One of the drillholes (BB_14-011) was drilled down-dip for geotechnical purposes.

All holes were drilled with orientation information, using either a Devico Devicore or ACT II Reflex tool to provide accurate structural information. Eight holes were subject to pump testing to provide information on the potential for water bearing fracture zones. Each hole was measured for deviation using a Devicore Deviflex gyroscope.

Figure 10-2 **Kati drilling rig (2014 drill programme)**

Source: NIO

Table 10-2 2014 NIO drill programme summary

Hole ID	Collar X	Collar Y	Collar Z	Depth m	Azimuth °	Dip °	Completed	Orientation	Water pressure
BB_14-001	503966.65	6664675.74	-47.54	430.05	336	-78	Y	Y	N
BB_14-002	504010.83	6664605.24	-49.44	464.28	336	-75	Y	Y	N
BB_14-003	504059.67	6664542.24	-47.97	509.90	336	-75	Y	Y	N
BB_14-004	504107.87	6664434.06	-45.55	596.70	336	-78	Y	Y	N
BB_14-005	503776.23	6664625.74	-44.55	464.30	322	-78	Y	Y	Y
BB_14-006	504301.01	6664578.15	-48.77	564.10	322	-80	Y	Y	Y
BB_14-007	504436.00	6664550.14	-46.78	635.70	322	-78	Y	Y	Y
BB_14-008	503623.60	6664584.70	-44.32	455.80	322	-78	Y	Y	Y
BB_14-009	504236.00	6664685.00	-42.75	525.00	322	-78	Y	Y	N
BB_14-010	504380.00	6664468.00	-42.75	665.00	322	-78	Y	Y	Y
BB_14-011	504191.05	6664870.50	-49.48	664.30	142	-75	Y	Y	Y
BB_14-012	504153.76	6664582.01	-49.68	497.70	328	-78	Y	Y	Y
BB_14-013	504473.21	6664652.83	-46.64	620.00	325	-77	Y	Y	Y
			Total	**7092.83**					

Source: NIO

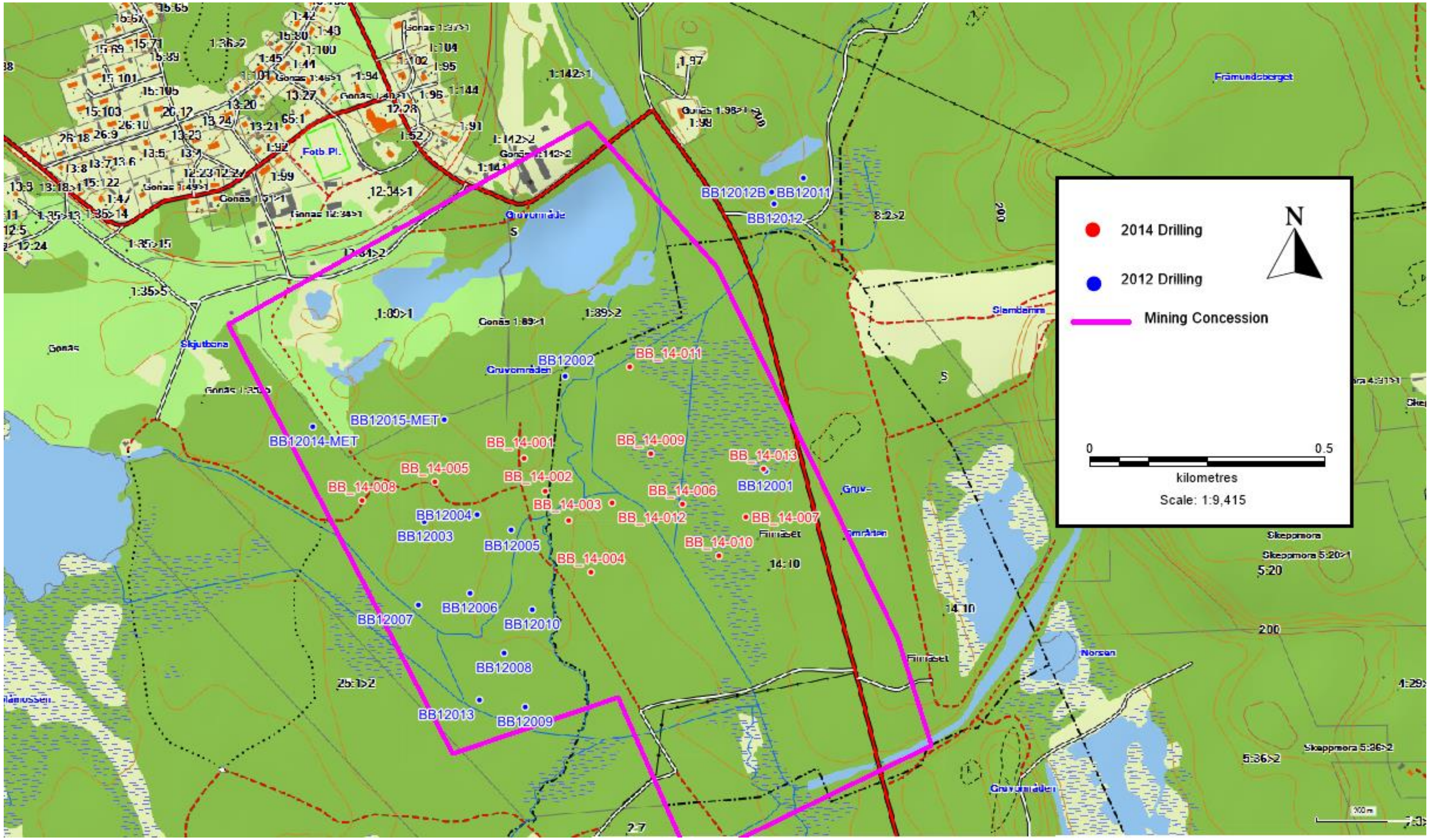


Figure 10-3 **2012 and 2014 drillhole locations**

Source: NIO

10.3 Logging

10.3.1 Re-Logging of Historical Core

During 2011/12 Berg och Gruvundersökningar AB (“**BGU**”) was engaged by NIO to log and sample historical cores that were stored at the SGU repository in Malå.

13 cores, totalling 5077.21 m were logged for geological and geotechnical data (for RQD), then photographed (dry and wet).

A detailed report containing geological descriptions, profiles and sections was created entitled “*Geological Logging and Sampling of Archive Cores at the Central Core Archive in Malå 2012-13”*. This report was updated once the 2012 drilling campaign had been completed. The updated report was entitled “*Core Logging, Geological Description and Interpretation of Blötberget, Finnäset – Väsman and Håksberg 2013”*. Both of these reports were compiled by BGU.

10.3.2 2012 & 2014 Programmes

The core from the 2012 and 2014 drill programmes was collected in the field by NIO’s technicians or geologists and transported directly to the NIO core logging and sample preparation facilities in Grängesberg. The core storage, logging and sampling facilities are clean, well-organised and provide suitable conditions for logging and sampling (Figure 10-4).

Figure 10-4 **Grängesberg core storage facility**

Source: 2014 GeoVista Resource Report

The core is firstly placed on roller tables and inspected for mislabelling or depth inaccuracies. Any discrepancies are checked against daily drilling logs and followed up with the drill teams.

The core is further checked using a magnetic pen and areas of “pull” are marked onto the core boxes and a UV lamp is used to detect presence of Scheelite (tungsten).

The core is then placed in V-rails and the orientation line is drawn from the marks placed by the drilling teams from their orientation tool. Once completed the Geologist logs the core utilising the geological logging template, shown in Figure 10-6.

Figure 10-5 Core logging facility showing roller tables

Source: DMT

HOLE ID	BB_141-013
General comments:	

From	To	Length	Description	Lith 1	%	Lith 2	%	Lith 3	%	Lith 4	%	Lith 5	%	LITH TOTAL	Colour 1	Light/Dark	Colour 2	Light/Dark	Colour 3	Light/Dark	Texture 1	Texture 1 occurance	Texture 2	Texture 2 occurance	Texture 3	Texture 3 occurance	mineral type 1	mineral type 1 occurance	mineral type 2	mineral type 2 occurance	mineral type 3
0.00	6.90	6.90		OVB	100									100																	
6.90	8.04	1.14		BMV	80	RFZ	15	SARG	5					100	GR	DARK	GY	DARK	GR	LIGHT	FOL	STRONG	B	LOCAL			AMP	STRONG	CHL	WEAK	BIO
8.04	14.42	6.38		SARG	90	RFZ	10							100	BK		GY	DARK	GY	LIGHT	FOL	STRONG	B	WEAK			BIO	STRONG	AMP	LOCAL	CHL
14.42	15.38	0.96		GRA	95	RFZ	5							100	PK	DARK	GY	LIGHT	GR	LIGHT	FOL	STRONG	GNE	WEAK	B	LOCAL	POTA	WEAK	BIO	LOCAL	EPI
15.38	19.68	4.30	veins purple dark -> possibly precipitation of hematite/magnetite	GRA	75	RFZ	25							100	PK	DARK	RD	DARK	BK		M	STRONG	MOT	STRONG			POTA	STRONG	BIO	WEAK	EPI
19.68	25.16	5.48	GRA MASSIVE (30CM) AT 21,45M + 2 VEINS OF PEG : ONE (4CM) AT 21,40M AND ONE (8CM) AT 21,95M	GRA	85	BMV	10	RFZ	5					100	PP	DARK	PP	LIGHT	YL	LIGHT	FOL	STRONG	GNE	WEAK	UG	LOCAL	POTA	WEAK	BIO	WEAK	EPI
25.16	26.70	1.54		GRA	95	RFZ	5							100	GR	DARK	GR	LIGHT	GY	LIGHT	FOL	STRONG	GNE	WEAK			POTA	WEAK	BIO	WEAK	
26.70	29.89	3.19		GRA	90	RFZ	10							100	GY	LIGHT	PK	LIGHT	YL	LIGHT	FOL	STRONG	GNE	WEAK			POTA	WEAK	BIO	WEAK	EPI
29.89	32.31	2.42		RFZ	60	BMV	40							100	GR	LIGHT	BR	DARK	YL	LIGHT	BREC	STRONG	B	LOCAL			AMP	STRONG	CHL	WEAK	EPI
32.31	33.34	1.03		GRA	95	RFZ	5							100	YL	LIGHT	GY	LIGHT	GR	LIGHT	FOL	STRONG	GNE	WEAK			POTA	WEAK	BIO	WEAK	EPI
33.34	48.65	15.31	RFZ (55CM) AT 38,30M + BRECCIA (15CM) AT 42,25M + VEIN OF PEG (13CM) AT 43,20M + CALCITE INTO THE HOLES OF THE GRA AT 48M ON 25CM	GRA	90	RFZ	10							100	GY	LIGHT	GY	DARK	PK	LIGHT	FOL	STRONG	GNE	WEAK	UG	LOCAL	POTA	WEAK	BIO	WEAK	EPI
48.65	55.55	6.90	RFZ (70CM) AT 49,25M (PLUS SEVERAL SMALL (5-15CM OTHER RFZ) + veins purple dark -> possibly precipitation of hematite/magnetite + GRA HAS A GNESOSITY TEXTURE AND MASSIVE TEXTURE + VEIN (12CM) OF PEG AT 50,30M	GRA	80	BMV	10	RFZ	10					100	PK	LIGHT	GY	LIGHT	YL	DARK	BREC	STRONG	FOL	LOCAL	M	LOCAL	POTA	WEAK	BIO	WEAK	EPI
55.55	61.03	5.48	GRA (132CM) POROUS BETWEEN 57,85-59,30M + veins purple dark -> possibly precipitation of hematite/magnetite + VEIN (8CM) OF PEG AT 59,95M	GRA	100									100	PP	DARK	PK	DARK	GR	LIGHT	FOL	STRONG	GNE	WEAK	UG	LOCAL	POTA	STRONG	BIO	WEAK	CC
61.03	62.26	1.23		BMV	85	GRA	15							100	GR	DARK	PK	DARK	YL	LIGHT	FOL	STRONG	B	WEAK	GNE	LOCAL	AMP	STRONG	BIO	LOCAL	POTA
62.26	63.68	1.42	RED COLOR -> JASPELITE?	SARG	90	GRA	10							100	RD	DARK	GR	DARK	GY	DARK	FOL	STRONG	GNE	WEAK	M	LOCAL	BIO	WEAK	POTA	WEAK	
63.68	64.87	1.19	BIG CONCENTRATION OF AMPHIBOL GRAIN AT 64M ON 10 CM	BMV	100									100	GR	DARK	RD	DARK	BK		FOL	WEAK	MOT	LOCAL			AMP	STRONG	JASP	LOCAL	PYR
64.87	72.08	7.21	VEIN (5CM) OF PEG AT 71,20M + veins purple dark -> possibly precipitation of hematite/magnetite	GRA	100									100	RD	DARK	PK	DARK	BK		M	STRONG	MOT	STRONG	UG	LOCAL	POTA	STRONG	BIO	WEAK	EPI

Figure 10-6 NIO geological logging sheet

Source: NIO

NIO staff produce logs of the drill core using industry standard procedures. This includes core recovery, geotechnical and lithological logging and photography (wet and dry). In addition, point load tests are conducted on the core in order to assess the rock mechanical properties.

10.3.2.1 Core Recovery & Geotechnical Logging

Geotechnical logging information is collected to determine the Rock Quality Designation ("**RQD**") and Barton's Q classifications.

Fractures, foliation and joints are measured for their angle using a Drill Core Structure Orientation ("**DCSO**") device, supplied by Petroteam Engineering. This device uses two lasers to measure alpha and beta angles and core diameter. The DCSO is connected directly to the logging laptop and the software included with the device enables the data measured to be captured directly into the database. Once the logging has been completed, selection of mineralised core to be sent for sampling takes place (Figure 10-7).

Figure 10-7 Geologist using DSCO device

Source: NIO

11 SAMPLE PREPARATION, ANALYSES & SECURITY

11.1 Introduction

The core from the 2012 and 2014 drill programmes was collected in the field by NIO's technicians or geologists and transported directly to the NIO core logging and sample preparation facilities in Grängesberg.

NIO compiled its own Sampling Quality Manual, which sets out best practice procedures relating to core handling, sampling, analysis and QA/QC procedures and is summarised in the following sections.

11.2 Sampling & Assaying of Historical Samples

Samples from historical drillholes, which have not been re-assayed, contain only % Fe grades as standard, presented as either % Fe HCl (to determine only Fe oxide species), or % Fe Total determinations.

Assays for % SiO_2 and % P have been discovered for less than half these historical assayed samples. Formerly, these historical samples were sampled by visual inspection, i.e. those samples that appeared to be above a grade of 35 % Fe. Material below this subjectively applied high grade 'cut-off' was not considered economic. This methodology has resulted in data gaps in the mineralised material as samples deemed to contain <35 % Fe have not been sampled.

712.8 m of mineralised material was sampled using current industry best practice at the SGU Malå logging facilities. Core boxes were then transported to CL Prospecting in Malå for sawing and density measurements. Samples were then packaged and sent to ALS, Piteå for analysis. In addition samples were sectioned out for metallurgical testing to be carried out on assay rejects at Minpro AB, in Stråssa, Sweden. Samples were also sent for environmental and leach testing by Golder Associates AB, Sweden.

No QA/QC samples were sent along with these samples for assaying by ALS. Density was determined for historical core samples using the Archimedes method.

11.3 Sample Preparation (2012 & 2014 Programmes)

After the lithological and geotechnical logging, sectioning for assaying takes place.

11.3.1 Core Mark-Up

Geologists mark the assay sections on the core boxes as well as on the core itself and insert a sample ticket into the core box. All material with Fe over 5-10 % (determined with hand held XRF, magnetic pull and geological competence), greater than 50 cm in length and within a mineralised section, is selected for sampling.

Sampling lengths are constrained within lithological boundaries in order to assist with sectioning of the mineralised core. Sampled core of similar composition is split into 2 m lengths.

In addition to sampling mineralised core, 1 m of hanging wall (material above the identified mineralised section) and 1 m of footwall (material below) was sampled for each mineralised sample to enable definition of the mineralised zones.

11.3.2 Core Photography

High resolution digital photography (wet and dry) is carried out for each core box from the 2012 and 2014 drill programmes (Figure 11-1 and Figure 11-2).

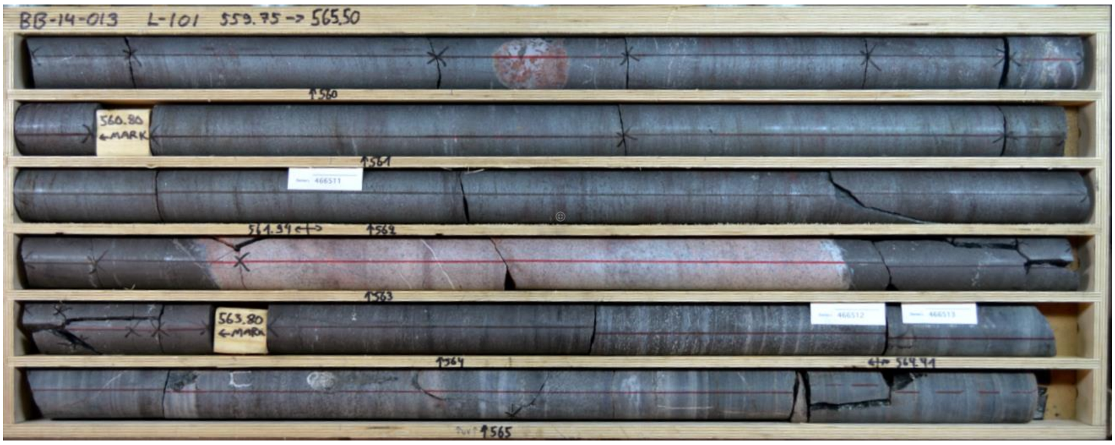


Figure 11-1 Core photography (dry) – BB 14-013

Source: NIO

Figure 11-2 Core photography (wet) – BB 14-013

Source: NIO

11.3.3 Point Load Tests

An industry approved Point Load testing unit is used to assess the rocks mechanical properties (Figure 11-3). The geologist selects a representative, homogenous piece of core based on the geological logging. If the rock has foliation, six measurements are taken – three parallel and three perpendicular within 15-20 cm of each other.

The point load equipment is calibrated each day before use to ensure all measurements are accurate.

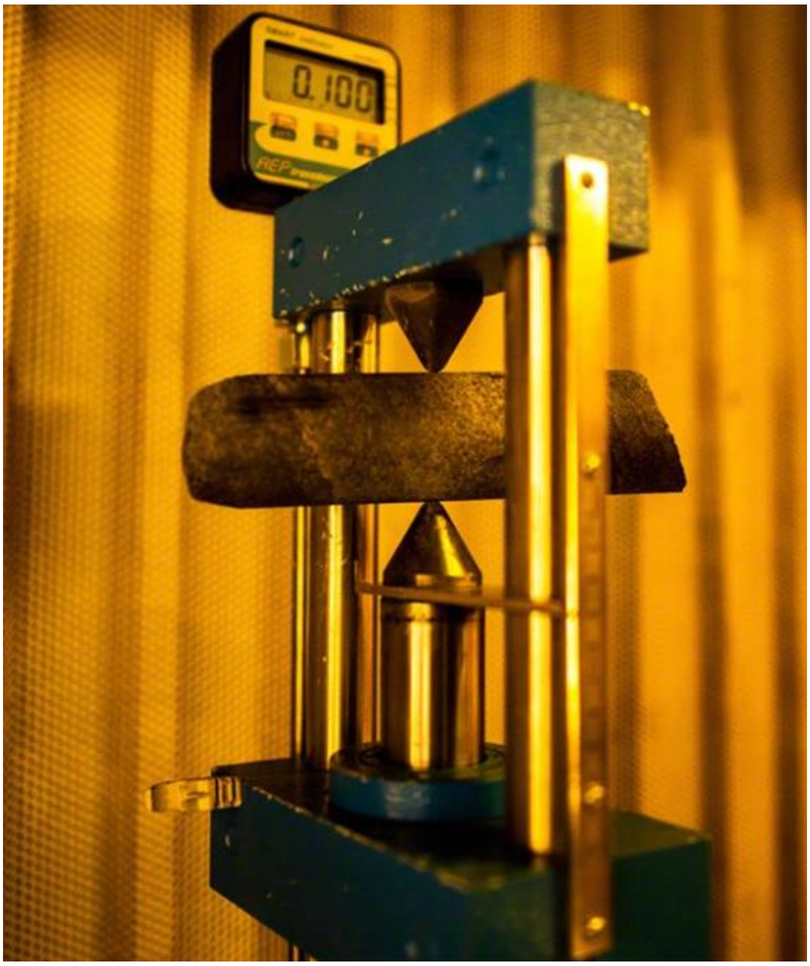


Figure 11-3 Point load device

Source: NIO

11.3.4 Core Splitting

The core is split by diamond sawing, with ¼, ⅓, or ½ core sent for analysis. At the beginning of the 2012 drilling and sampling campaign, ½ core was used. This was subsequently changed to ⅓ core, which is now the standard procedure, in order to preserve more material for later test work, if needed.

The re-assays of the historical core used ¼ core due to the majority of core in Malå having already sampled ½ core historically (Figure 11-4).

Figure 11-4 Diamond saw

Source: DMT

11.3.5 Density Determination

After splitting with a diamond saw, the core is dried in an oven at 70 ° C for 24 hours. Following this, 1 kg of representative core is selected and weighed in air, then placed in water and weighed again while suspended in the water. All sections selected for assay by NIO, from the re-logged historical core as well as from core from the recent drilling campaigns, have had the density determined using the Archimedes method. The weighing machines used during this process are calibrated before use and after every 40 samples using an accredited 1 kg calibration weight supplied by the manufacturer.

The 632 samples taken during the SGU Malå historical re-logging campaign had density measurements carried out on only 100-300g of representative material.

When the Grängesberg core logging and storage facility came into operation in 2011, this density measurement was changed to select samples weighing more than 1000 g (1 kg) and typically using half core. This resulted in a better sample representation and for density determination in the tonnage calculations. Primary bulk density was carried out on all core sent for sampling. Data was thus compiled for low, medium and high grade mineralisation; non- mineralised country rock and internal waste. In total 814 samples with less than 15 % Fe have been analysed.

A correlation curve between bulk density and the grade of Fe is illustrated in Figure 11-5. This is based on 1,527 number of samples tested globally across the Blötberget dataset. There is a clear correlation between bulk density and Fe grade which has allowed the assignment of variable densities to all blocks in the block model (see Section 13).

However, it is considered that an insufficient number of waste sections have had their density determined to date and it is acknowledged that further such determinations will have to be completed for the forthcoming mine planning.

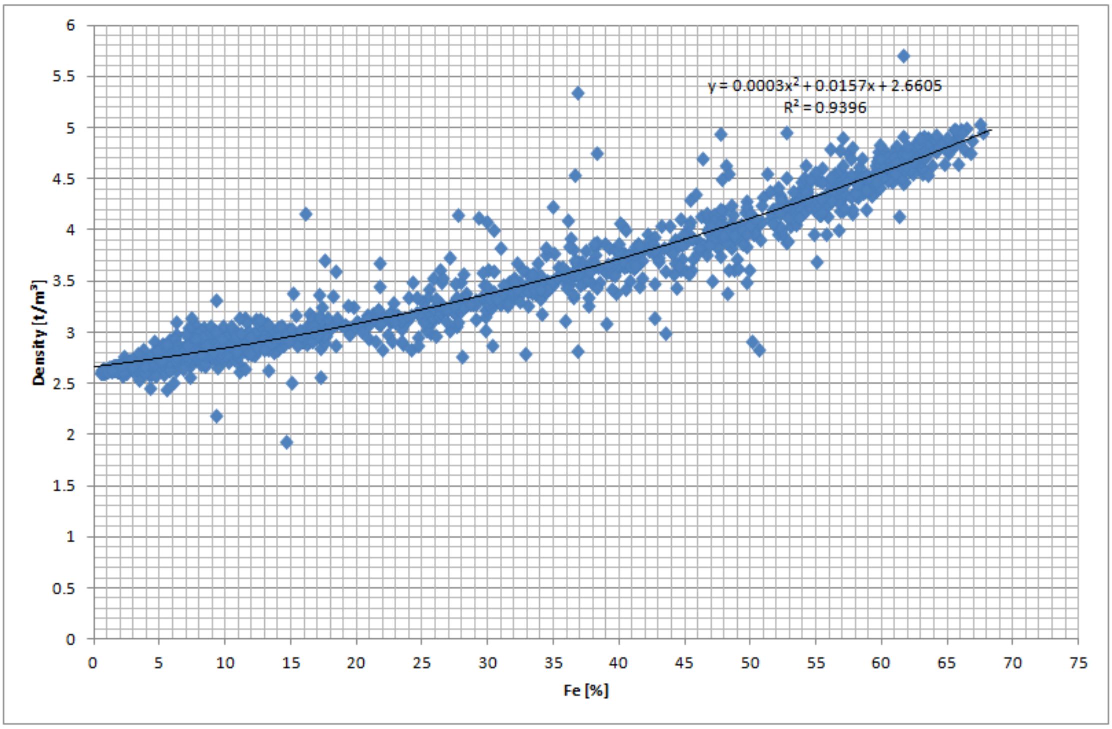


Figure 11-5 **Correlation plot – Density vs. Fe %**

Source: NIO

11.3.5.1 Density Attribution for Non-Oxidized & Oxidized Material

0% Fe = **2.66 t/m³**

8% Fe (equals the average Fe grade of all material not logged as ore) = **2.81 t/m³** (equals the density of all material not logged as ore)

40% Fe = **3.77 t/m³**

11.3.6 Packaging, Dispatch & Transport

Once density determinations have been completed, the sampled core is packaged in a marked plastic bag with a sample tag stapled to the top and a sample tag inside.

Samples are then placed into boxes in batches along with accompanying QA/QC samples and a sample list, prior to being dispatched to the ALS certified laboratory in Piteå, Sweden for sample preparation. All sample batches are sent using a registered Swedish courier service, either Bussgods or DHL. Once samples are picked up, the consignment details are emailed through to ALS preparation labs in Piteå. Once ALS receives the batches in Piteå, they are weighed and recorded in their global tracking system.

11.4 Sample Security

All drill samples from the 2012 and 2014 drill programmes were collected under direct supervision of Project staff from the drill rig and remained within the custody of staff up to the moment the samples were delivered to the laboratory then picked up or delivered to a national courier service for delivery to the ALS, Piteå laboratory.

Samples, including duplicates, blanks and certified reference materials are stored in the secure Grängesberg storage area within the locked and fenced Grängesberg core facility.

Chain of custody procedures consisted of filling out sample submittal forms, which are sent to the laboratory with sample shipments, and also emailed directly to the laboratory to make certain that all samples are received by the laboratory. All samples dispatched are assigned tracking IDs which enabled the laboratory and NIO to track consignments to ensure they arrived successfully.

DMT believes that the sample preparation, dispatch and transit procedures for samples from the 2012 and 2014 drill programmes are in accordance with industry best practice.

11.5 Analysis

Analyses for the 2012 and 2014 samples was carried out by ALS Global in Vancouver.

The ALS sample preparation lab crushes to 70% <2 mm and 250g is riffle split off. This 250g is then pulverized into 85% passing 75 microns. From the ALS sample preparation laboratory in Piteå, ampoules of approximately 250 g of the crushed and milled material is sent to ALS in Vancouver for analysis.

11.5.1 Malå Historical Sampling Campaign

Of the 632 samples taken during the SGU Malå historical core logging project undertaken by BGU in 2011-12, 560 were analysed using x-ray fluorescence (“**XRF**”) equipment - ME-XRF11b and ME-XRF21n.

XRF is the method of choice for analysis of oxide iron ores throughout the industry. The lithium borate fusion technique coupled with XRF, offers a robust and repeatable method, consistent with industry requirements. The relatively low flux to sample ratio offers good sensitivity for the majority of elements and creates a matrix which is not subject to particle size effects. With very few spectral interferences and high instrument stability, the XRF method delivers highly accurate and precise results across the full range of iron oxide ore types.

During 2012, analysis was carried out using XRF with either ME-XRF15b or ME-XRF21n equipment. ME-XRF15b was used when samples were suspected of containing larger amounts of sulphide minerals as this method of analysis is more reliable for sulphide-rich samples. In total, 59 samples were analysed with ME-XRF15b.

616 samples were also sent for ME-ICP61a analysis. During the analysis, the sample is digested in a mixture of nitric, perchloric and hydrofluoric acids. Perchloric acid is added to assist oxidation of the sample and to reduce the possibility of mechanical loss of sample as the solution is evaporated to moist salts. Elements are determined by inductively coupled plasma – atomic emission spectroscopy ("**ICP-AES**"). This method is useful for analysis of trace elements such as rare earth elements ("**REEs**").

Table 11-1 Malå historical sampling summary

Total samples analysed	XRF11b/XRF21n	XRF15b	ICP61a
632	560	59	616

Source: NIO

11.5.2 2012 & 2014 Programmes

As of March 2013, only ME-XRF21n analysis was used for samples from Blötberget as the amount of sulphides encountered in the Blötberget 2012 drill programme showed that this method had sufficient detection limits for the amount of sulphur present in the samples.

Samples were subject to Loss on Ignition ("**LOI**") testing using a Thermo Gravimetric Analyser ("**TGA**").

11.5.3 Coarse Rejects & Pulps

After return of the coarse rejects from Piteå and the analysed pulps from Canada. The pulps were subjected to SATMAGAN analysis as well as magnetic susceptibility determination.

11.5.3.1 SATuration MAGnetisation Analyser ("SATMAGAN")

SATMAGAN is used to measure the total magnetic moment of a sample in a saturating magnetic field.

A 1.2cm^3 of pulp material of <0.3 mm grind size is placed into the SATMAGAN. The SATMAGAN weighs the sample in air, then in a magnetic field. When combined with the Fe % assay results, the proportion of Fe bound to magnetite can be determined.

The results from the SATMAGAN produce quantified percentages of magnetic material, which has been utilised in conjunction with Fe % Total XRF and inductively coupled plasma ("**ICP**") assays to determine magnetite and hematite ratios effectively. The results of the SATMAGAN determinations are presented in the Figure 11-6.

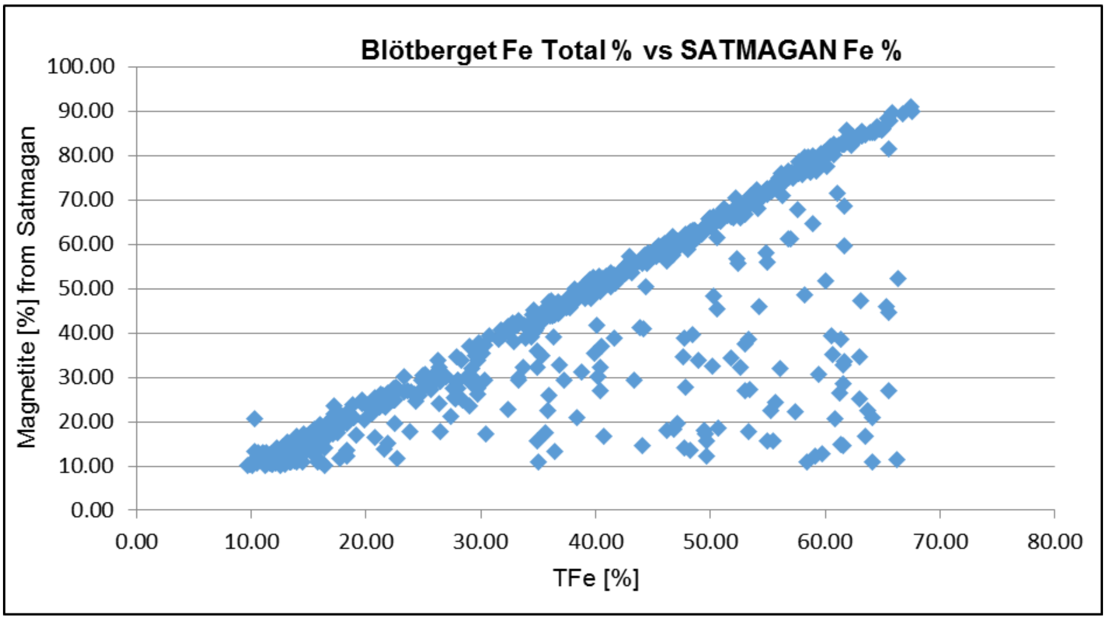


Figure 11-6 **Correlation plot –Fe Total % vs. SATMAGAN Fe %**

Source: NIO

11.5.3.2 Magnetic Susceptibility

A KT10 magnetic susceptibility (“**Magsus**”) meter has been used to analyse all return pulps currently available on site at the core storage facility in Grängesberg.

The pulp bags are agitated by hand to homogenise the sample and then three reading are taken, once from the top, bottom and side of the pulp bags. An average for each sample is then calculated from these values.

The magnetic susceptibility metre determines the degree of magnetization of the material in response to an applied magnetic field, which enables accurate determination of Fe bound to magnetite when combined with the certified Fe % from the ALS results.

When the Magsus values are plotted against SATMAGAN contained magnetite percentage, the results correlate reasonably well up to a SATMAGAN value of approximately 70 % magnetite, above which the correlation is more scattered (Figure 11-7).

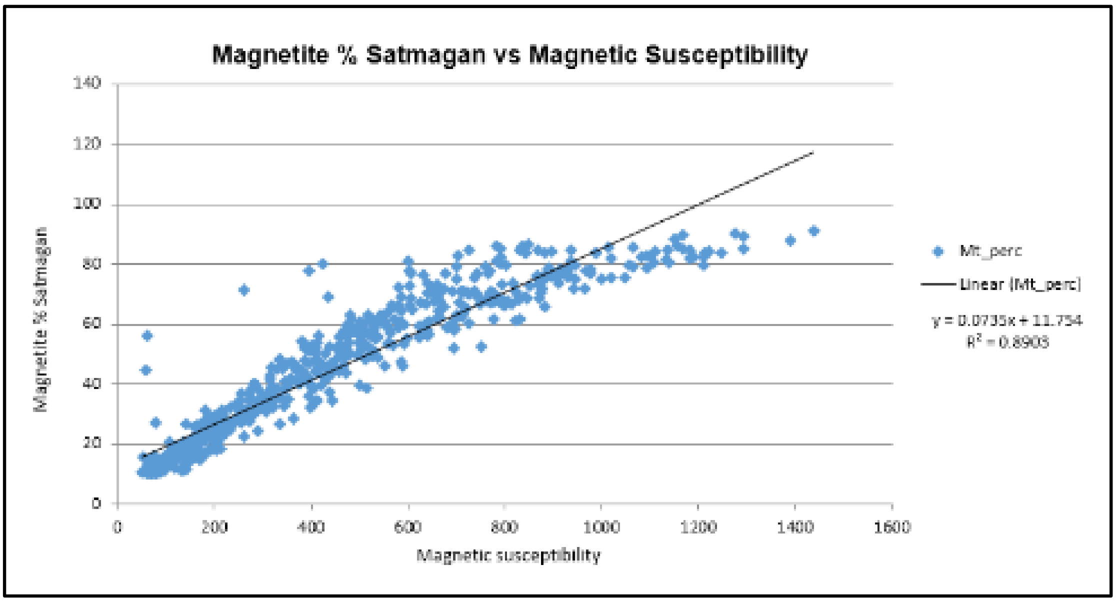


Figure 11-7 **Correlation plot –Magnetite % SATMAGAN vs. Magnetic Susceptibility**

Source: NIO

11.6 Quality Assurance / Quality Control ("QA/QC")

11.6.1 Historic

There are no records documenting the QA/QC control procedures during historical logging, sampling and analysing (i.e. prior to 2012). The drill core was logged following the mine/industry standard at the time. Geological mine level sections were marked where core sections were visually assessed to be "mineable". A check sample was then taken with a follow –up analysis in the onsite laboratory. Recent re-analysis of 22 of the historical holes that had assay data available has been plotted against the re-assayed results. These results indicate that Fe has been under represented but is consistent <15 Fe % error which means there is reasonable confidence levels in the historical analysis (Figure 11-8). Phosphorous, however, was not representative, as the plot shows there was an error <40 % which indicates historical phosphorus data cannot be used in the current resource model. (Figure 11-9).

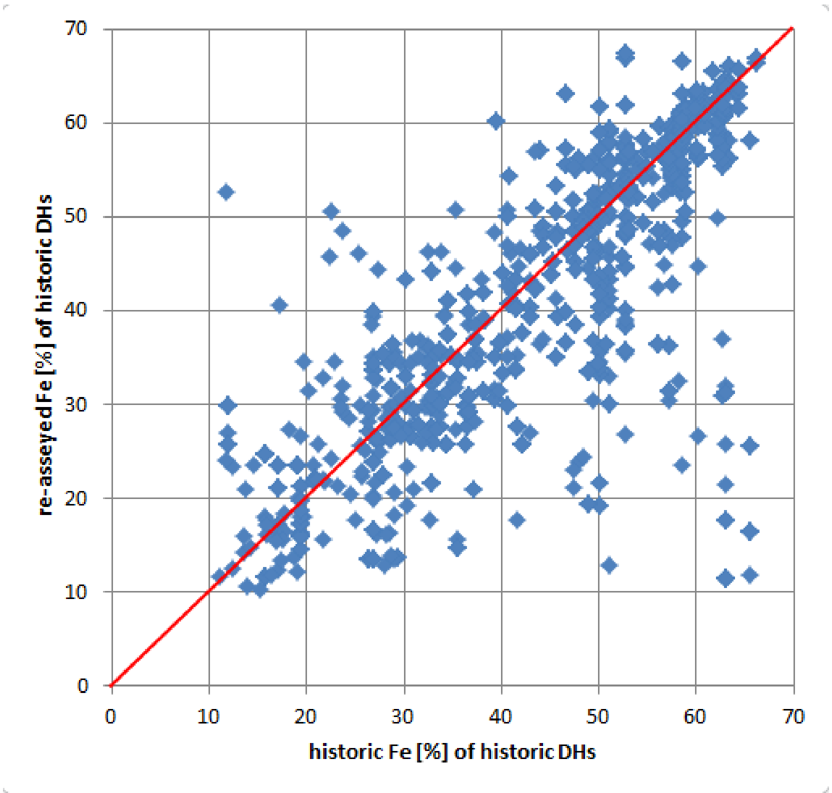


Figure 11-8 **Correlation plot –Fe Re-Assay % vs. Fe Historic %**

Source: NIO

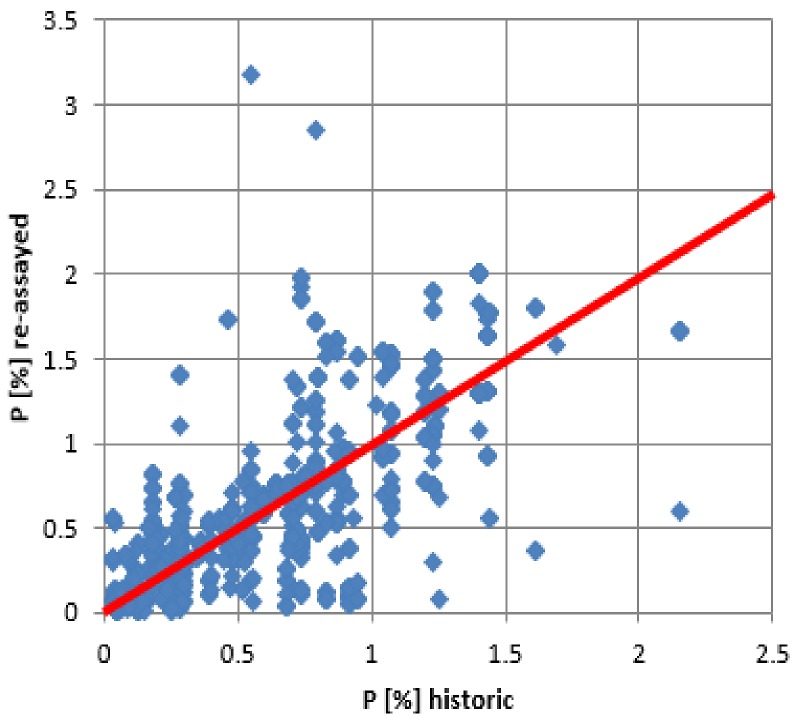


Figure 11-9 **Correlation plot –Phosphorous Re-Assay % vs. Phosphorous Historic %**

Source: NIO

The ALS Chemex laboratory in Vancouver used by NIO is accredited to ISO/IEC 17025:2005 standards.

All of the methods of analysis used in 2012 and 2014 are considered appropriate by DMT for iron ore projects of this type.

All results from every analysis were recorded within an MS Excel worksheet, checked using the QA/QC data then and transferred to the NIO Blötberget Access database. All coarse rejects, pulps and drill cores are then fully catalogued and stored in Grängesberg for reference at a later date.

11.6.2 Equipment Accuracy & Tolerances

The analytes and ranges for the ME- ICP61a, ME-XRF21n and XRF15b equipment at ALS Vancouver are illustrated in Table 11-2.

Table 11-2 Analytes and ranges for ME- ICP61a, ME-XRF21n and XRF15b from ALS Labs

ANALYTES & RANGES (ppm)								CODE
Ag	1-200	Cr	10-100,000	Na	0.05%-30%	Ti	0.05%-30%	ME-ICP61a
Al	0.05%-30%	Cu	10-100,000	Ni	10-100,000	Tl	50-50,000	
As	50-100,000	Fe	0.05%-50%	P	50-100,000	U	50-50,000	
Ba	50-50,000	Ga	50-50,000	Pb	20-100,000	V	10-100,000	
Be	10-10,000	K	0.1%-30%	S	0.05%-10%	W	50-50,000	
Bi	20-50,000	La	50-50,000	Sb	50-50,000	Zn	20-100,000	
Ca	0.05%-50%	Mg	0.05%-50%	Sc	10-50,000			
Cd	10-10,000	Mn	10-100,000	Sr	10-100,000			
Co	10-50,000	Mo	10-50,000	Th	50-50,000			

ANALYTES & RANGES (%)						DESCRIPTION	CODE
Al_2O_3	0.01-100	K_2O	0.001-6.3	Sn	0.001-1.5	Fused disc XRF	ME-XRF21n (normalized) ME-XRF21u (un-normalized)
As	0.001-1.5	MgO	0.01-40	Sr	0.001-1.5		
Ba	0.001-10	Mn	0.001-25	TiO_2	0.01-30		
CaO	0.01-40	Na_2O	0.005-8	V	0.001-5		
Cl	0.001-6	Ni	0.001-8	Zn	0.001-1.5		
Co	0.001-5	P	0.001-10	Zr	0.001-1		
Cr_2O_3	0.001-10	Pb	0.001-2	Total	0.01-110		
Cu	0.001-1.5	S	0.001-5				
Fe	0.01-75	SiO_2	0.01-100				

ANALYTES & RANGES (%)						CODE
Al_2O_3	0.01-100	La_2O_3	0.01-50	Sn	0.005-20	ME-XRF15b*
As	0.01-10	MgO	0.01-40	Sr	0.01-5	
BaO	0.01-66	Mn	0.01-30	Ta	0.002-16.4	
Bi	0.01-5	Mo	0.005-2	Th	0.002-5	
CaO	0.01-40	Nb	0.005-20	TiO_2	0.01-30	
CeO_2	0.01-50	Ni	0.005-20	U	0.001-5	
Co	0.01-7	P_2O_5	0.01-25	V	0.01-5.6	
Cr	0.01-10	Pb	0.005-20	W	0.001-15.9	
Cu	0.005-20	S	0.01-20	Zn	0.005-20	
Fe	0.01-75	Sb	0.005-20	Zr	0.01-20	
K_2O	0.01-6.3	SiO_2	0.01-100			

Source: ALS

11.6.3 Check Samples

Check samples have consisted of certified standards, blanks and duplicates of previously assayed samples.

No QA/QC samples were analysed for the re-assayed historic core project undertaken by BGU at Blötberget.

During the 2012 drilling and sampling campaign, samples used for quality assurance and quality control purposes were inserted, on average, one every 20 samples (5% insertion rate). During the 2014 drilling programme this changed to one in every 15 samples (6.7% insertion rate) for each type of check sample, namely; standards, duplicates and blanks.

Standards were inserted dependant on the Fe grade of the material that was being sampled.

QA/QC samples were inserted at a rate of 6 %, comparing favourably to the documented protocol. In total, 45 duplicates, 8 blanks at 1 % Fe, and 42 at 2 % Fe, 17 GIOP-94, 15 GIOP-120, 9 GIO-48 and 2 GIOP-126, standards were analysed. The results of the analyses are plotted in figures below.

The potential effects of bias in the sampling of the 2012 and 2014 core, where different proportions of core were used at different times, has not been investigated to date.

11.6.3.1 Duplicates

The duplicate analyses show an excellent correlation between original and duplicate % Fe Total, magnetite and phosphorus results.

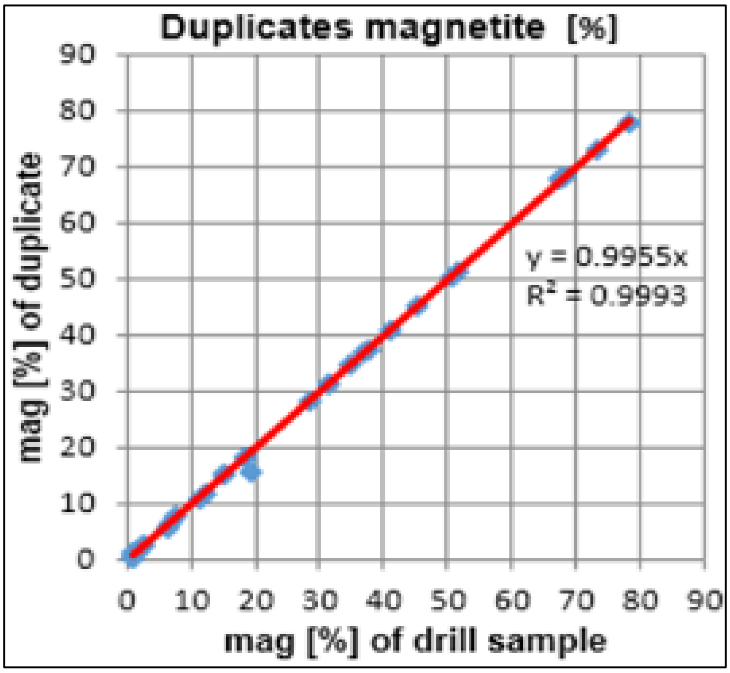


Figure 11-10 **Duplicate analyses – Magnetite**

Source: NIO

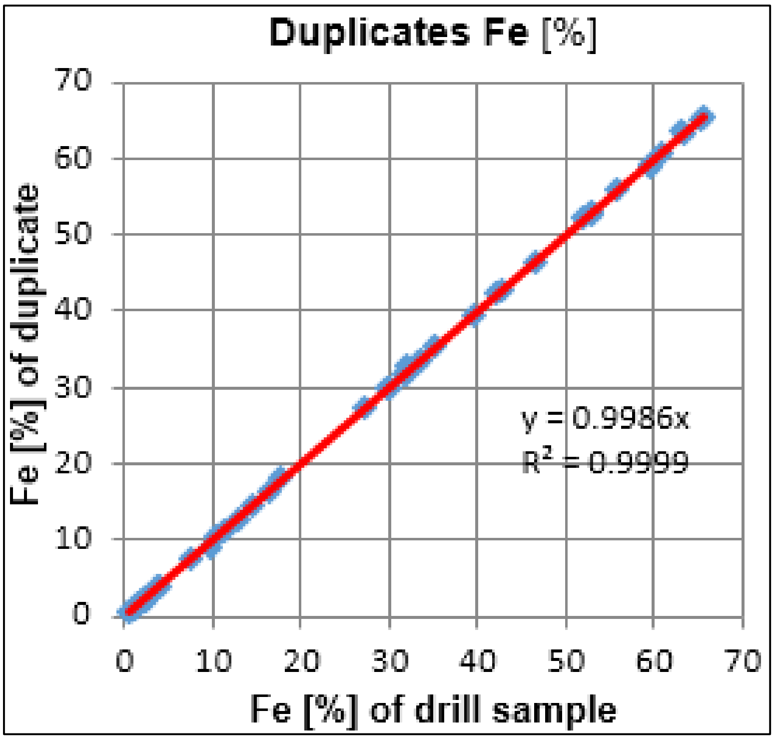


Figure 11-11 Duplicate analyses – Fe Total

Source: NIO

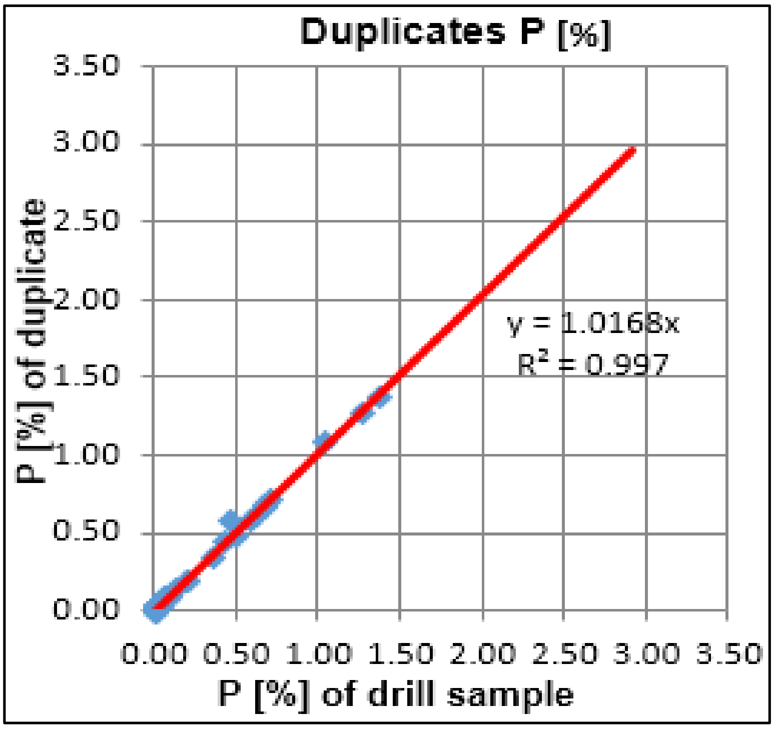


Figure 11-12 Duplicates analyses – Phosphorus

Source: NIO

11.6.3.2 Blanks

There are two blanks in the NIO database; the NIO blank material is a quartz-rich sand from a nearby quarry, and the ALS Chemex blank material is a quartzite. The first blank was used until mid-way through the 2012 drilling programme which averaged 1 % Fe, and the second blank that replaced the first blank standard averages 2 % Fe.

The blank analyses show two separate trends, with the initial blank standard grading ~1 % Fe Total and the replacement blank which grades at ~2 % Fe Total. The results of the two blank materials used (NIO and ALS) are shown in Figure 11-13. The red line on the graph indicates the point of change of the blank materials in 2012.

The results indicate that there have been no contamination issues.

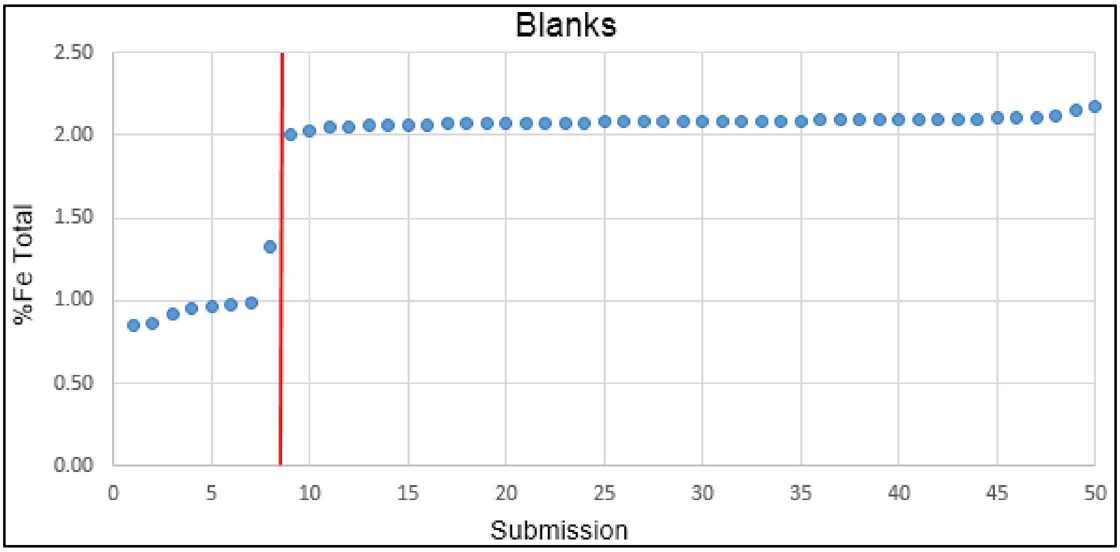


Figure 11-13 Blanks analyses - % Fe Total

Source: NIO

Assay results of blank material show significant amount of Fe and magnetite. Due to the good reproduction demonstrated from duplicates and certified standards, it is considered by DMT that the preparation of the blank sample requires more attention.

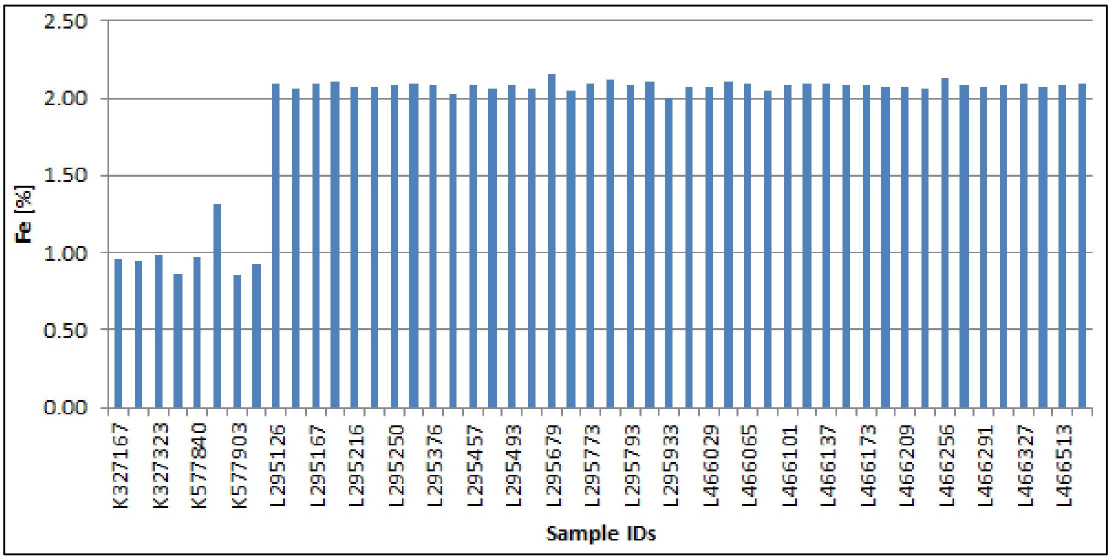


Figure 11-14 **Assay results of blank samples - Iron**

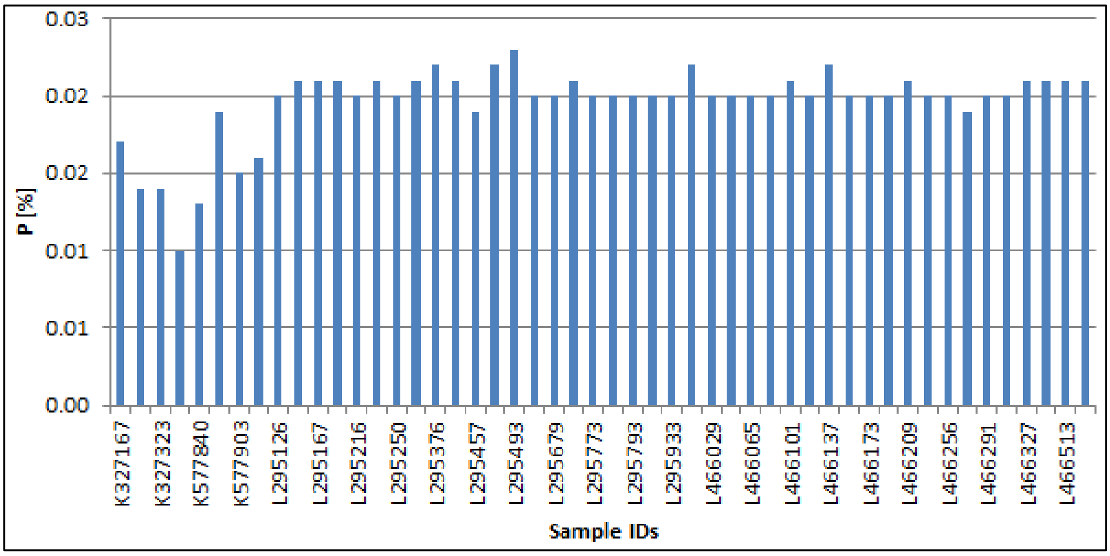


Figure 11-15 **Assay results of blank samples - Phosphorous**

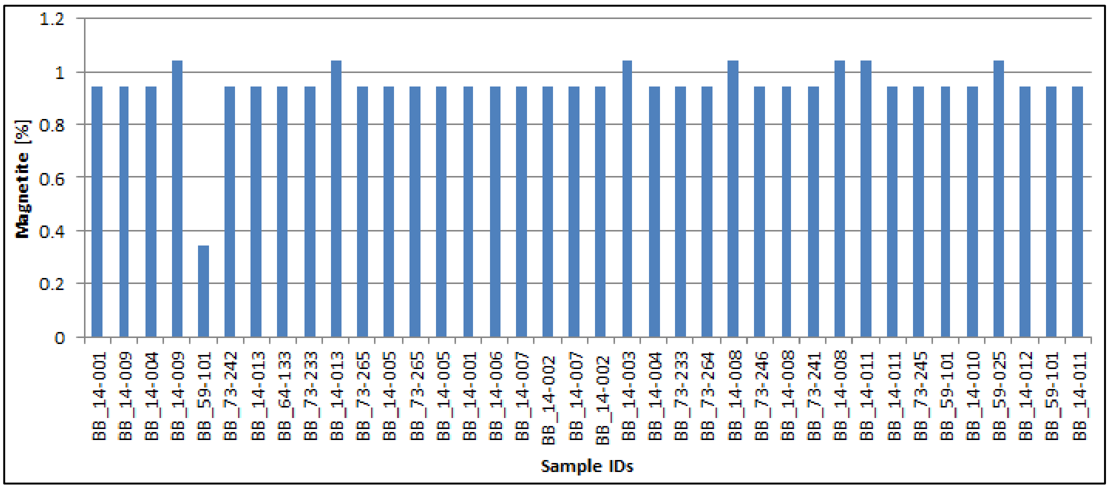


Figure 11-16 **Assay results of blank samples – Magnetite**

Sources: NIO

11.6.3.3 Certified Standards

The standards represent an appropriate spread of % Fe Total grades, as shown below:

- **GIOP-126** (certified mean of 49.61% Fe Total)
- **GIOP-120** (certified mean of 2.83% Fe Total)
- **GIOP-94** (certified mean of 23.97% Fe Total)
- **GIOP-48** (certified mean of 45.93% Fe Total)
- **OREAS-701** (certified mean of 23.98 % Fe Total containing Fe_3O_4 17.95%) (For use calibrating the SATMAGAN)

The four standards used show a slight negative bias to the data compared to the certified mean and the lower confidence limits set by the manufacturer. The majority of the data fall within errors of < 1% for iron and phosphorous and <2 % error for magnetite from the SATMAGAN analysis of the confidence intervals given by the standards manufacturer.

According to NIO, the standards often fall outside the lower or upper confidence levels. In this case, the total error of digestion and analysis for Fe is <1% for laboratory work (not 1 % Fe, 1 % total error).There is no indication of a systematic error by applied digestion or analysis method. Viewed in the context of the overall resource estimate error, the <1% error of the standards is deemed acceptable.

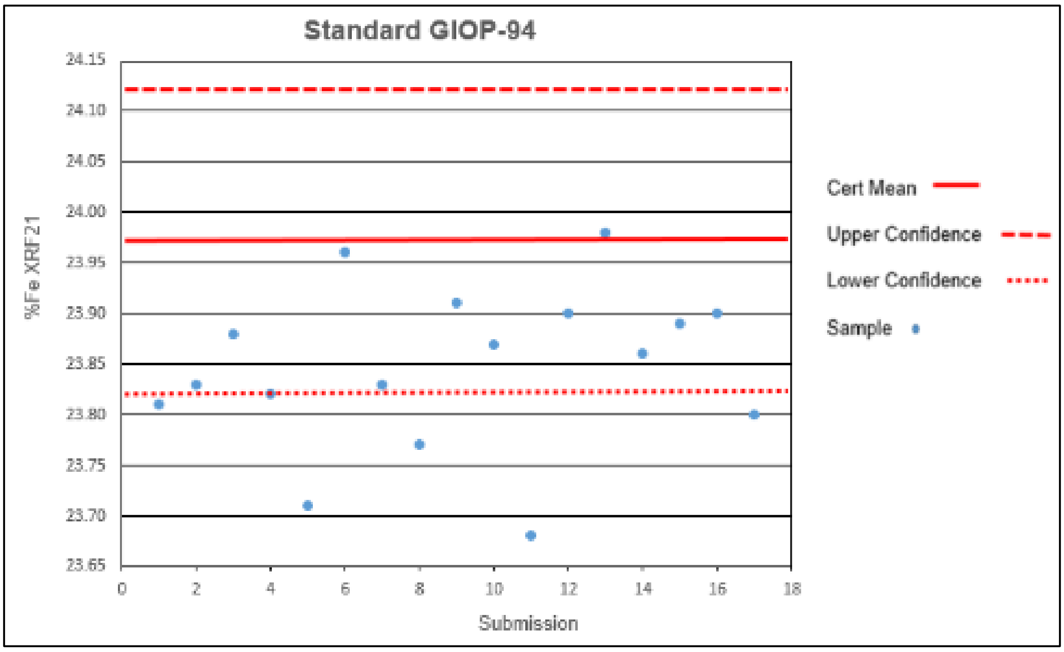


Figure 11-17 **Standards analyses – GIOP-94**

Source: NIO

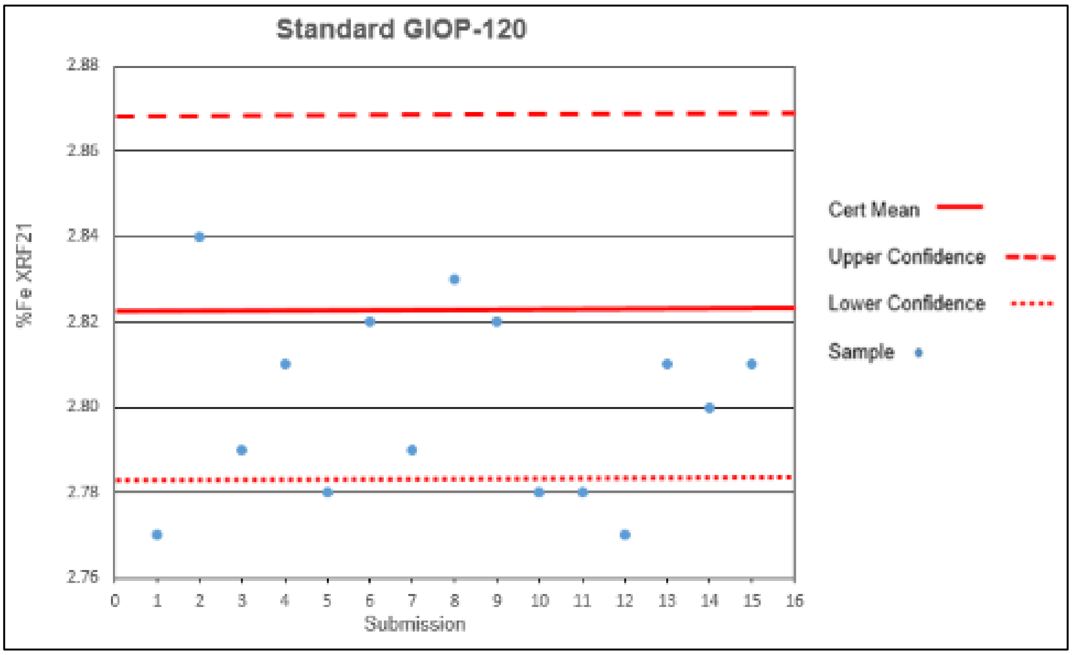


Figure 11-18 Standards analyses – GIOP-120

Source: NIO

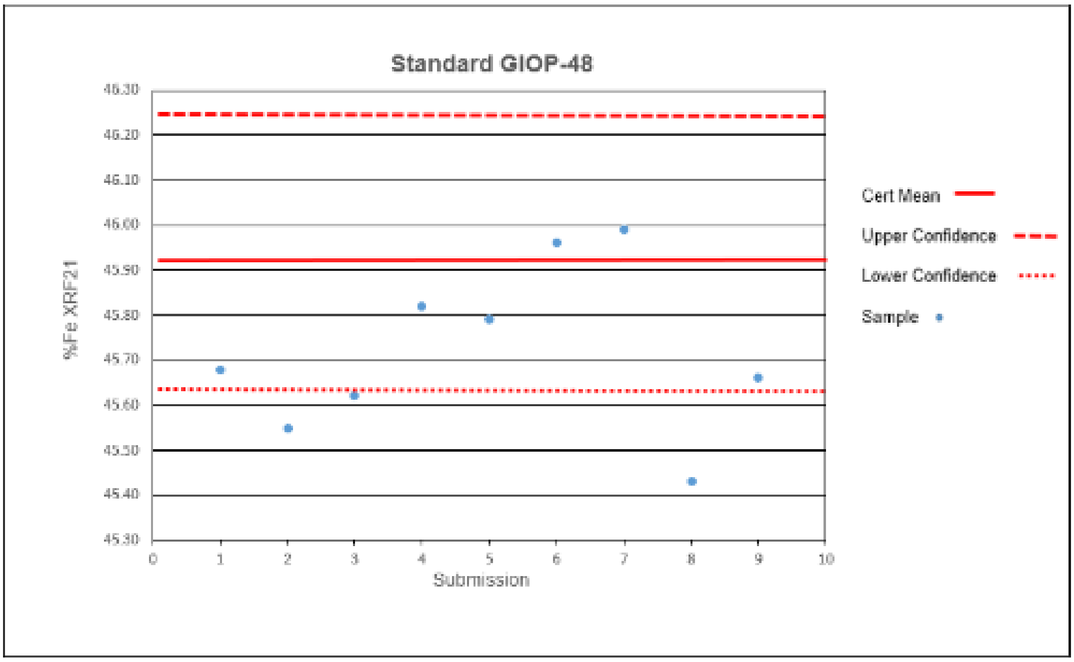


Figure 11-19 Standards analyses – GIOP-48

Source: NIO

Figure 11-20 Standards analyses – GIOP-126

Source: NIO

11.6.4 SATMAGAN

Before the SATMAGAN is used, it is calibrated using the 10 supplied calibration samples from the manufacturer, Rapiscan. These calibration samples are 0, 10, 20, 30, 40, 50, 60, 70, 80, 90 and 100 % Fe bound to magnetite.

During analysis, the Standard OREAS 701 (provided by Ore Research and Analysis Australia) is inserted 1 in 5 samples (20 %). This standard has a certified mean of 23.98 % Fe Total and contains Fe_3O_4 17.95 %. NIO's blank sample and a sample of duplicate material created during sample selection by the geologists is also inserted at a rate of 1 in 5 (20 %). The SATMAGAN is factory calibrated annually by Holger Andreasen AB in Örebro, Sweden.

11.6.5 Magnetic Susceptibility

The KT-10 Magsus equipment is factory calibrated each year by Radiation Detection Systems AB in Falun, Sweden.

12 DATA VERIFICATION

12.1 Introduction

The accuracy and precision of data acquisition methods has been verified by DMT in order to assess the adequacy, reliability and representativeness (reproducibility) of the resulting data.

The verification was focused on the following data:

- Drilling location (collar) and drillhole orientation information to confirm the correct position of samples;
- Drilling and sample recovery in order to verify unbiased analytical results;
- The QA/QC sample set (Certified Standards, Blanks and Duplicates); implemented in each sample batch in order to verify the representativeness of results produced by sample preparation, digestion and chemical and mineralogical analysis;
- Davis Tube Recoveries in order to verify the magnetite data obtained by NIO using SATMAGAN instrumentation; and
- Density determinations.

All data from 2012 and 2014 have been acquired based on standard operating procedures ("**SOPs**"). Surface historical drillholes have been re-surveyed and a number of historical drillholes have been re-logged and re-assayed.

The location of all historic drillholes and geological underground maps have been converted from a local mining grid to map datum SWEREF99-TM. A local height reference system has been established, the current "zero" for which is 226.15 m below the RH2000 height system. The work of geo-referencing and coordinate conversion has been done by the Tyréns Company, Sweden.

Drillhole collar surveying was completed by Ludvika Kommun ("**LK**") after the completion of the 2012 and 2014 drilling programmes. LK surveyed drillhole collar locations (X, Y, and Z), dip and azimuth using high resolution Real Time Kinetic ("**RTK**") Global Positioning Systems ("**GPS**"). During the 2014 programme, LK also re-surveyed historic drillhole locations to confirm the translation of historic coordinates.

All coordinates and height data given in this report are in the projected reference system SWEREF99-TM + RH2000 minus 227.95 m.

12.2 Data Availability

12.2.1 Drill Hole Data

The license area has been investigated by several historic and recent drilling programmes.

Table 12-1 and give an overview of the scope of available data acquired from historic drillholes and drillholes from the 2012 and 2014 drilling programmes.

As part of the verification exercise, NIO has re-logged and re-assayed many of the located historical cores. In total, 45 drillholes from Blötberget were found in Malå and 15 at the former mine storage facility in Håksberg.

There has been re-logging of 31 of these cores (6036 m), 950 m of mineralisation has been re-sampled and re-assayed according to current industry practice and standards. This included mineralised core that had not been sampled historically as it fell below the (visual) historic cut-off grade of 35 %. Approximately 5-10 m of mineralised core was sampled beyond the boundaries of the historical sampled sections.

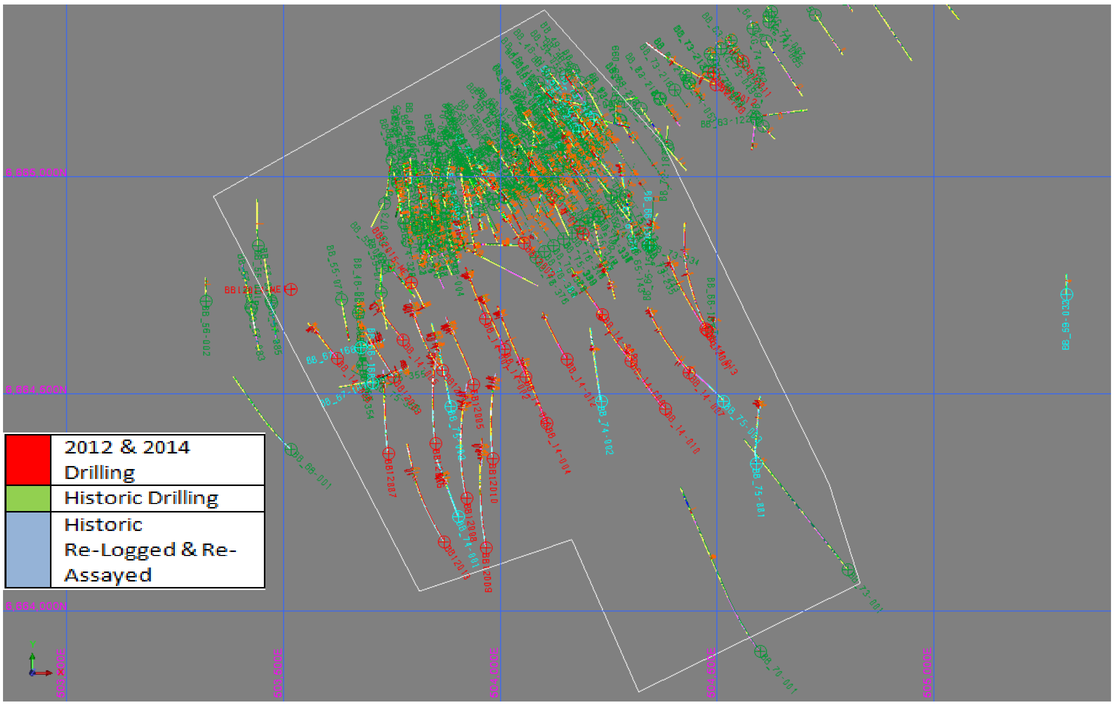


Figure 12-1 **Plan of drillholes in license area (white border)**

Source: DMT

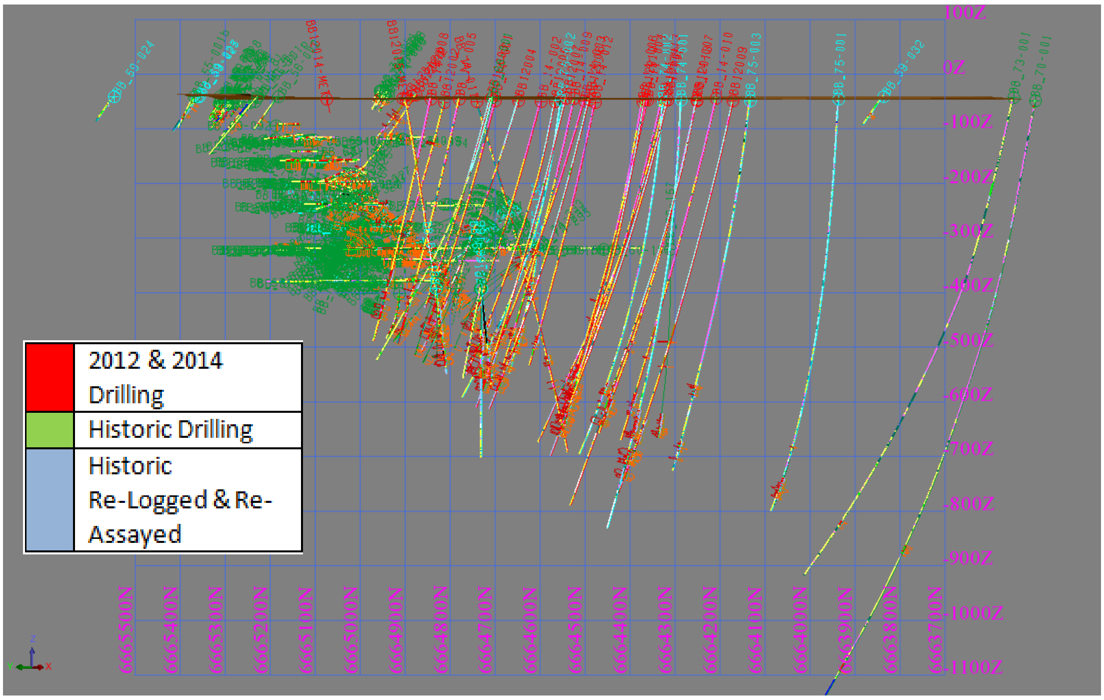


Figure 12-2 **Section View W-E (Direction 55°N) of drillholes**

Source: DMT

Table 12-1 Summary of data validation (drilling and sampling)

Type	Drillholes with known location and orientation	Metres drilled	Drillholes with down hole deviation data	Drillholes with geological data	Metres geologically logged	Drillholes with chemical data	Metres chemically analysed	Drillholes with magnetite analysis (SATMAGAN and mag sus)	Metres of magnetite analysis (SATMAGAN and mag sus)	Drillholes with density data	Metres of density data
Historic	391	32,750.59	8	161	19,389.61	329	6,036.61	13	326.32	22	703.14
Historic Re-logged / Re-assayed				29	6036	31	950	31	950	26	780
2012	16	7,426.05	16	14	7,385.65	13	621.55	13	600.25	13	620.00
2014	13	7,092.83	13	13	7,046.77	13	549.86	13	549.86	13	548.14
Total	**420**	**47,269.47**	**37**	**188**	**33,822.03**	**355**	**7,208.02**	**39**	**1,476.43**	**48**	**1,871.28**

Source: DMT

Table 12-2 Summary of drill hole data available in interpreted wireframes

Wireframes	Number of Drillhole Intersections	Metres of Drillhole Intersections	Sampled metres assayed by ALS	Metres with digitised Fe results	Total metres with Fe results	Sampled metres with SATMAGAN (Mg) analysis	Metres of density data
Hugget/ Flygruven	273	4806	769	2312	3081	769	714
Kalgruven	103	1579	447	571	1018	447	421
Sandell	18	121	18	73	91	18	18
Total	**394**	**6506**	**1234**	**2955**	**4190**	**1234**	**1153**

Source: DMT

12.2.2 License Area

The license area 'Blötbergsgruva K nr 1' (ID: 2010001141) held by NIO covers an area of 1.26 km². Table 12-3 shows the coordinates of the boundary line defining this license area. These coordinates are given in map datum SWEREF99-TM, as digitised by Tyréns, on behalf of NIO.

Ownership and license status has not been independently verified by DMT.

Table 12-3 Coordinates of licence area in map datum SWEREF99-TM.

Licence Area Boundary Point	Easting	Northing	Licence Area Boundary Point	Easting	Northing
1	504 319.11	6 663 813.90	5	504 102.01	6 665 384.70
2	504 164.88	6 664 164.89	6	504 374.64	6 665 079.13
3	503 813.46	6 664 045.65	7	504 758.10	6 664 293.09
4	503 338.52	6 664 955.53	8	504 829.87	6 664 064.04

Source: DMT

12.2.3 Topography

A Digital Terrain Model ("**DTM**") has been prepared by Tyréns, on behalf of NIO. This DTM is a triangulation based on contour lines with 5 m spacing. The morphology in the license area has a relatively low topographic range (-46 m to -36 m); predominantly on the -46 m, as illustrated in Figure 12-3.

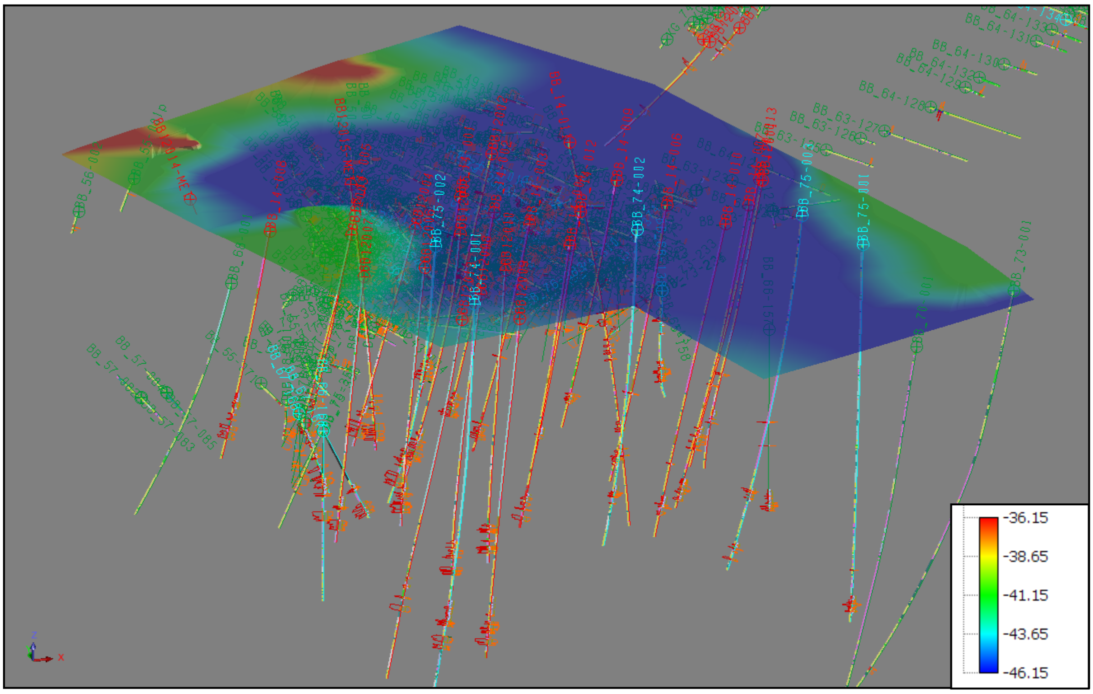


Figure 12-3 DTM of license area

Source: DMT

12.2.4 Historical Mine Maps & Sections

In accordance with Swedish mining regulations, 1:800-scale maps and level plans were kept updated, during historic periods of mining, and these have been scanned and georeferenced.

Only the horizontal maps have been ortho-rectified and imported into Surpac. The sections were only used for interpretation and were digitised by Tyréns on behalf of NIO (Figure 12-4 and Figure 12-5)

A tabulation of the historic geological maps and sections that have been utilised are given in Appendix B.

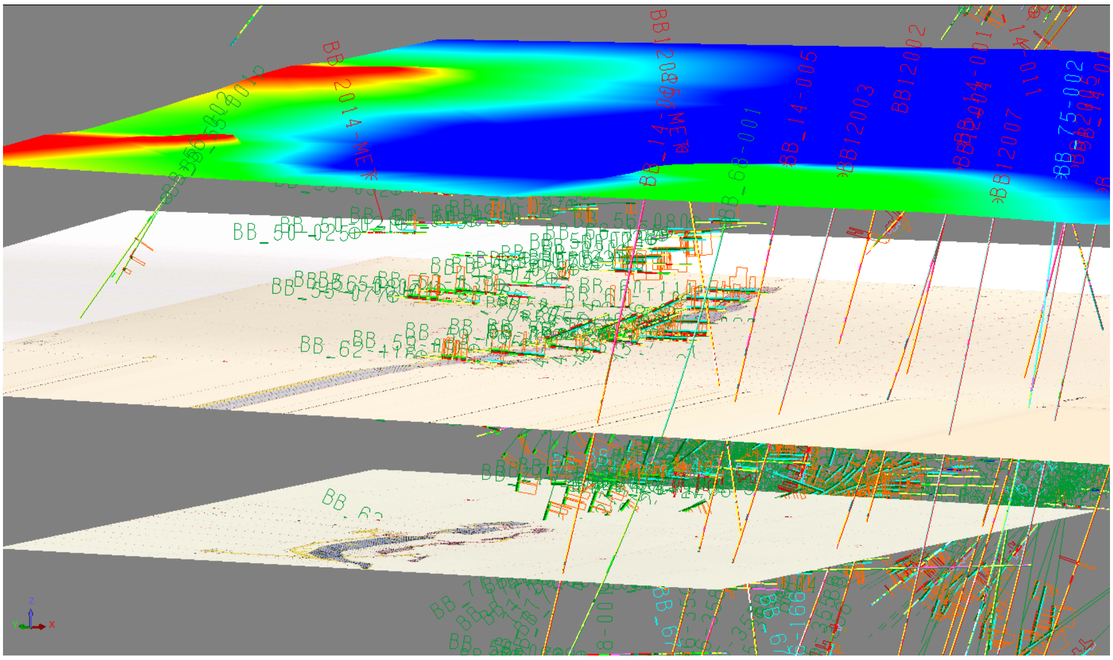


Figure 12-4 Geo-referenced historic level plans

Figure 12-5 Geo-referenced level plan within the wireframe

Geological maps and underground level mine plans have been used to estimate the maximum depth level of historical mining activities and the volume of mined out material.

This exercise was undertaken by NIO technical personnel and a surface was supplied to DMT showing the depth limit of mining activities (Figure 12-6).

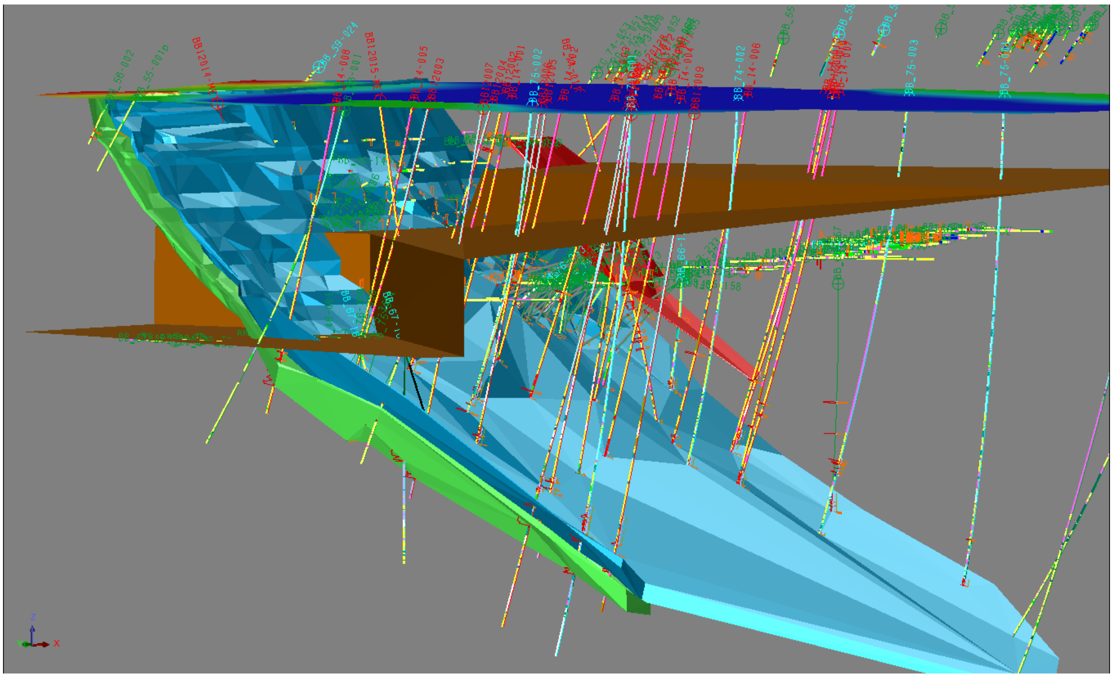


Figure 12-6 Topographical surface (dark blue) and lowest mined level (brown)

12.2.5 Drill Hole Locations & Orientation

Collar locations and elevations from the 2012 and 2014 drilling programmes have been surveyed by a registered surveyor contracted by NIO. Some differences were identified between surveyed collar elevation and elevation of topography of between 1 m and 3 m. This variance in elevation does not significantly affect the thickness of the respective wireframe and related domain intersections.

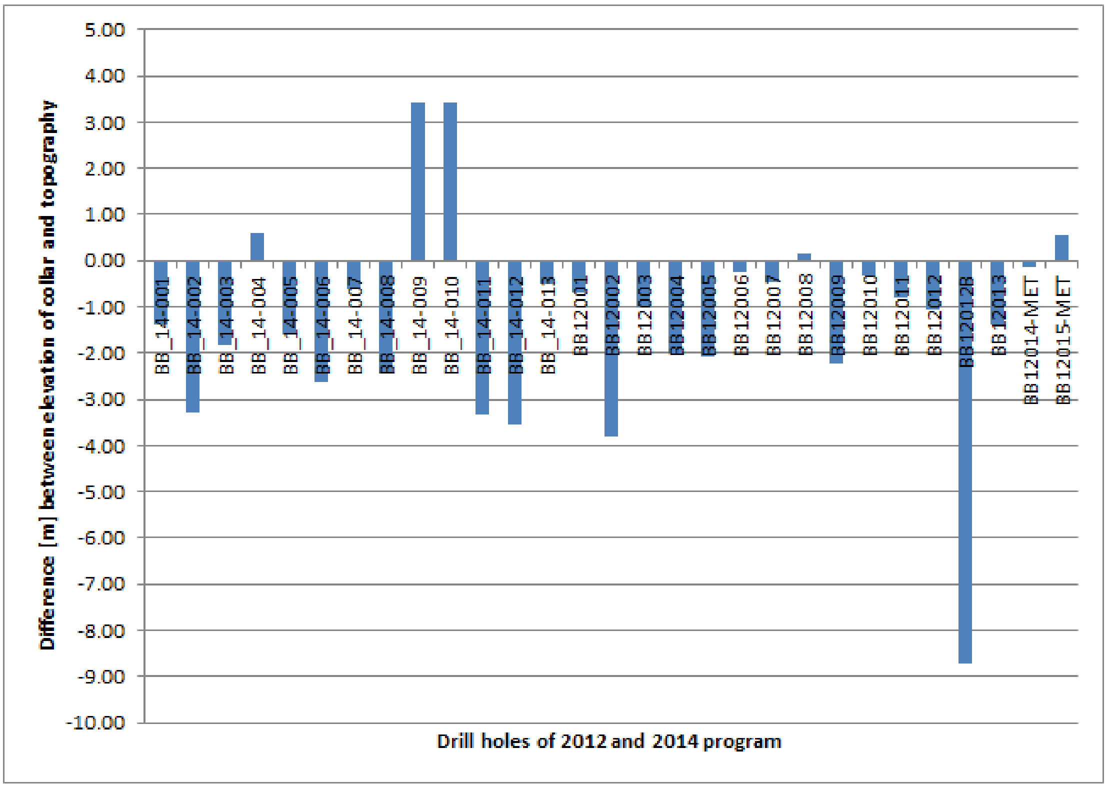


Figure 12-7 Offset of surveyed surface collar elevation from terrain model

Note: Negative values: collar elevation is below topography

Positive values: collar elevation is above topography

The collar locations, dips and azimuths of the underground historical drillholes, in all but a few instances, are very close to the information given in the historical protocols. The error in location is typically less than 0.5 m. For a small number of drillholes, no collar information has been found in the records. As a result, their locations have been taken from the plans.

12.3 Data Preparation & Management

All data from the historical and recent drilling programmes has been stored in an industry standard database software that includes the following information:

- Collar survey;
- Down hole survey;
- Geology (and abbreviation codes);
- Sampling;
- Laboratory assay data;
- Digitised chemical data;
- Magnetite data of SATMAGAN;
- Davis tube recovery;
- QA/QC sample set; and

- Bulk density.

The database has been rigorously checked by DMT for completeness and error for each drillhole and cross checked with the core photographs. All data have been exported and implemented into an industry standard (Geovia Surpac) modelling software package.

12.3.1 Drilling Recovery

Drilling recovery for the 2012 and 2014 drilling programmes is typically very close to 100 %. Consequently, any artificial bias caused by poor core and sample recovery can be excluded for the 2012 and 2014 drilled holes. For the historic drilling, no data relating to core recovery is available.

12.3.2 Historical Chemical Data

In total, 22 historical drillholes with chemical data of Fe and/or P were re-sampled and re-assayed. These drillholes have been used to verify the Fe and P data of historical drillholes (Table 12-4).

Table 12-4 Historical drillholes re-sampled

Item	Drillhole ID	Item	Drillhole ID
1	BB_59-101	12	BB_73-245
2	BB_59-103	13	BB_73-246
3	BB_64-134	14	BB_73-264
4	BB_66-138	15	BB_73-265
5	BB_66-157	16	BB_74-001
6	BB_66-166	17	BB_74-002
7	BB_67-167	18	BB_74-310
8	BB_67-168	19	BB_75-001
9	BB_73-236	20	BB_75-002
10	BB_73-241	21	BB_75-003
11	BB_73-242	22	BB_76-383

Figure 12-8 and Figure 12-9 show correlation plots comparing re-assayed Fe and P with historic Fe and P.

The data of Fe could be reproduced with an error of 12 %. No systematic error could be observed in the majority of the data. The data of P could be reproduced with an error of 40 %.

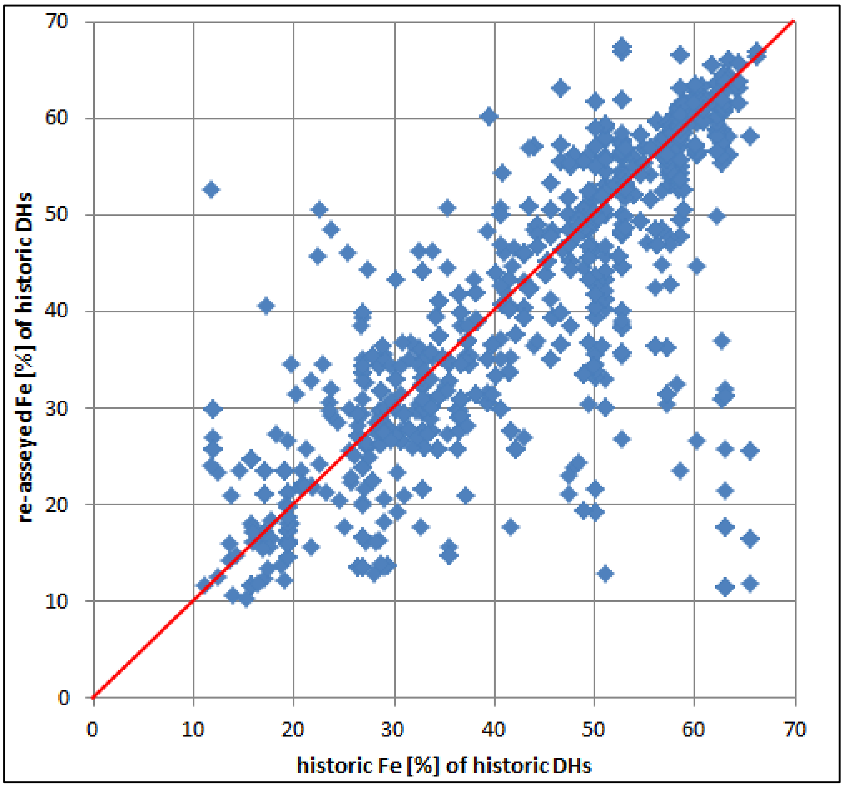


Figure 12-8 Correlation plot of re-assayed Fe vs. historic Fe

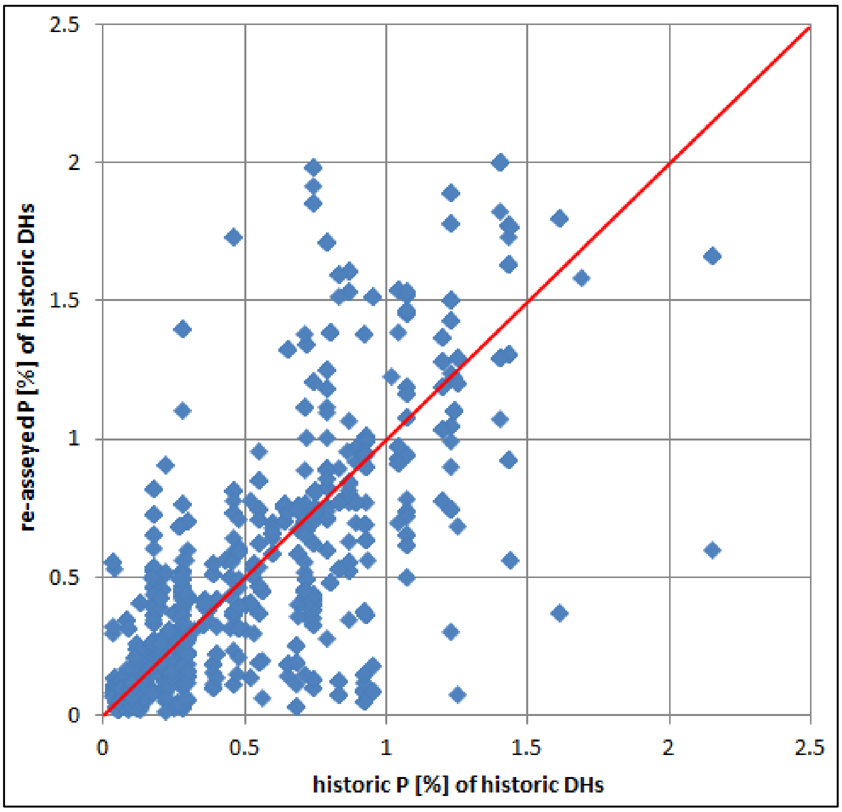


Figure 12-9 Correlation plot of re-assayed P vs. historic P

12.4 Comments on Data Quality

The accuracy and precision of applied sample preparation and assaying method has been verified by DMT and the resulting data for Fe, P and magnetite have been assessed as reliable and representative to be used in resource modelling.

While the historic Fe data are assessed as acceptable to be used in the resource model, the historic P data were not considered acceptable and were not used in the resource model and subsequent resource estimate.

The drilling recoveries from the 2012 and 2014 drilling programmes is close to 100 %. Consequently, sample or assay bias caused by poor core recovery is negligible.

A slight offset in sample location or orientation will not have any influence on the resource estimate.

No significant risks with the underlying data, used for mineral resource estimation, were not identified by DMT.

13 MINERAL RESOURCE ESTIMATE

13.1 Introduction

DMT has built a new resource block model for an updated MRE for the Blötberget Project.

The MRE has been prepared in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves 2012 ("**the JORC Code**").

The geological model was prepared using an exploration database cut-off date of the 1st January, 2015. The resource estimate itself, has an effective date of the 30th January 2015, (issue date of 10th April 2015).

13.2 Geological Model

The geological interpretation has been based on the geological environment, deposit type and geological features controlling style and characteristics of the mineralisation.

The majority of the iron bearing lenses or zones at Ludvika are classified as magnetite rich lava flows, hosted by the Svecofennian, 1.91–1.89 Ga felsic metavolcanic rocks and generally form seam-shaped bodies. The flows are occasionally of pure magnetite, with additional detrital magnetite units assumed to be volcaniclastic sediments.

For the Blötberget area it is reported that the mineralisation relates to sub-aerial terrestrial volcanism. This has caused a partial oxidation of the primary magnetite mineralisation and hence produced large areas of martite (haematite formed after replacement of magnetite) mineralisation (*GeoVista Resource Estimate, January 2014*).

Prior to geological modelling, a series of cross sections were defined perpendicular to the strike direction of 55°N of all of the mineralised zones within the license area. Geological interpretation was carried out on cross sections at varying intervals dependant on drill spacing.

13.3 Wireframe Modelling

The interpretation followed the geological concept of a laterally continuous seam-like geometry, which is flexured along the dip direction of 145º with dips ranging from 50º at the surface to 35º at a depth of 800 m below surface.

Three main iron rich zones lie as narrow mineralised envelopes, from the upper (hanging wall) zone to lower (footwall) zone these three zones are referred to as:

- Sandell ("**SAND**");
- Hugget-Flygruvan ("**HUGFLY**"); and
- Kalvgruvan ("**KALV**").

Each of these zones required wireframe modelling for grade estimation purposes.

Individual hanging and footwall triangulated surfaces were created based on drillholes intersecting the mineralised zones and a set of underground maps of historic mining.

The surfaces were extended with half the distance to nearest drillhole as lateral limit of mineralisation. A fully enclosed 3D triangulated solid of each zone was achieved by cross-linking the boundary strings. A 15 % Fe cut-off grade has been applied to model the contacts of the mineralised zones. Some intersections did not show a composite grade above 15 % Fe. These low grade intersections were also included in the mineralised zone in order to honour the lateral continuity of the seam-like lava flow model.

The three solid models are representing the most optimistic envelopes which also consider waste material and low grade ore.

The shape and orientation of the mineralisation and the geological and mineralogical data (including dip and dip direction) suggest that there is no additional tectonic influence on the distribution of mineralisation.

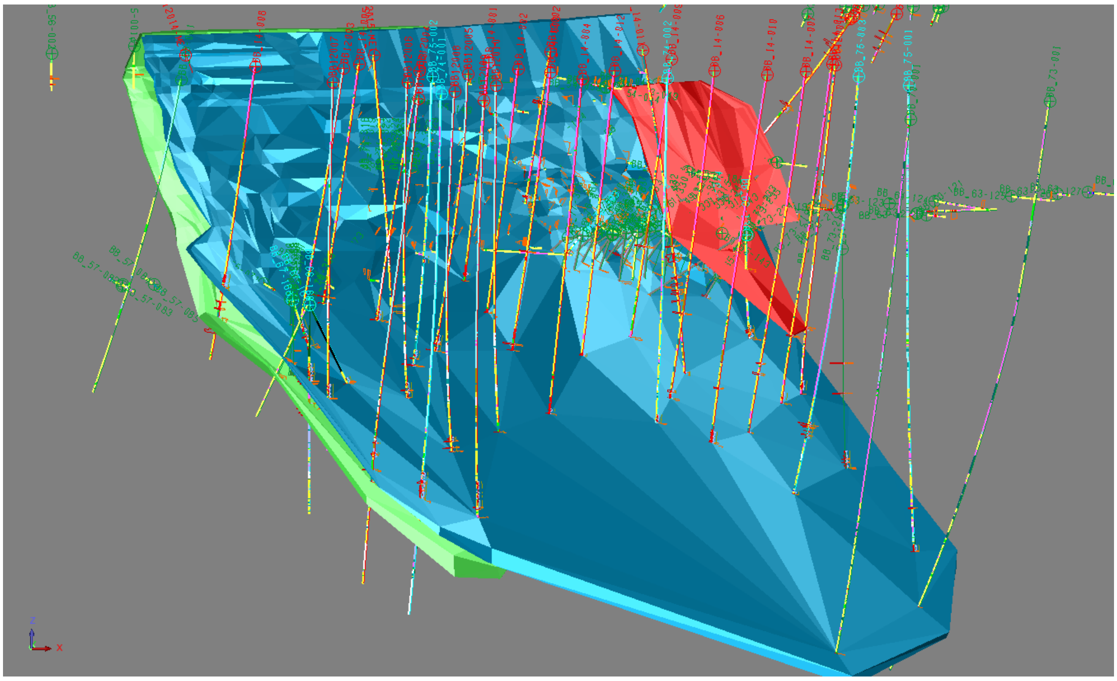


Figure 13-1 Wireframe 3D view North

Note: SAND (red), HUGFLY (blue), KALV (green)

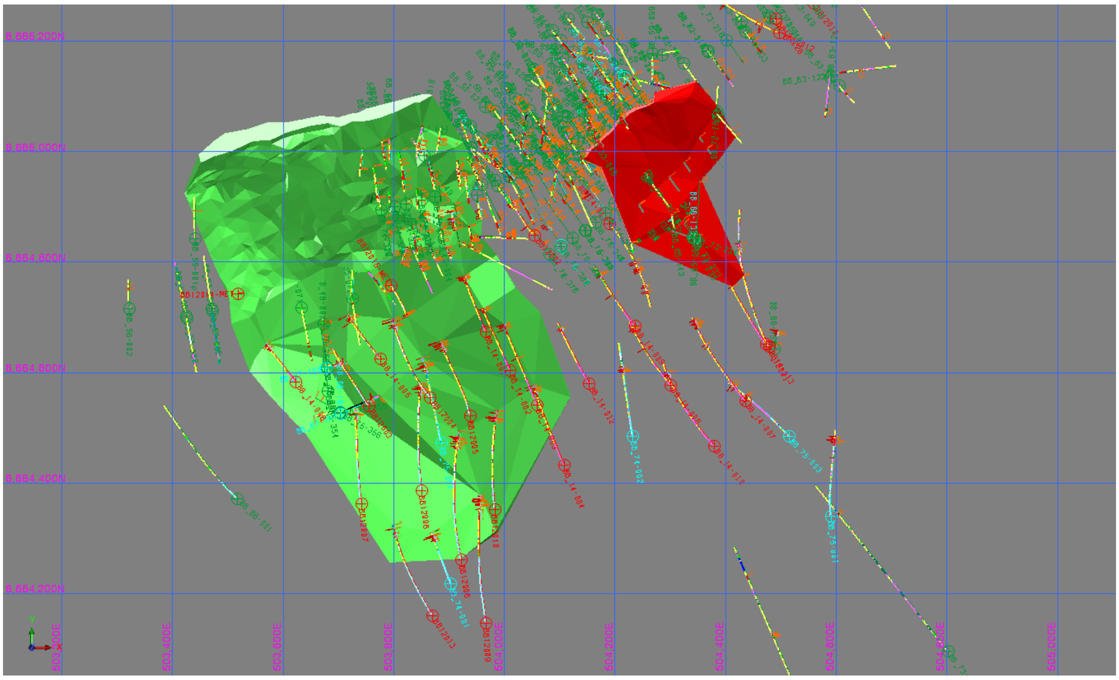


Figure 13-2 Wireframe 3D of the mineralised envelopes: (HUGFLY removed)

Note: SAND (red), HUGFLY (not shown), KALV (green)

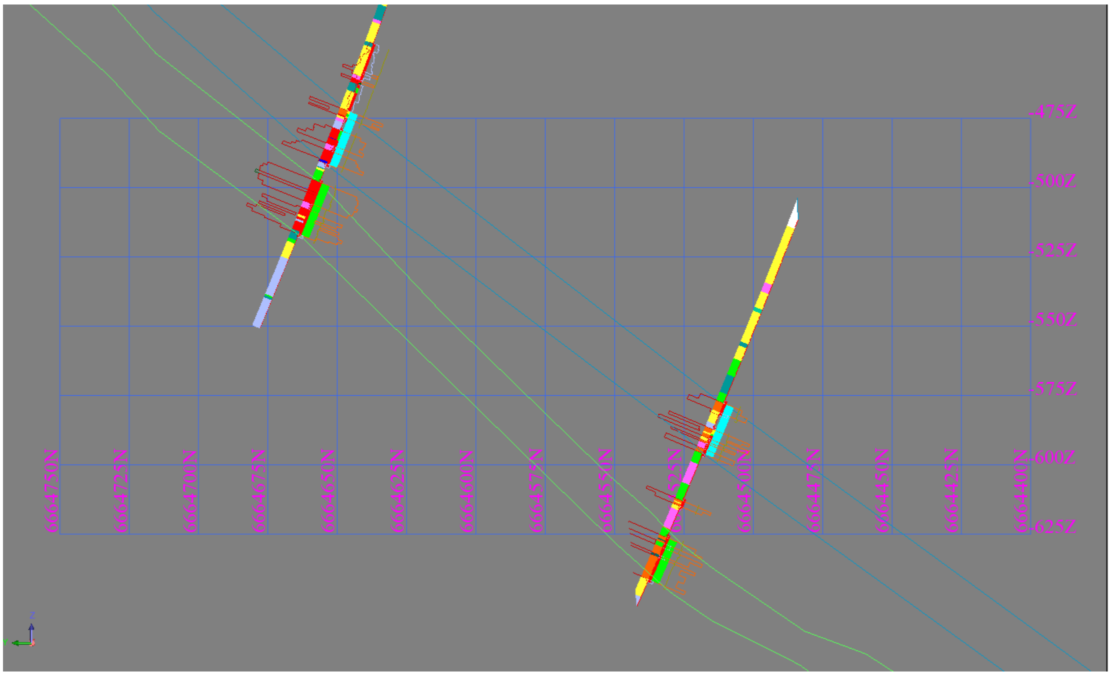


Figure 13-3 Cross section looking NE through HUGFLY

13.3.1 Topography

A DTM has been prepared by Tyréns (Sweden) on behalf of NIO. This DTM is a triangulation based on contour lines with 5 m spacing. The morphology in the license area has a relatively low topographic range (-46 m to -36 m), predominantly on the -46 m.

13.3.2 Weathering Profile

It is assumed that there is no weathering profile affecting the current geological model.

13.4 Resource Database

Table 13-1 Summary of drillholes and data used in the Resource Estimate

Wireframe domain	Number of Drillhole Intersections	Metres of Drillhole Intersections	Sampled metres assayed by ALS	Metres with digitised Fe results	Total metres with Fe results	Sampled metres with SATMAGAN (Mg) analysis	Metres of density data
Hugget / Flygruvan	273	4806	769	2312	3081	769	714
Kalvgruvan	103	1579	447	571	1018	447	421
Sandell	18	121	18	73	91	18	18
Total	394	6506	1234	2955	4190	1234	1153

13.5 Statistical Analyses & Geostatistics

Based on the wireframes of HUGFLY, KALV and SAND, univariate and bivariate statistical analyses were carried out in order to investigate the non-spatial element distribution (histograms) and inter-element relationships (correlation plots).

A correlation plot of magnetite and iron (Fe) demonstrates that the magnetite of KALV follows a regression equation based on the Fe grade:

Magnetite % = 0.0047 * Fe %² + 1.1142 * Fe % - 2.7834

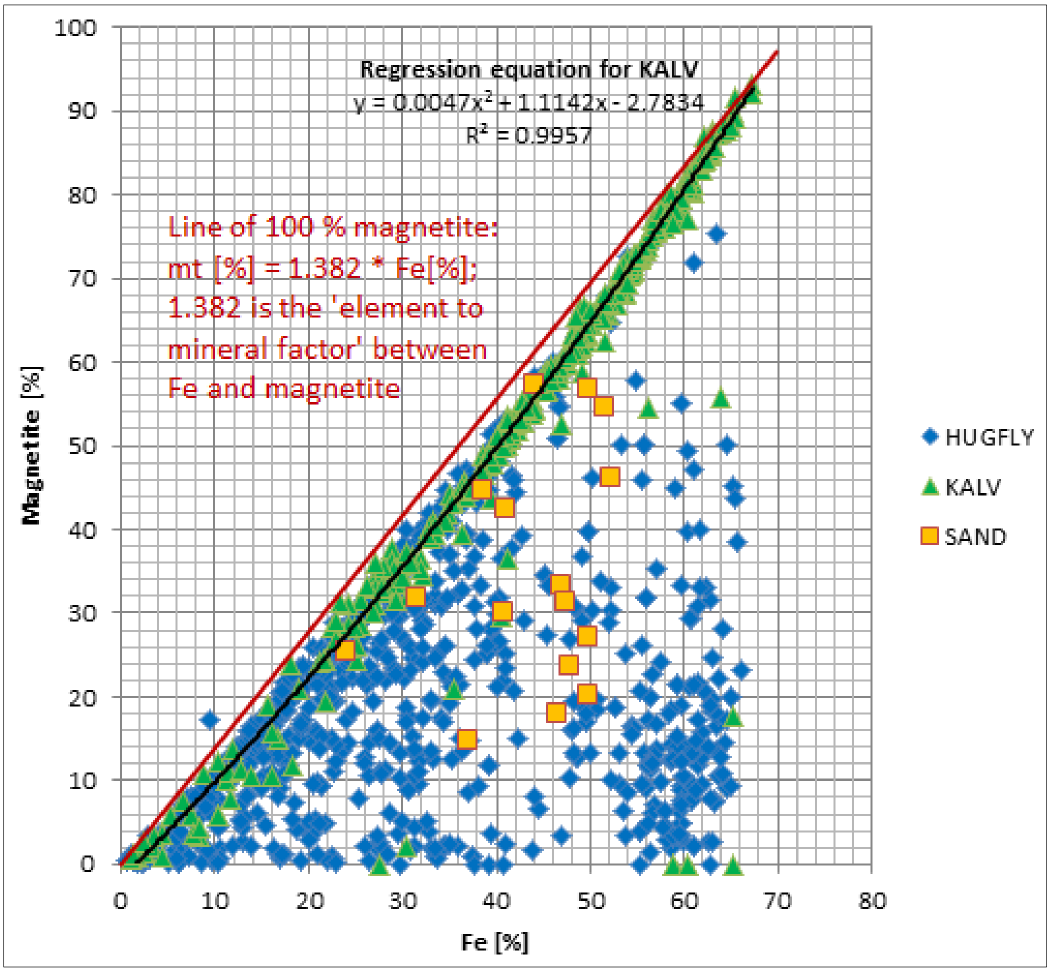


Figure13-4 Correlation Plot of Blötberget Magnetite and Iron

Figure13-4 also demonstrates that the magnetite-hematite ratios of HUGFLY and SAND do not follow a regression line but are varied in their distribution. This effect has been discussed in former reports on the Blötberget field. According to these reports, oxidation of the primary magnetite material has produced large areas of martite mineralisation (haematite formed after replacement of magnetite).

The oxidisation relationship is also shown in the histograms of Fe and magnetite given in Figure 13-6 and Figure 13-7.

The Fe at KALV shows normal distribution since most of the Fe is bound to magnetite. Oxidisation of the HUGFLY and SAND zones causes a bimodal distribution of Fe bound to primary magnetite and Fe bound to martite (as an oxidisation product of magnetite). Consequently in KALV, magnetite is more highly concentrated than in HUGFLY or SAND (Figure 13-6).

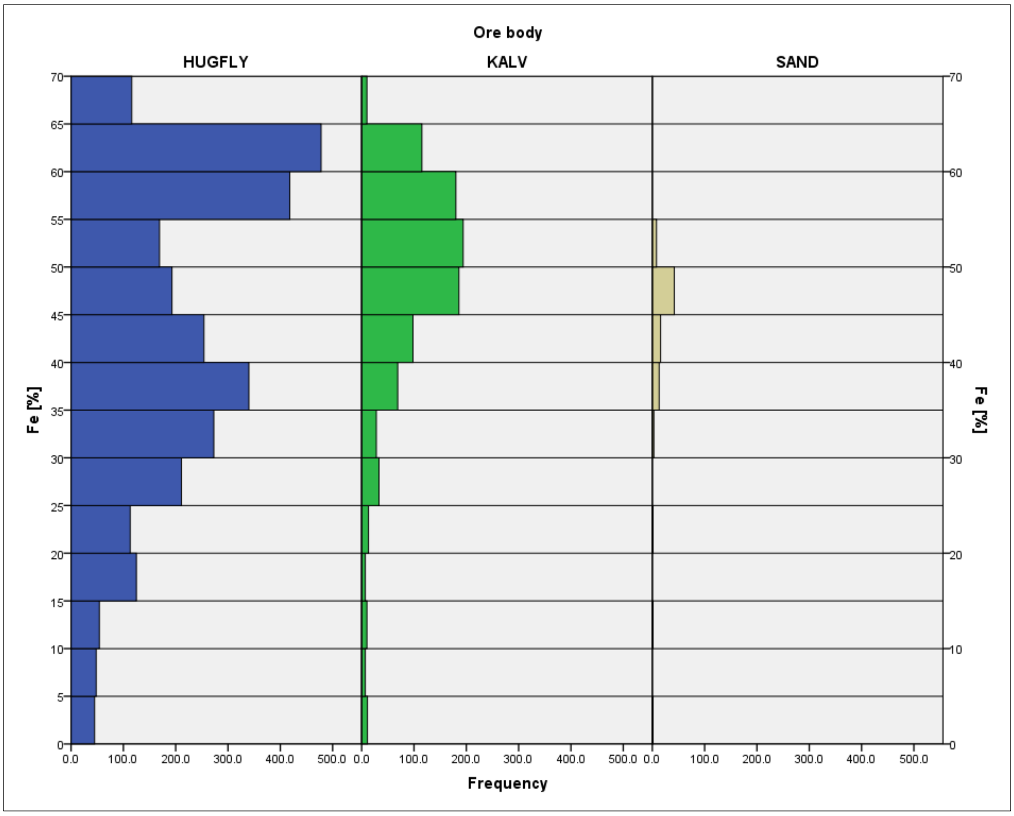


Figure 13-5 **Histogram of Fe for HUGFLY, KALV and SAND**

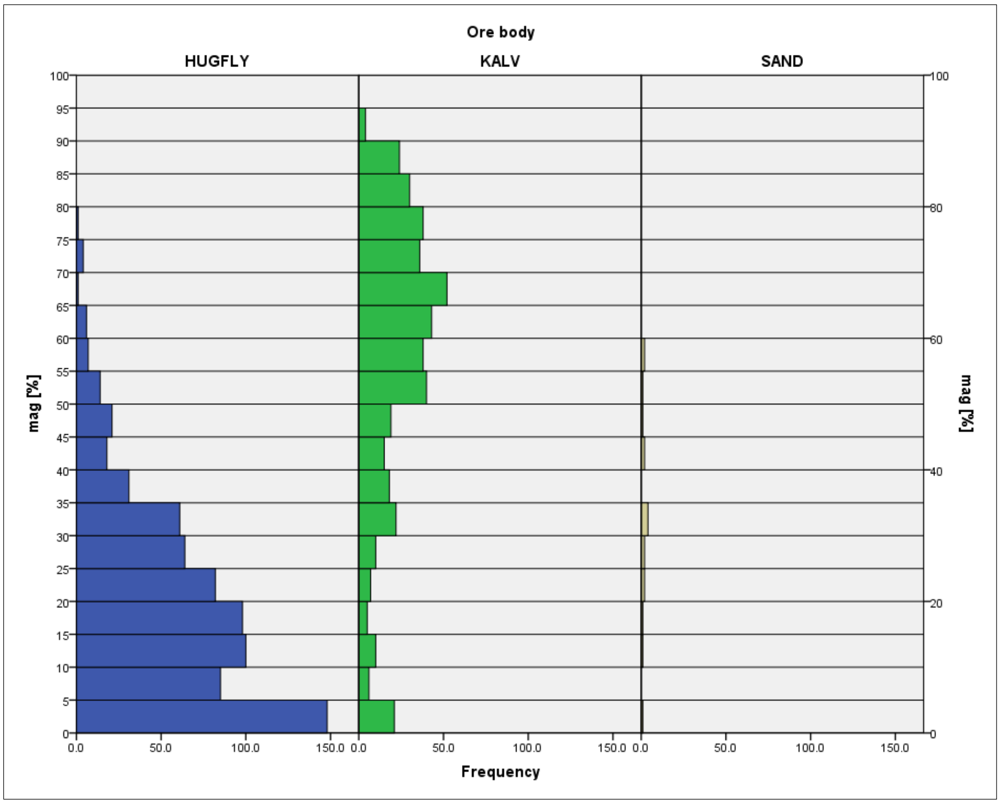


Figure 13-6 **Histogram of magnetite for HUGFLY, KALV and SAND**

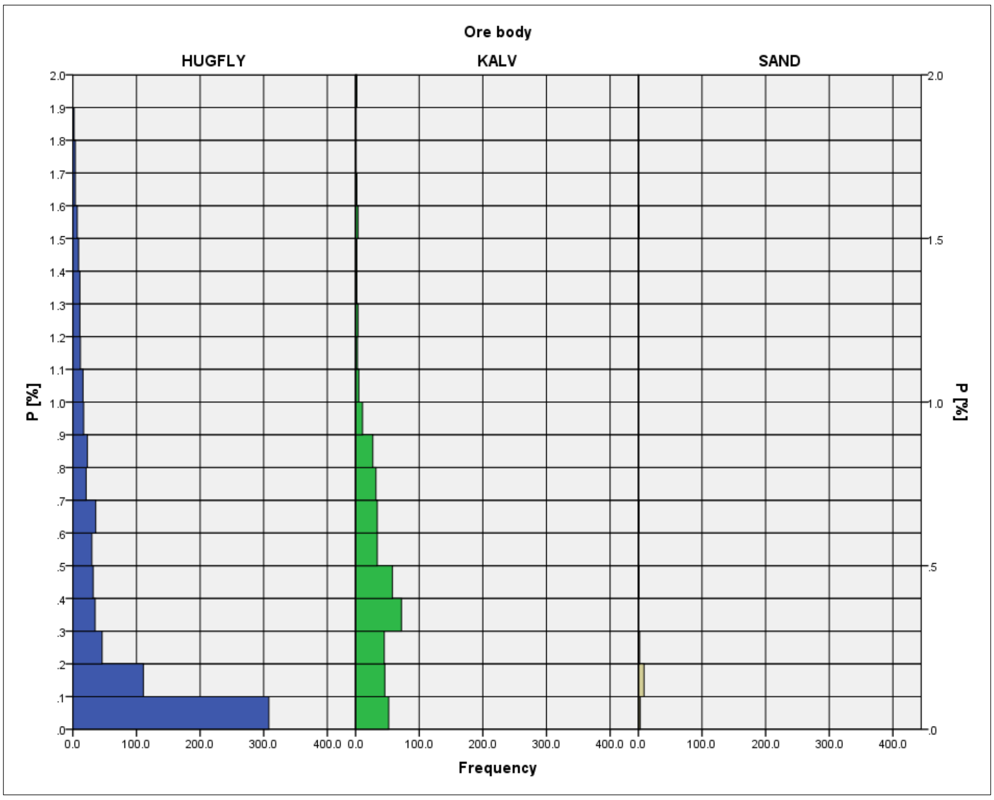


Figure 13-7 **Histogram of P for HUGFLY, KALV and SAND**

13.5.1 Bulk Density

The bulk density determined by the levels of magnetite and hematite and thus with the grade of total Fe bound to magnetite and hematite. Based on this approach, a regression equation has been used which allows a bulk density to be calculated from the Fe grade:

Density = 0.0003 * Fe %² + 0.0157 * Fe % + 2.6605

All blocks of the block model which have a minimum volume portion of wireframe HUGFLY, KALV or SAND of 0.1 % have been attributed with its associated bulk density using the regression equation stated above.

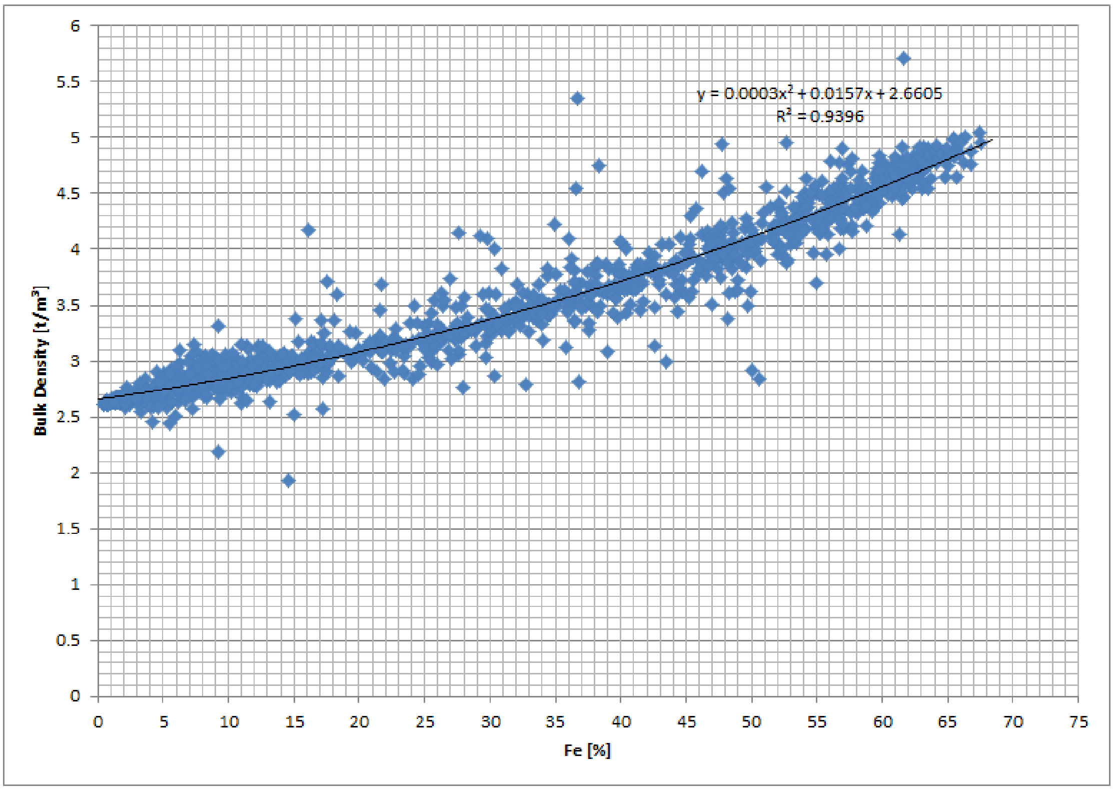


Figure 13-8 **Correlation/regression curve of bulk density vs. assayed Fe grade (1744 samples)**

13.5.2 Grade Capping

Histograms for Fe, magnetite and P have been checked for isolated high grades ("**outliers**"). No outliers which could bias the interpolation have been found, hence an upper grade cut has not been applied to the dataset.

13.5.3 Compositing

Compositing ensures that all assays will have the same influence on geostatistical analyses and interpolation.

A total of 90 % of all samples are less than 4.5 m in length. However, most parts of the model are covered by the samples recovered from the 2012 and 2014 drill programmes which have a shorter sampling interval. A total of 90 % of all these later samples are less than two metres in length. This two metres sample length has been defined as the optimal composite length based on the frequency distribution of lengths of all samples from obtained from the 2012 and 2014 drilling programmes.

In previous estimates no sample grade has been assigned to internal waste or other ‘barren’ country rock intervals. Equally, other intervals which historically have not been sampled or assayed for Fe have been assigned a nominal grade of contained iron. DMT has calculated a grade of 8 % Fe to be applied as the average grade of all other country and waste rock sampled and assayed in the mineralised zones of HUGFLY, KALV and SAND.

Intervals not assayed for phosphorus were not assigned a grade for phosphorus.

After tagging mineralised zones into the database, composite samples were prepared from assays by using a downhole compositing tool in Surpac, and tagged to the three main mineralised zones of HUGFLY, KALV and SAND. Sample lengths have been used by DMT as a weighing factor. Chemical grades of Fe, magnetite and P have been composited. The composite is accepted for the interpolation phase if 50 % of a two metre target length is achieved.

Table 13-2 Percentiles of sample lengths

Percentiles	All samples [3,804]	Historic samples [2,089]	Re-assay and 2012/14 samples [1,715]
10	0.60	0.44	0.80
20	0.95	0.75	1.00
30	1.00	1.12	1.00
40	1.00	1.58	1.00
50	1.30	2.05	1.00
60	1.75	2.60	1.10
70	2.10	3.27	1.31
80	2.90	3.95	1.80
90	4.18	5.58	2.00

13.5.4 Mineralisation Continuity & Variography

Variography studies of the composite data were carried out to support the mineral resource estimation work.

As there is distinction between the magnetite-rich ore in KALV and the hematite-rich ore in HUGFLY/SAND, variography has been carried out for Fe in KALV and in HUGFLY/SAND separately.

The variograms of Fe in KALV and in HUGFLY/SAND indicate that the ranges and the sills are very similar (Table 13-3). This implies a similar spatial distribution of Fe in the magnetite-rich KALV and the hematite-rich HUGFLY and SAND.

For KALV, the magnetite has been calculated based on the regression equation established previously. There appears to be no correlation between Fe and magnetite in HUGFLY and SAND, hence magnetite has be interpolated separately from Fe in HUGFLY and SAND.

Omni-directional variography has been undertaken for all composite data of each ore type (domains) separately with a spatial orientation of -45 degree dip and 145 degree dip direction (strike N55°E). Attempts at directional variography were unsuccessful due to the sparse data within the modelled domains which prevented meaningful interpretation and accurate modelling.

All omni-directional variograms for hematite (Fe) and magnetite (Mag) show reasonable structure, allowing reliable variogram models to be produced (Figure 13-9, Figure 13-10 and Figure 13-11).

For the variography analysis of Phosphorous, all sub-domains have been combined into a single domain on which to undertake the geostatistical study, due to the limited data in some of the domains. The nugget and ranges are relatively easily generated, providing an appropriate level of confidence. All variograms were modelled using a fixed nugget effect of 0 which is appropriate for this continuous, seam-like iron deposit.

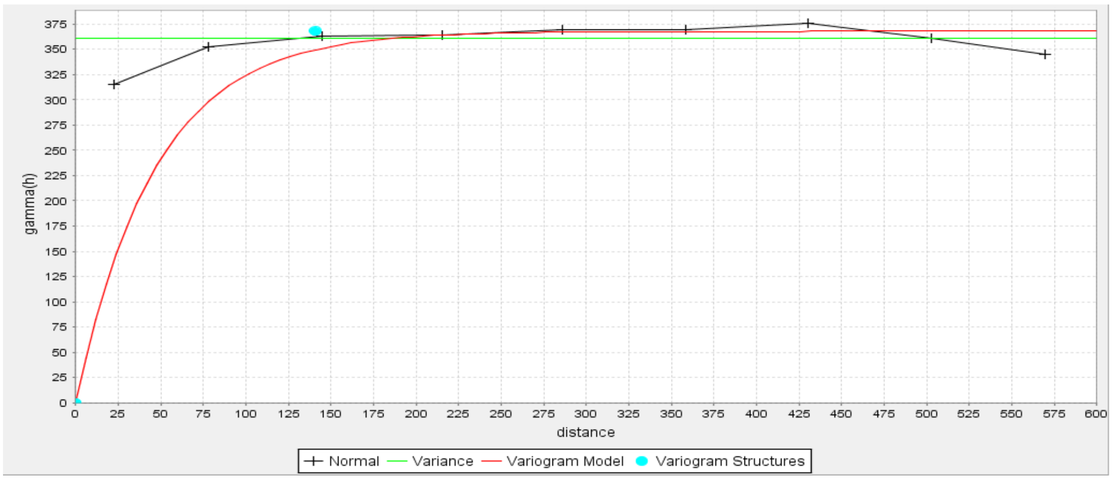


Figure 13-9 Variogram of Fe in HUGFLY/SAND

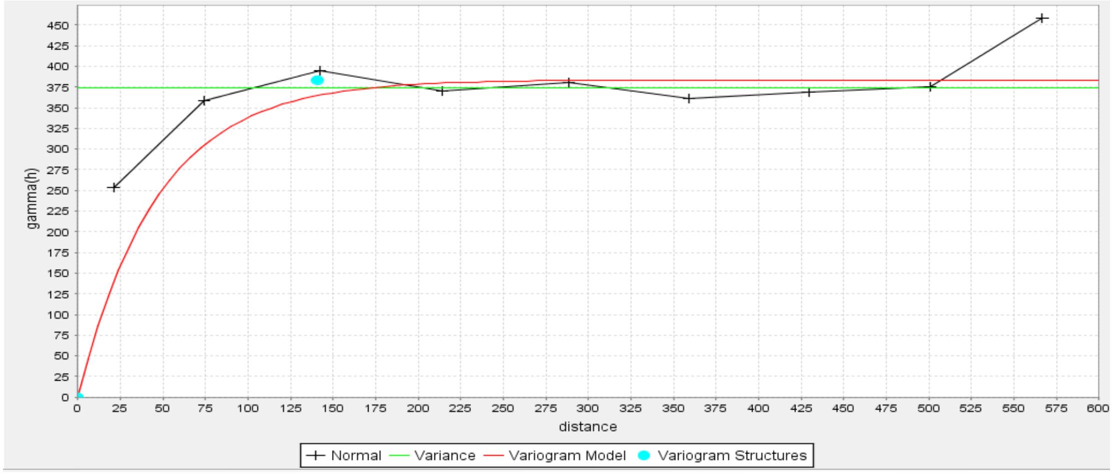


Figure 13-10 Variogram of Hematite in KALV

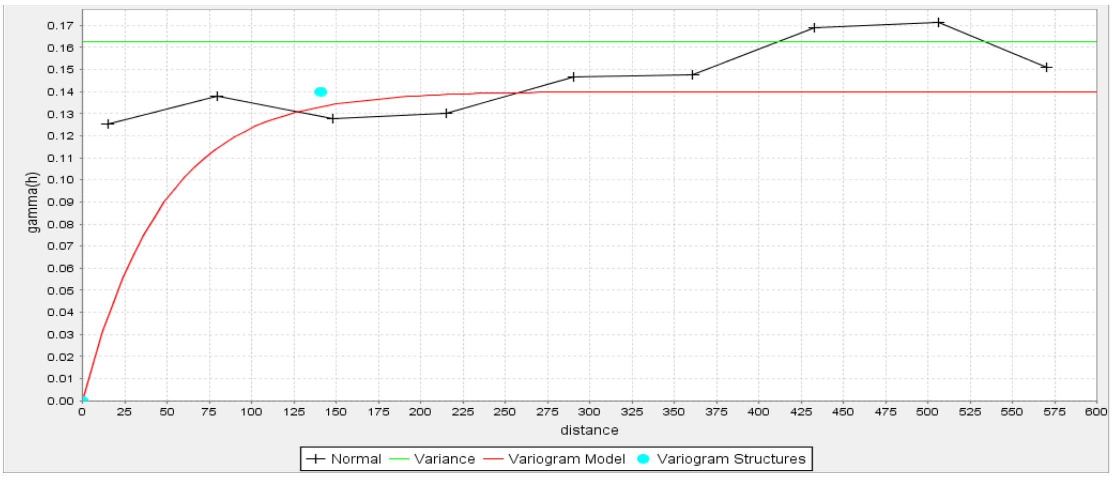


Figure 13-11 Combined domain variogram of P in all zones

Table 13-3 Nugget, sill and range from variograms for Hematite (Fe), Magnetite (Mag) and P

Zone	Nugget	Sill	Range
Fe in HUGFLY and SAND	0	367	140
Fe in KALV	0	380	140
Mag in HUGFLY and SAND	0	175	140
P in HUGFLY, SAND and KALV	0	0.14	140

13.5.5 Interpolation Search Parameters & Grade Interpolation

The variography results allowed for grade estimates for each of the two modelled domains of magnetite-rich ore of KALV and hematite-rich ore of HUGFLY and SAND to be performed using Ordinary Kriging (“**OK**”), applying hard boundaries to the two different estimation domains.

OK has been carried out in three passes for each domain, and the search ellipse parameters for the individual domains are included in Table 13-4.

OK is used to inform the parent cells, with a discretisation of 10 × 5 × 10 in the X, Y, and Z directions respectively. The dip and rotation of the ellipse has been adapted to the overall dip and strike of the domains. The dip direction of major axis has been set to 152°. To honour the curved structure of the domains perpendicular to strike direction, the dip of the major axis has been set to 35° above -400 m and to 50° below -400 m.

The first search uses the 2/3 variogram range, the second search is double this, and the third search is four times the size. These multiple searches ensure all blocks within the modelled mineralised domains are interpolated a grade value. The minimum number of samples was set to three and the maximum number of samples was set to 15 (Table 13-4).

Grade has been estimated into the block model with properties as described in Section 13.8.

Table 13-4 Search parameters for each of the applied passes of OK

Parameter	Max -Major Search Distance m	Max-Major / (Minor) Search Ratio m
1st Pass	100	16 (=6.25)
2nd Pass	200	8 (=25)
3rd Pass	400	4 (=100)

13.6 Resource Classification

The definitions for resource categories used in this report are consistent with the JORC Code 2012.

Under the JORC classification system, a Mineral Resource is defined as:

…"a concentration or occurrence of natural, solid, inorganic or fossilised organic material in or on the Earth's crust in such form and quantity and of such grade or quality that it has reasonable prospects for economic extraction.

"The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

Resources are classified into Measured, Indicated and Inferred categories based upon geological knowledge and confidence (Figure 13-12).

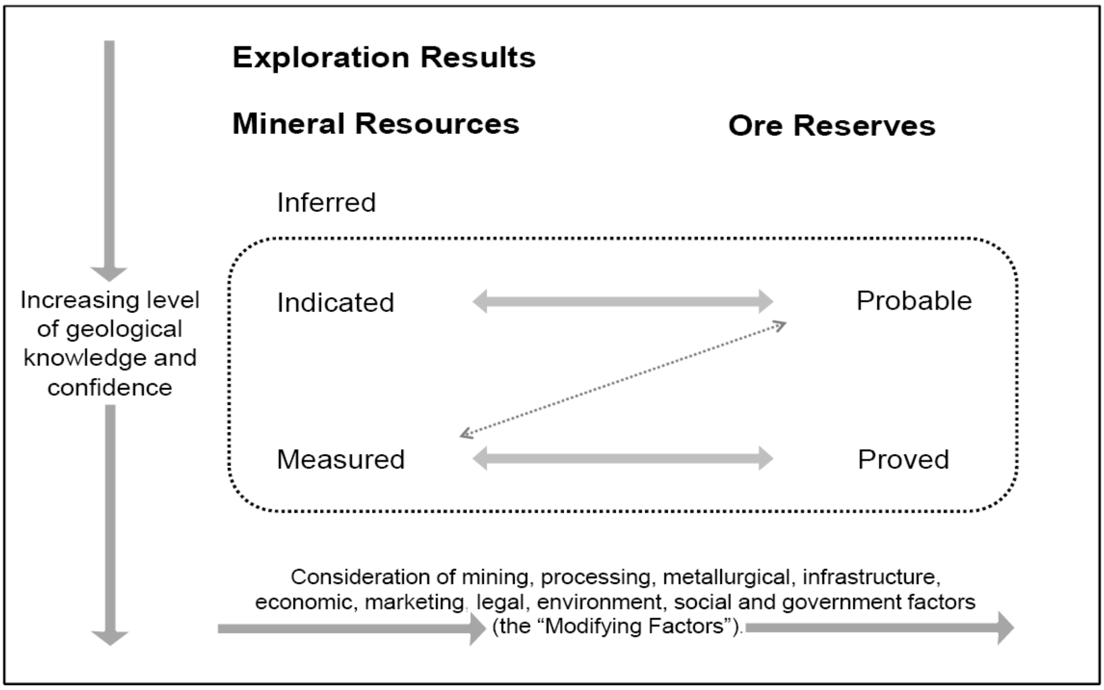


Figure 13-12 Relationship between Exploration Results, Mineral Resources & Ore Reserves

Resource classification within mineralisation envelopes is generally based on drillhole spacing, grade continuity, and overall geological continuity. The distance to the nearest composite and the number of drillholes are also considered in the classification.

In classifying the resource estimate, the following key factors have been considered:

- Confidence in data quantity and specifically sample spacing of Fe and magnetite data;
- Confidence in the geological interpretation and continuity (geological complexity); and
- Confidence in mineralisation / grade continuity (complexity of spatial grade distribution).

Considering the above, the following criteria have been applied for classification into the various mineral resource categories for this estimate:

13.6.1 Measured Resources

- All blocks whose distance to the nearest magnetite sample is less than 2/3 of the variogram range (i.e. <100 m) - excluding distally located drill hole BB_75-001.
- All blocks which are surrounded by measured blocks; and
- All blocks near the historic underground workings.

13.6.2 Indicated Resources

- All blocks whose distance to the nearest magnetite sample is equal or above 90 m and less than the full variogram range of 140 m - including distally located drill hole BB_75-001.

13.6.3 Inferred Resources

- All blocks which are not defined as Measured or Indicated but are included in the interpreted wireframes.

13.7 Preliminary Economic & Mining Assumptions

Initially, DMT did not apply any economic cut-off grades or mining criteria to the global resource estimate which was generated within the confines of the wireframes.

DMT used the wireframes and a set of technical and economic input assumptions, summarised in Table 13-5, to create a preliminary block model, using Geovia Surpac software, in order to constrain the estimated Mineral Resources and to demonstrate reasonable prospects for eventual economic extraction.

Commodity price assumptions are based on typical China import sales, over the past five years, of iron ore fines (62 % Fe) (Figure 13-13).

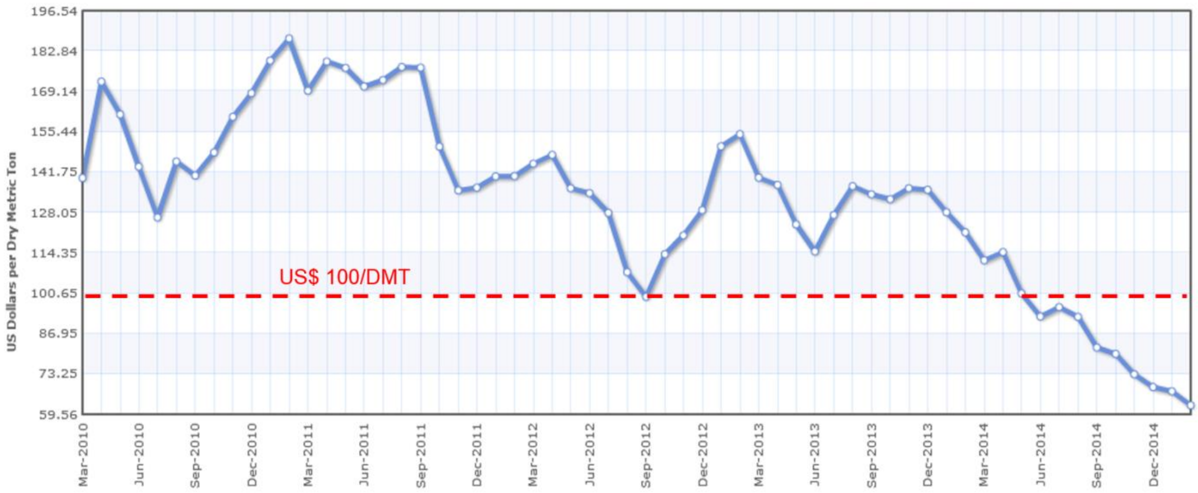


Figure 13-13 Five year historic China import spot price (FR Tianjin port)

Source: Index Mundi

Using preliminary economic input parameters for the proposed mining method, processing and selling-related costs (Table 13-5), the cut-off grade has been estimated by applying the below formula:

$$\frac{\text{Costs for mining plus processing [US\$/t ore]}}{\left(\frac{\text{Price of concentrate [US\$/t conc]}}{\text{Fe grade of concentrate [Fe[\%]/t conc]}}\right)} * \frac{1 + \text{Dilution [fraction]}}{\text{Processing recovery [fraction]}} = Fe\ Cutoff\ grade$$

Table 13-5 Block Model preliminary economic input parameters

Parameter	Cost/Value t/Revenue
Costs for mining plus processing [US$/t ore]	20
Price of concentrate [US$/t conc.]	100
Fe grade of concentrate [Fe[%]/t conc.]	63
Dilution [fraction]	0.1
Processing recovery [fraction]	0.9

The wireframe shells provide a constraint for the reported block model resources based on the JORC definition of Mineral Resources having “reasonable prospects for economic extraction”.

When the basic economic input parameters (Table 13-5) are applied to the wireframes, an indicative COG of 23 % Fe is arrived at.

A tonnage / grade sensitivity study has been carried out by DMT at COGs ranging from 0 % to 60 % Total Fe (Table 13-6). A COG of 25 % has been highlighted as this is the nearest rounded up percentage COG.

Table 13-6 Fe grade cut-off sensitivity results

Fe cut-off %	Volume Mm³	Tonnage Mt	Density t/m³	Fe %	Magnetite %	Hematite %	Magnetite proportion %	Hematite proportion %	Phosphorous %
0	17.2	62.2	3.6	36.2	31.6	19.0	0.62	0.38	0.46
5	17.2	62.2	3.6	36.2	31.6	19.0	0.62	0.38	0.46
10	16.9	61.3	3.6	36.6	31.9	19.3	0.62	0.38	0.46
15	16.2	59.1	3.7	37.5	32.7	19.8	0.62	0.38	0.47
20	14.6	54.3	3.7	39.2	34.2	20.7	0.62	0.38	0.48
25	**12.5**	**47.8**	**3.8**	**41.5**	**36.1**	**22.0**	**0.62**	**0.38**	**0.51**
30	10.5	41.1	3.9	43.8	38.4	22.9	0.63	0.37	0.54
35	8.4	33.8	4.0	46.2	41.3	23.3	0.64	0.36	0.57
40	6.4	26.2	4.1	48.7	44.9	23.1	0.66	0.34	0.60
45	4.1	17.7	4.3	51.6	48.6	23.5	0.67	0.33	0.60
50	2.1	9.2	4.5	55.5	51.8	25.7	0.67	0.33	0.64
55	1.0	4.6	4.6	58.7	54.5	27.6	0.66	0.34	0.70
60	0.3	1.6	4.8	61.5	58.6	27.3	0.68	0.32	0.66

Note: For Measured and Indicated Resources only

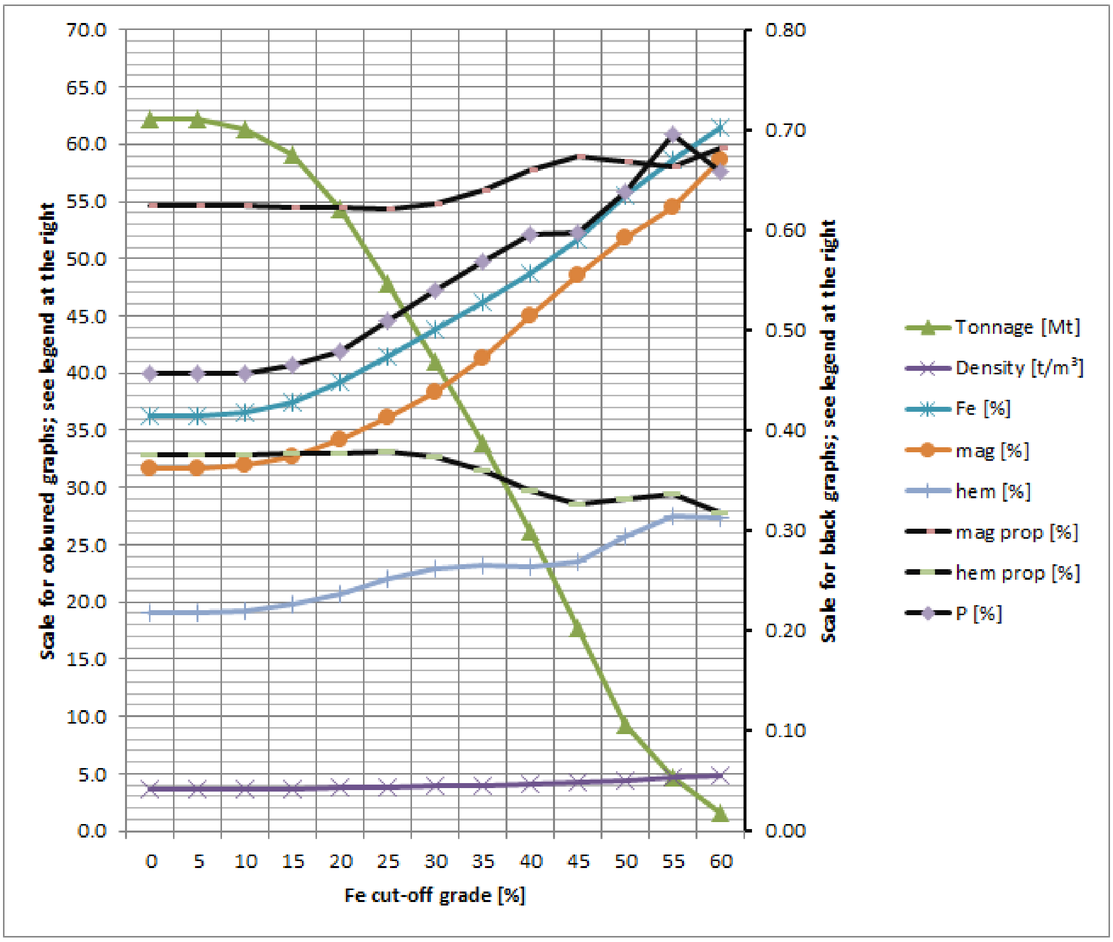


Figure 13-14 Resource Grade-Density-Tonnage Curves

13.8 Block Model

The block model uses regular block size of 10 m length (x), 5 m width (y) and 10 m height (z). These block dimensions are considered to be the most appropriate, considering the geometry of the mineralisation and the proposed mining method. The block model is rotated to the same strike as the mineralisation, N55°E.

The maximum dimensions of the block model are 1,540 m along strike and 1,940 m perpendicular to strike (down-dip); adapted to the drilled area and license area. The total elevation ranges from 200 m to -1,200 m. The total number of blocks is 427,525. No sub-blocking is applied.

Attributes have been added to the block model and populated. Detailed explanations about the attributes are given in Appendix A of this report.

For all blocks lying within or intersected by the HUGFLY and SAND wireframes, a partial percentage attribute has been created named ‘ore_oxidized_perc’. This attribute adds a volume portion ranging from 0.000 to 1.000 (i.e. 0-100 %) to each block lying within or intersected by these wireframes, which has been used for volume correction in the resource estimate (Figure 13-15).

For all blocks lying within or intersected by the KALV wireframe, a partial percentage attribute has been calculated named ‘ore_non_oxidized_perc’. This attribute adds a volume portion ranging from 0.000 to 1.000 to each block lying within or intersected by this wireframe, which has been used for volume correction in the resource estimate.

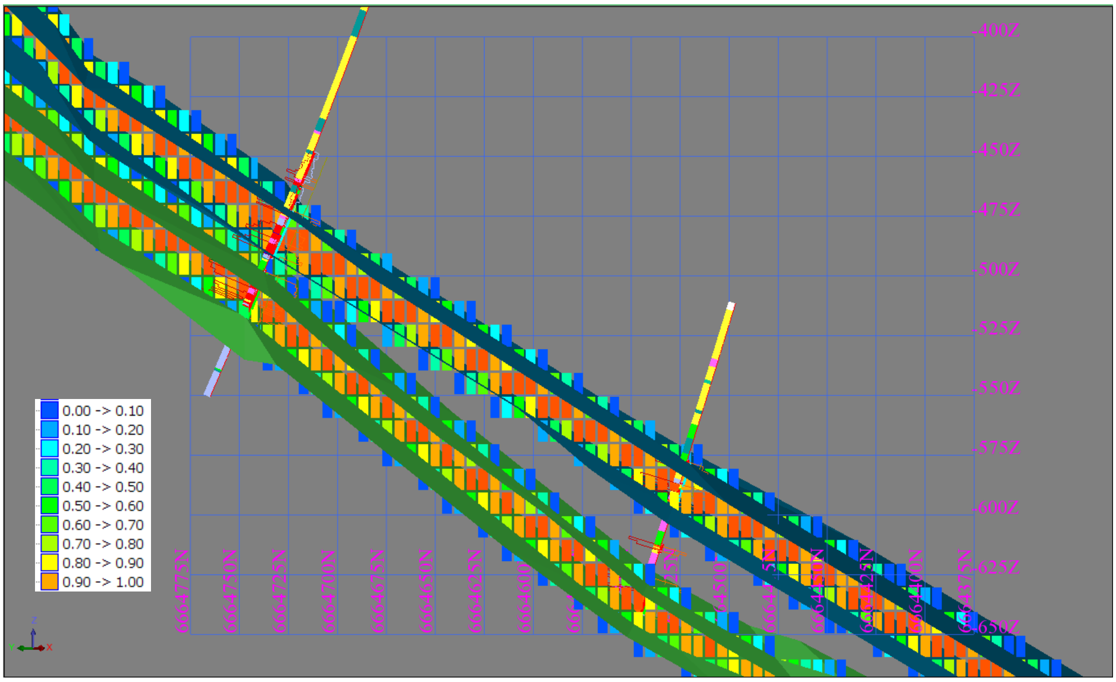


Figure 13-15 Example Section showing attribute ‘ore_total_perc‘ ranging from 0 to 1

A set of geometrical attributes for geology (block ID for each wireframe), topography (topo), license area (conc) or mined out area (hist_mine) have been assigned to the block model.

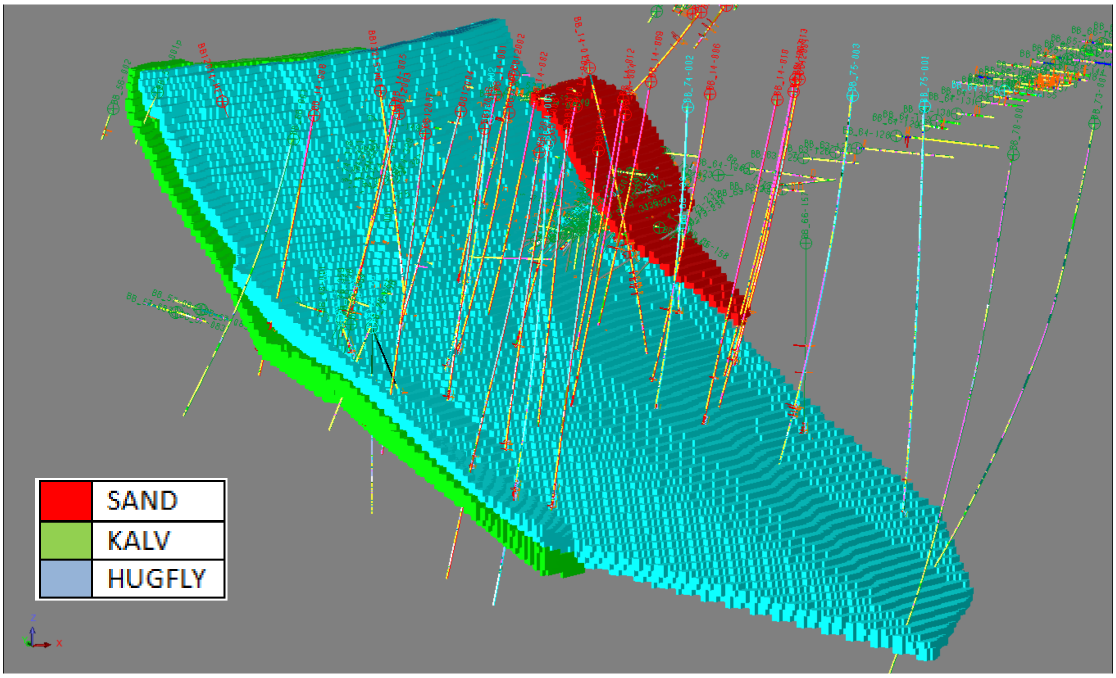


Figure 13-16 3D view to N showing attribute ‘geol’ with integer values

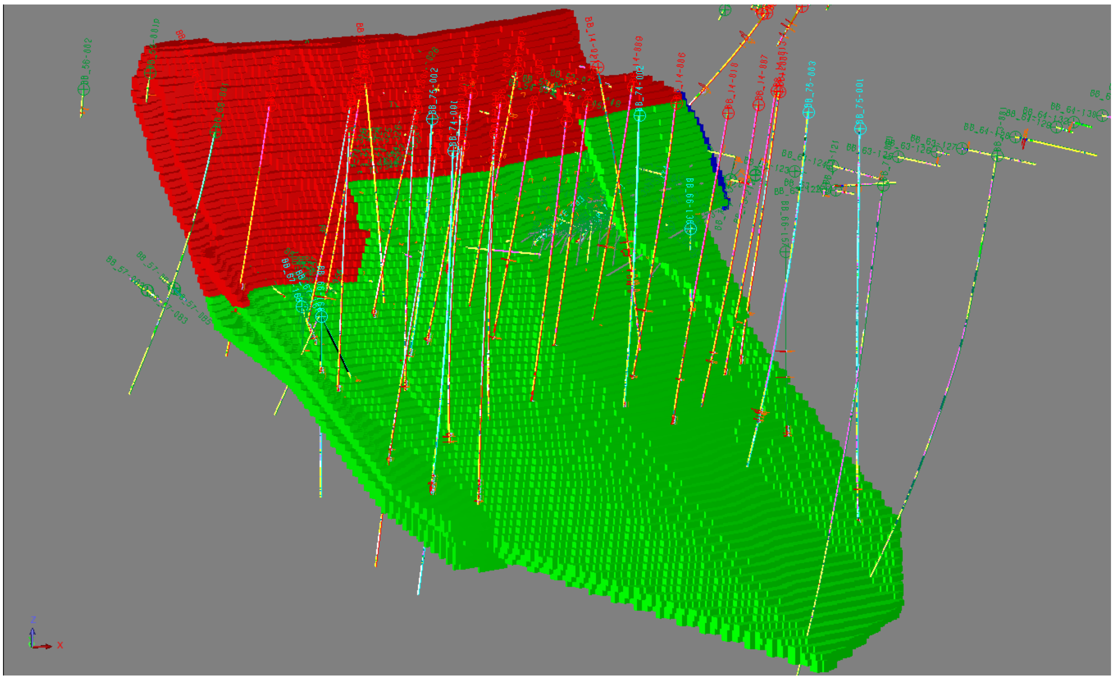


Figure 13-17 3D view to N showing attribute 'hist_mine'

Note: Red = mined

Green = not mined

Grades were interpolated for Total Fe, P and magnetite.

For KALV, the magnetite was not interpolated but assigned to each block with the help of a regression formula based on the interpolated grade of Fe (Figure 13-18, Figure 13-19, Figure 13-20, Figure 13-21 and Figure 13-22).

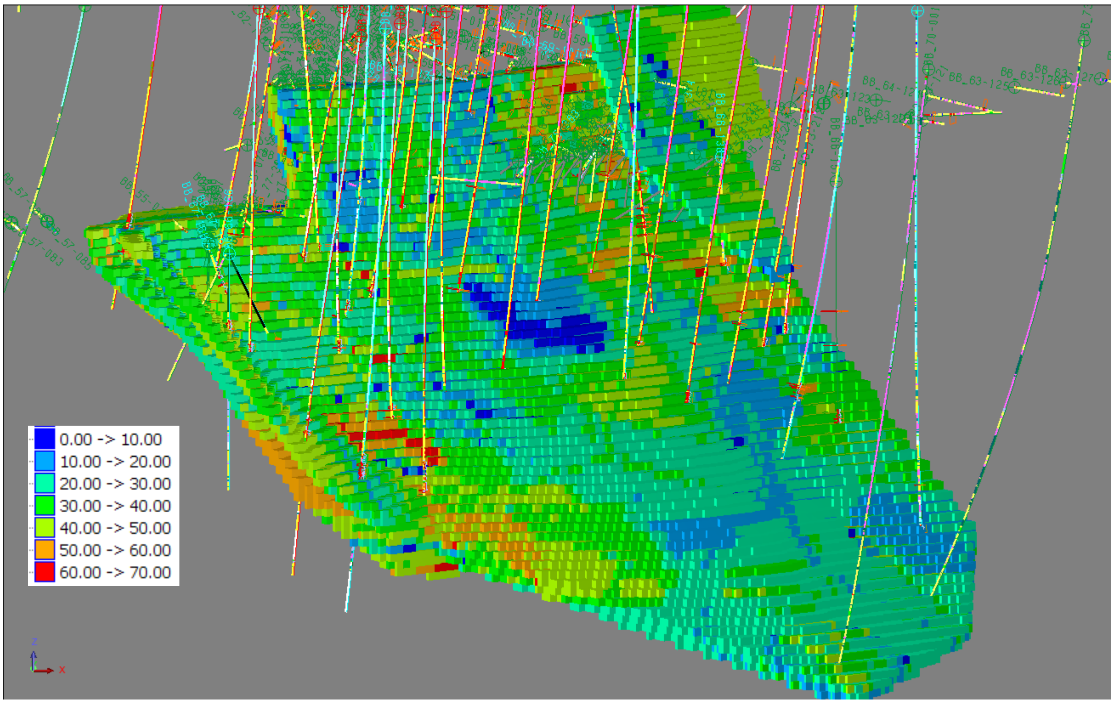


Figure 13-18 3D view to N showing attribute 'fe_perc_tot'

Note: Includes hematite and magnetite-rich ore

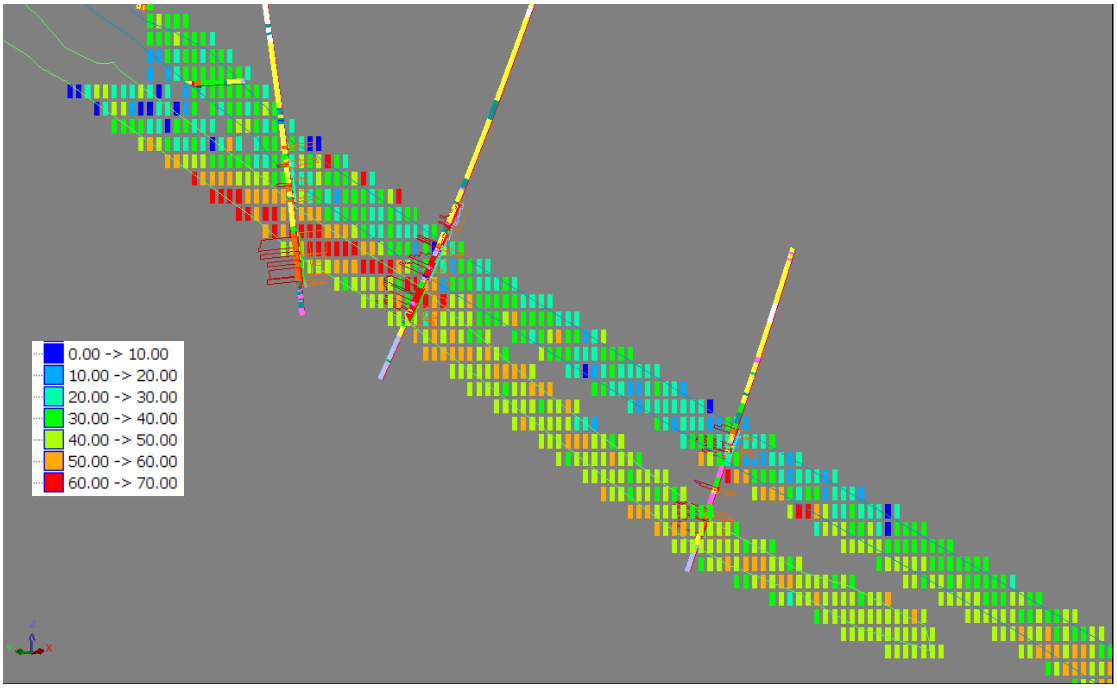


Figure 13-19 Example cross section showing distribution of Fe [%] (attrib.: Fe_perc_tot)

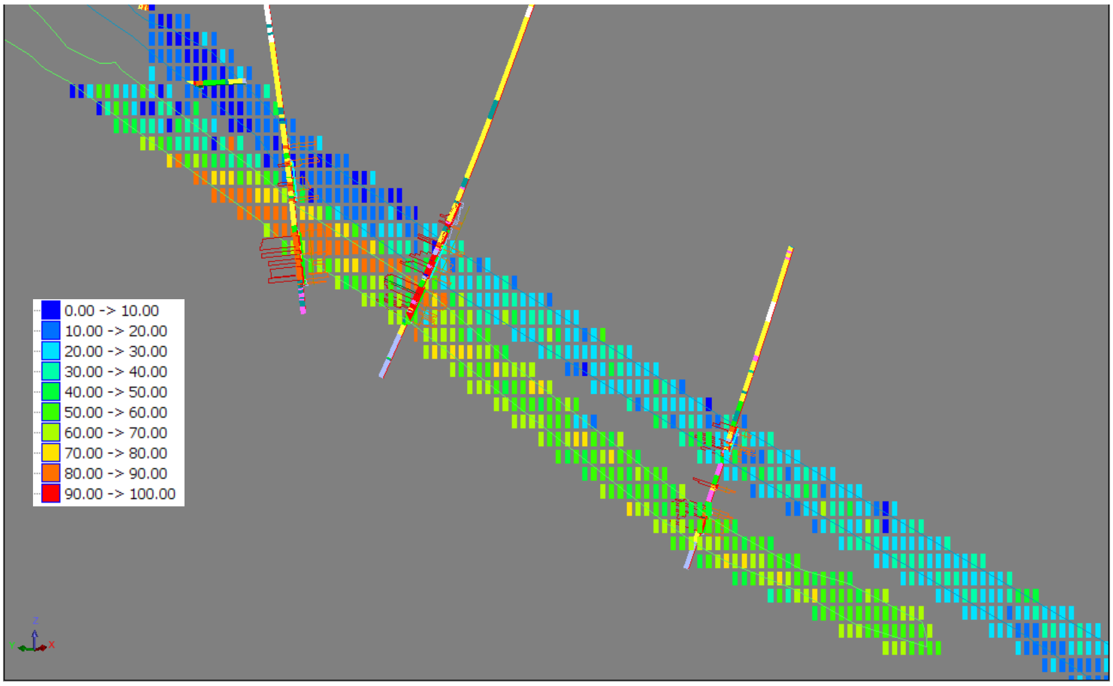


Figure 13-20 Example cross section showing magnetite distribution [%] (attrib.: 'mt_perc_tot')

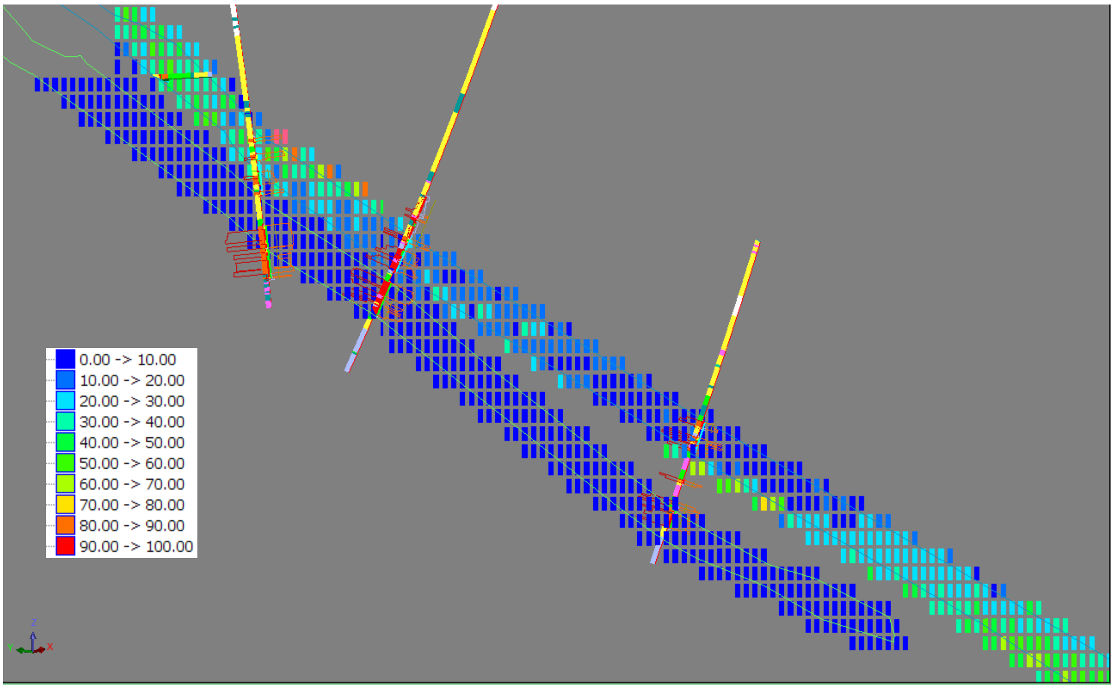


Figure 13-21 Example cross section showing distribution of hem [%] (attrib.: hem_perc_tot)

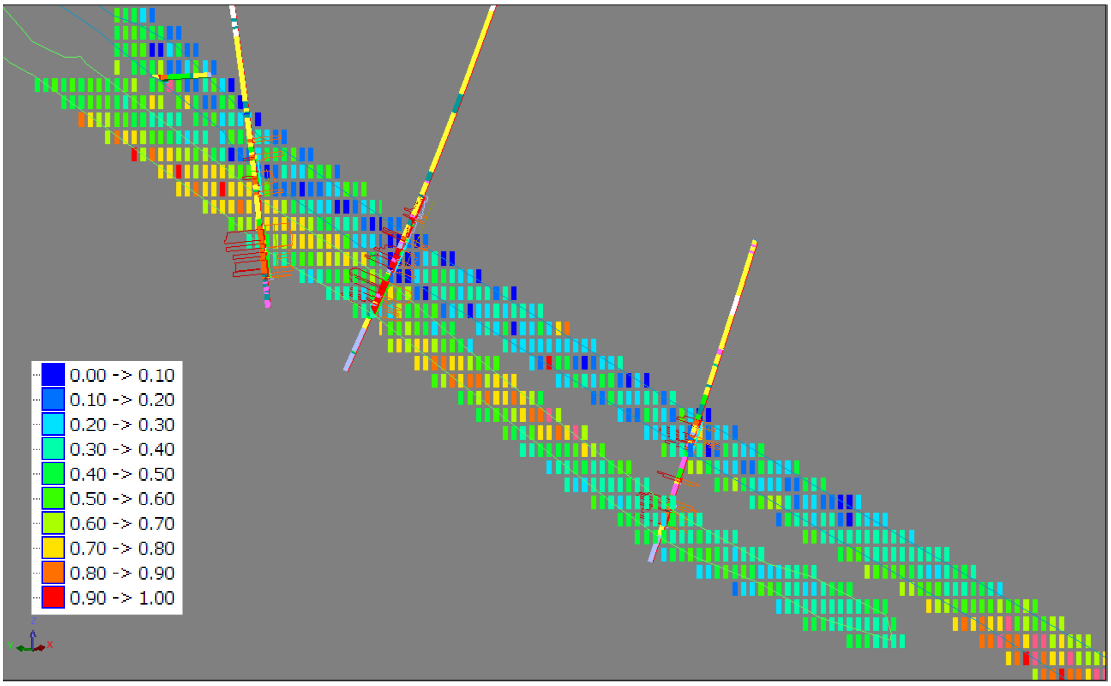


Figure 13-22 Example cross section showing distribution of P [%] (attrib.: p_perc_tot)

The bulk density was assigned to each block with the help of a regression formula based on the interpolated grade of Fe.

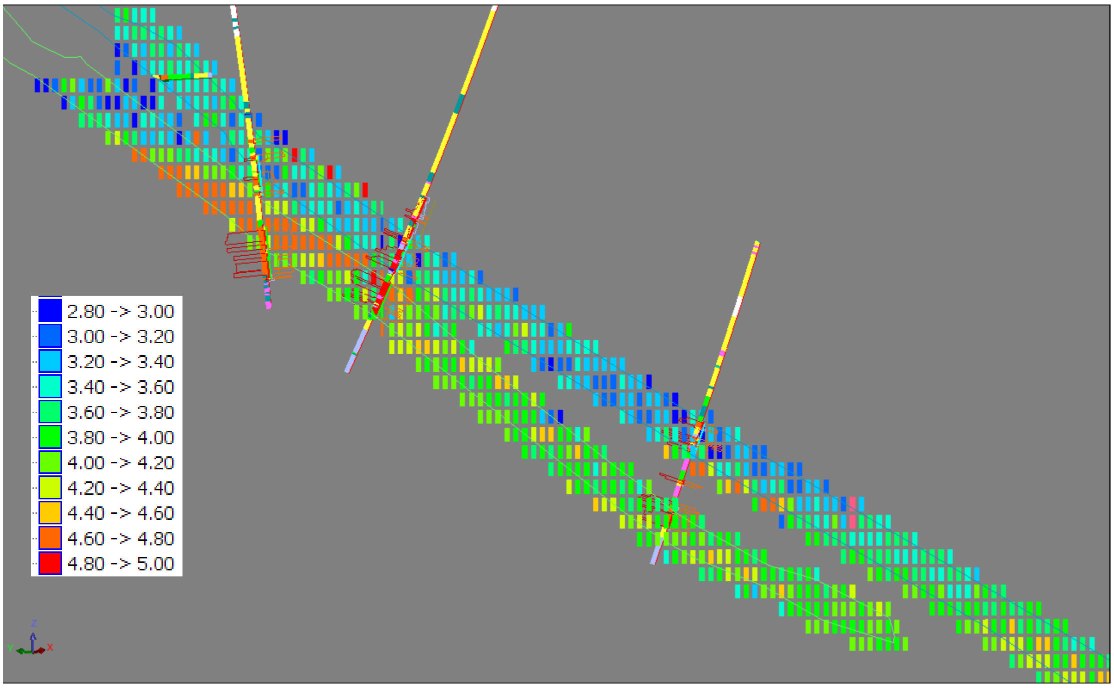


Figure 13-23 Example cross section showing ore density distribution [%] (attrib.: dens_ore)

Table 13-7 Desciptions of block model attributes

Priority	Process	Attribute Name	Type	Decimals	Background	Description
1	inside concession string; keep partial	conc	Integer	-	-99	Concession area; 0: outside concession, 1: inside
2	partial percentage inside concession string	conc_perc	Real	2	-99	Volumetric portion of concession area; 0 to 1: 0 % inside concession area to 100
19	calculation; =0.0003*(fe_perc_tot^2)+(0.0157*fe_perc_tot)+2.6605; ore_total_perc>0	dens_ore	Real	2	-99	Bulk density of ore; t per m³
20	calculation; =2.8; topo=1, waste_perc>0	dens_waste	Real	2	-99	Bulk density of waste; t per m³
16	interpolation of magnetite; distance to nearest magnetite sample; ore_total_perc>0	dist	Real	2	-99	Distance to nearest drill hole
11	interpolation; Fe diluted with non-mineralized intervals as 8%Fe intervals; ore_oxidized_perc>0	fe_perc_ox	Real	2	-99	Grade of Fe in oxidized zone; %
12	interpolation; Fe diluted with non-mineralized intervals as 8%Fe intervals; ore_non_oxidized_perc>0	fe_perc_nonox	Real	2	-99	Grade of Fe in non-oxidized zone; %
18	calculation; ((fe_perc_ox*ore_oxidized_perc)+(fe_perc_nonox*ore_non_oxidized_perc))/(ore_oxidized_perc+ore_non_oxidized_perc); ore_total_perc>0	fe_perc_tot	Real	2	-99	Grade of total Fe of oxidized and non-oxidized zone; %
	interpolation; Fe diluted with non-mineralized intervals as 8%Fe intervals; waste_perc>0	fe_perc_waste	Real	2	-99	Grade of Fe in waste; %
5	inside wireframes from 1 to 3; topo = 1 keep partial	geol	Integer	-	-99	Geology (mineralization and non-mineralization); 0: 1: mineralized body HUGFLY, 2:
23	calculation; =(fe_perc_ox-(mt_perc_ox*0.7236))*(1/0.6994); ore_oxidized_perc>0	hem_perc_ox	Real	2	-99	Grade of hematite in oxidized
24	calculation; =(fe_perc_nonox-(mt_perc_nonox*0.7236))*(1/0.6994); ore_non_oxidized_perc>0	hem_perc_nonox	Real	2	-99	Grade of hematite in non-oxidized zone; %
25	calculation; ((hem_perc_ox*ore_oxidized_perc)+(hem_perc_nonox*ore_non_oxidized_perc))/(ore_oxidized_perc+ore_non_oxidized_perc); ore_total_perc>0	hem_perc_tot	Real	2	-99	Grade of total hematite of oxidized and non-oxidized zone; %
26	calculation; =hem_perc_tot/(mt_perc_tot+hem_perc_tot); ore_total_perc>0	hem_prop_tot	Real	2	-99	Proportion of total grade of hematite in grade of
	empty attribute; placeholder for resource estimate table	hem_perc_waste	Real	2	-99	
	empty attribute; placeholder for resource estimate table	hem_prop_waste	Real	2	-99	
8	partial percentage of wireframe of HUGFLY and SAND	ore_oxidized_perc	Real	3	-99	Volumetric portion of oxidized ore in block; 0 to 1:
7	partial percentage of wireframe of KALV	ore_non_oxidized_perc	Real	3	-99	Volumetric portion of non-oxidized ore in block; 0 to 1:
9	calculation; =ore_oxidized_perc+ore_non_oxidized_perc; (ore_oxidized_perc>0 or ore_non_oxidized_perc>0), topo=1	ore_total_perc	Real	3	-99	Volumetric portion of total non-oxidized ore and oxidized ore in block; 0 to 1: 0%
10	calculation; =1-ore_total_perc; topo=1; inside outside_cut30.dtm	waste_perc	Real	3	-99	Volumetric portion of waste; 0 to 1: 0% waste to 100% waste
13	interpolation; mt as it is, not diluted or mt diluted with non-mineralized intervals as 6%mt intervals; ore_oxidized_perc>0	mt_perc_ox	Real	2	-99	Grade of magnetite in oxidized zone; %
21	calculation; =0.0047*(fe_perc_nonox^2)+1.1142*fe_perc_nonox-2.7834; ore_non_oxidized_perc>0, fe_perc_nonox>=0	mt_perc_nonox	Real	2	-99	Grade of magnetite in non-oxidized zone; %
22	calculation; ((mt_perc_ox*ore_oxidized_perc)+(mt_perc_nonox*ore_non_oxidized_perc))/(ore_oxidized_perc+ore_non_oxidized_perc); ore_total_perc>0	mt_perc_tot	Real	2	-99	Grade of total magnetite of oxidized and non-oxidized zone; %
27	calculation; =mt_perc_tot/(mt_perc_tot+hem_perc_tot); ore_total_perc>0	mt_prop_tot	Real	2	-99	Proportion of total grade of magnetite in grade of
	empty attribute; placeholder for resource estimate table	mt_perc_waste	Real	2	-99	
	empty attribute; placeholder for resource estimate table	mt_prop_waste	Real	2	-99	
14	interpolation; P as it is, not diluted; ore_oxidized_perc>0	p_perc_ox	Real	2	-99	Grade of P in oxidized zone; %
15	interpolation; P as it is, not diluted; ore_non_oxidized_perc>0	p_perc_nonox	Real	2	-99	Grade of P in non-oxidized
22	calculation; ((p_perc_ox*ore_oxidized_perc)+(p_perc_nonox*ore_non_oxidized_perc))/(ore_oxidized_perc+ore_non_oxidized_perc); ore_total_perc>0	p_perc_tot	Real	2	-99	Grade of total P of oxidized and non-oxidized zone; %
	empty attribute; placeholder for resource estimate table	p_perc_waste	Real	2	-99	
17	calculation; 1: dist=> 0 and < 90; 2: dist=> 90 and <140; 3: dist>= 140	rclass	Integer	-	-99	Resource class; 0: not classified as resource 1: measured resource, 2:
3	below topography surface; keep partial	topo	Integer	-	-99	Topography; 0: air, 1: rock
4	partial percentage below topography surface	topo_perc	Real	2	-99	Volumetric portion of topography; 0 to 1: 0 % rock to
6	mined out; 0: conc=1, geol>0; 1: hist_mine=0, below cutting plane, not keep partial	hist_mine	Integer	-	-99	Historical mining activities; 0: ore mined out, 1:ore not

13.9 Model Validation

In order to check that the grade interpolation has worked appropriately, the interpolated block model has been validated against the corresponding domained composites using the following techniques:

- Comparison of wireframe volumes with the block model volume;
- Visual inspection of block grades in plan and section and comparison with drill hole grades; and
- Statistical comparison of global block grades and composite grades within mineralised domains (mean and frequency plots).

Table 13-8 Comaprison of wireframe to block model volumes

Domain / Zone	Wireframe Volume Mm^3	Block Model Volume Mm^3
HUGFLY	14.6	14.6
KALV	5.3	5.3
SAND	0.47	0.47

A statistical comparison of global block grades and composite grades within mineralised domains has been carried out using mean and frequency plots. These have demonstrated a close correlation (Figure 13-24).

The comparison illustrates that no obvious bias has been introduced during the block modelling process. The model blocks are slightly higher grade than their corresponding composites in the low-grade range. The opposite is true in the high-grade range; however, as shown in Figure 13-24 the two data sets are closely aligned. The frequency distribution of interpolated block data follows the distribution of composite data and shows the typical smoothing (upgrading of the lower grade mineralisation and downgrading of the higher grade mineralisation).

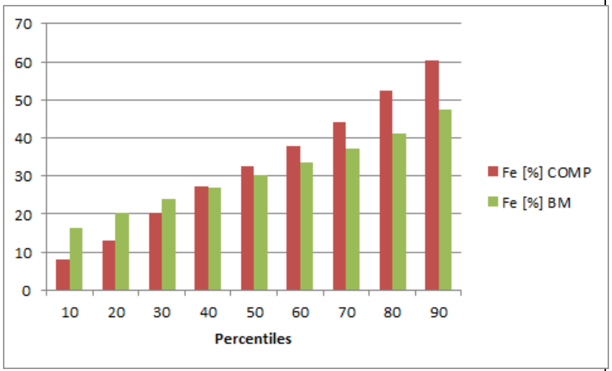
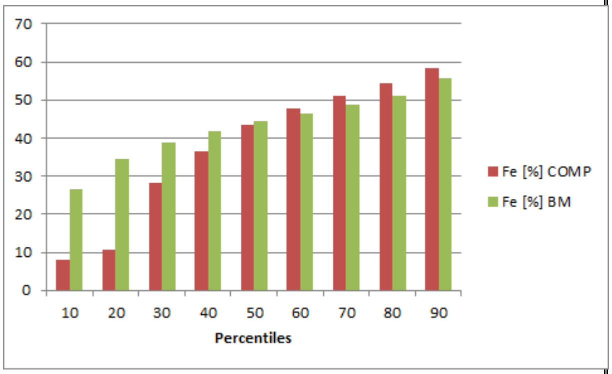
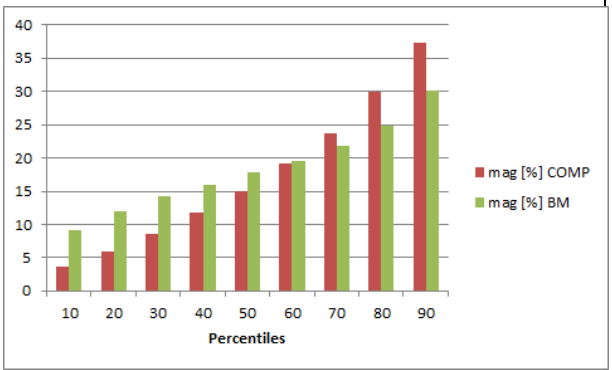
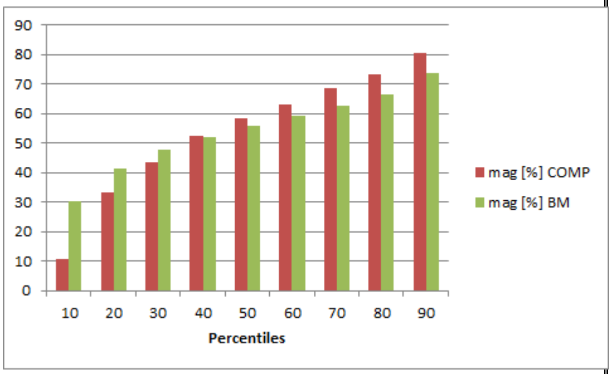
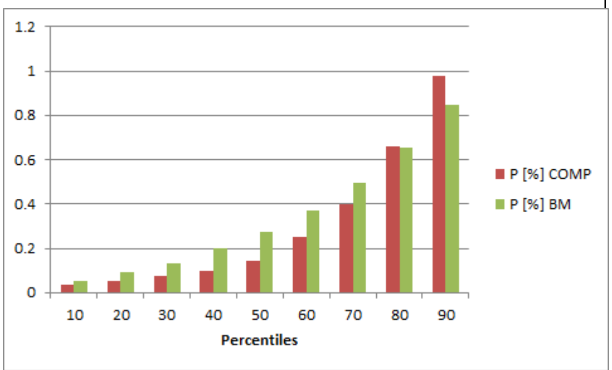
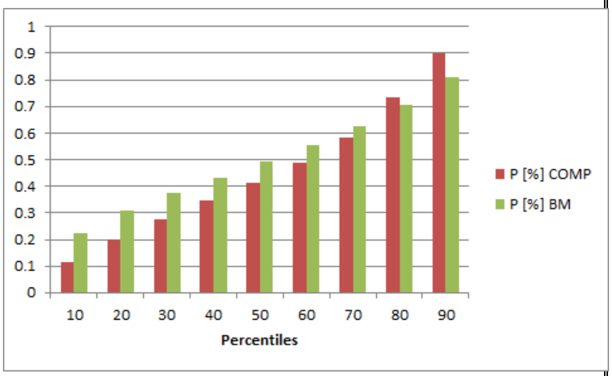
<table>
<tr><th>HUGFLY-SAND (hematite-rich ore)</th><th>KALV (magnetite-rich ore)</th></tr>
<tr><td>
</td><td>
</td></tr>
<tr><td>Mean (Composite) = 33.2
Mean (Block Model) = 31.1</td><td>Mean (Composite) = 37.5
Mean (Block Model) = 42.7</td></tr>
<tr><td>
</td><td>
</td></tr>
<tr><td>Mean (Composite) = 18.2
Mean (Block Model) = 19.0</td><td>Mean (Composite) = 53.1
Mean (Block Model) = 53.9</td></tr>
<tr><td>
</td><td>
</td></tr>
<tr><td>Mean (Composite) = 0.36
Mean (Block Model) = 0.39</td><td>Mean (Composite) = 0.48
Mean (Block Model) = 0.52</td></tr>
</table>

Figure 13-24 Frequency distribution of data for composites and the block model

On the basis of its review and validation procedures, DMT is of the opinion that the block model is valid and acceptable for estimating Mineral Resources.

The reader should note that a Mineral Resource **is not** an Ore Reserve as it has not been demonstrated to be economically mineable.

According to the JORC Code, Measured and Indicated Resources can be converted to Proved and Probable Ore Reserves when consideration of mining, processing, metallurgical, infrastructure, economic, marketing, legal, environment, social and government factors; ("**the Modifying Factors**") has been carried out (Figure 13-12).

The figure below illustrates the spatial distribution of Measured, Indicated and Inferred Mineral Resources within the Blötberget deposit.

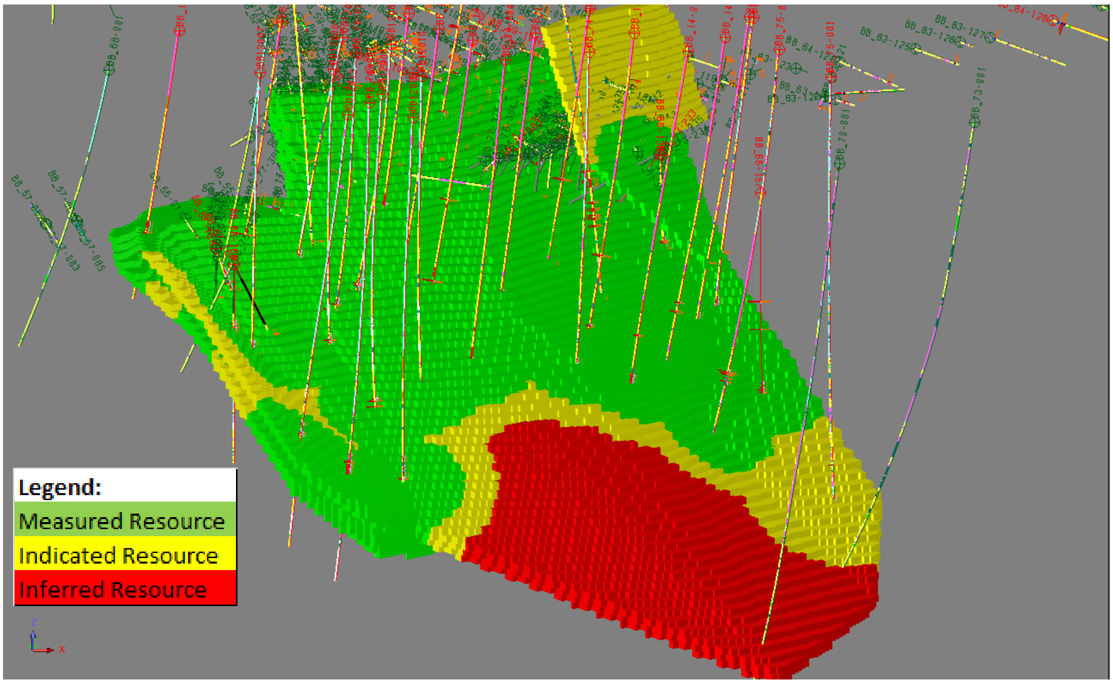


Figure 13-25 Blötberget 2015 Resource Block Model

13.10 Previous JORC Compliant Mineral Resource Estimates

Previous JORC Compliant MRE's have been undertaken by GeoVista, Sweden. GeoVista's initial MRE estimate was carried out in 2011, with subsequent updates in 2012, and 2014.

The GeoVista January 2014 MRE established a COG based on similar preliminary economic assumptions to those made in this DMT resource estimate. However, the parameters were not applied to the 2014 resource statement.

Table 13-9 therefore compares the resource estimates of 2014 and 2015 without a COG applied (i.e. 0 % Fe Total).

The January 2014 MRE did not allow for the loss of volume and tonnage created by the former mined out areas in the upper levels of the mine. Subsequently, and due to improved information, the 2015 estimate has excluded some areas that have been mined out or are believed to be "un-mineable" areas.

Since the 2014 MRE, additional resources have been added in the 'Wedge - Betsta' areas a result of the 2014 drilling programme. In 2014, no Fe grade (i.e. 0 %) was applied to the internal waste or country rock material, whereas the 2015 estimate has applied an 8 % average Fe grade to these parts of the block model.

The additional material recently explored from the Wedge area connects the two bodies HUG and FLY, to create HUGFLY. The upgrading of the January 2014 Inferred Resource for HUG and FLY (low Fe grade) to Measured and Indicated Resources leads to slightly lower Fe grades for Measured and Indicated Resources of HUGFLY but the overall tonnage has approximately doubled.

The DMT 2015 MRE shows a slightly higher tonnage for SAND but a lower Fe grade.

Estimates for density, magnetite, hematite and mag-hem ratios were not available in the 2014 MRE.

Table 13-9 Comparison of GeoVista 2014 estimate and DMT estimate March 2015

Resource Category	Resource Estimate 2014			Resource Estimate 2015		
	Tonnage Mt	Fe %	P %	Tonnage Mt	Fe %	P %
Measured	10.7	34.3	0.3	53.7	37.0	0.46
Indicated	27.4	44.8	0.5	8.5	31.1	0.43
Measured + Indicated	**38.1**	**41.9**	**0.4**	**62.2**	**36.2**	**0.46**
Inferred	21.7	33	0.4	10.5	27.3	0.48
Total	**59.8**	**38.6**	**0.4**	**72.7**	**34.9**	**0.46**

Note: These resources are **global** estimates with no cut-off parameters applied and are for comparison purposes only

13.11 Estimate of Mineral Resources

DMT has prepared a Mineral Resource estimate for the Blötberget Project with a drillhole database cut-off date of 1st January, 2015.

The Mineral Resource estimate has an effective date of 30th January, 2015 and has an issue date of 10th April 2015.

DMT applied basic mining and economic parameters (Table 13-5), including commodity price and wireframe assumptions, to estimate a cut-off grade for resource estimation of 25 % Fe (Total).

The total Measured and Indicated Resource estimated for the Blötberget Project, at a preliminary economic cut-off Grade of 25 % Fe, is 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P.

1) HUGFLY contains an estimated 26.5 Mt of Measured and Indicated Resources at a grade of 38.5% Fe (Total) and 0.5% P.
2) KALV contains an estimated 19.8 Mt of Measured and Indicated Resources at a grade of 45.6 % Fe (Total) and 0.5 % P.

3) SAND contains an estimated 1.4 Mt of Measured and Indicated Resources at a grade of 38.5% Fe (Total) and 0.5% P.

Of the total estimated contained Fe, the magnetite proportion is estimated at 62% and the hematite at 38%.

DMT has reported all the material of magnetite-rich ore of KALV and hem-rich ore of HUGFLY and SAND contained within the resource block model limited by the licence area and excluding the material mined out by historical mining activities.

DMT considers all of the material reported as Measured and Indicated Resources to have ‘reasonable prospect of economic extraction’ given appropriate economic and technical considerations.

Table 13-10 and Table 13-11 summarise the Mineral Resource estimate for the Blötberget Project as of 30th January, 2015. . The Block Model has been constrained using basic economic and mining parameters and the Mineral Resources are estimated at a COG of 25%.

Table 13-10 Measured and Indicated Resources for the Blötberget Iron Project - January 2015

Fe Cut-off % Fe	Resource Category	Volume Mm³	Tonnage Mt	Density t/m³	Fe %	Magnetite %	Hematite %	Magnetite proportion %	Hematite proportion %	Phos. %
25	Measured	11.1	42.5	3.8	41.9	36.8	21.9	0.63	0.37	0.51
	Indicated	1.4	5.3	3.7	38.2	30.5	23.2	0.57	0.43	0.5
	Measured + Indicated	**12.5**	**47.8**	**3.8**	**41.5**	**36.1**	**22.0**	**0.62**	**0.38**	**0.51**
	Inferred	1.5	5.4	3.5	33.5	23.5	23.5	0.50	0.50	0.52

Notes:

1) JORC 2012 definitions were followed for estimating Mineral Resources;
2) Mineral Resources are estimated at a cut-off grade of 25 % Fe;
3) Mineral Resources are estimated using a five year historical average price of US$ 100 per tonne (Source: IndexMundi); and
4) Figures may not total due to rounding errors.

Table 13-11 Deposit specific Resources for the Blötberget Iron Project - January 2015

Fe Cut-off % Fe	Deposit	Volume Mm³	Tonnage Mt	Density t/m³	Fe %	Magnetite %	Hematite %	Magnetite proportion %	Hematite proportion %	Phos. %
25	HUGFLY	7.2	26.5	3.7	38.5	20.4	34.0	0.37	0.63	0.5
	KALV	5.0	19.8	4.0	45.6	58.0	5.2	0.92	0.08	0.54
	SAND	0.4	1.4	3.8	40.6	25.4	31.8	0.44	0.56	0.25
	TOTAL	**12.5**	**47.8**	**3.8**	**41.5**	**36.1**	**22.0**	**0.62**	**0.38**	**0.51**

Notes:

1) JORC 2012 definitions were followed for estimating Mineral Resources;
2) Mineral Resources are estimated at a cut-off grade of 25 % Fe;
3) Mineral Resources are estimated using a five year historical average price of US$ 100 per tonne (Source: IndexMundi); and
4) Figures may not total due to rounding errors.

13.12 Conclusions

The Blötberget apatite-iron oxide deposit is located in the western part of the intensely mineralised Paleoproterozoic Bergslagen Province in south central Sweden.

The deposits in the neighbouring area occur along a ~40 km long, broad zone. This zone of mineralisation is the third largest iron ore deposit in Sweden by production, only outnumbered by the giant Kirunavaara and Malmberget iron ores in Norrbotten, northern Sweden.

The mineralised zone at Blötberget appears as a set of vertically narrow, elongated lenses dipping 50°–70° to the SE. Airborne geophysical surveys and historical drillholes indicate that mineralisation extends to a depth of at least 900 m below surface.

The Blötberget field consists of five mineralised bodies, from west to east, these are: Kalgruven; Flygruven; Hugget and Betsta (The Wedge); Sandell.

Mining and exploration in the Ludvika area has been carried out in different periods since the 1600's. The majority of this small scale mining was focused on iron production.

NIO applied for a mining concession within the Blötberget area in October 2010 and it was granted by the Mining Inspectorate of Sweden in August 2011. The mining concession, which runs for 25 years with possibility of extension, implies the right of exploitation and utilisation of iron, rare earths, and apatite. The environmental permit for this Project was granted in late March 2014.

The Hugget and Kalvgruvan/ Flygruvan zones had previously been mined down from near-surface to the 200 m and 240 m levels respectively. The units dip towards the southeast at between 50° and 55° in the near-surface mined-out areas, and flatten at depth to ~25°.

A drilling programme was undertaken by NIO during the summer and winter of 2012 and was completed in November 2012. This 16 hole programme included drilling to confirm the quality of historical drilling data, as well as infill and step-out drilling. NIO completed 16 drillholes totalling 7,430 m of drilling.

The 2014 drilling programme was designed to investigate the area between Flygruvan/Kalvgruvan and Hugget (formally known as “the wedge” or Betsta area) and to infill the intermediate depth extension of Hugget, in order to improve the confidence of the geological model. 13 drillholes, totalling 7,093 m, were drilled.

'The Wedge' was successfully explored during the 2014 drilling programme and, as a result, Kalvgruvan and Hugget/Flygruvan have now been shown to be continuous zones of mineralisation

Mine maps and historical drilling data have been collected from various sources and digitised, where possible. Drill core from historical exploration drilling in the Blötberget project area has been recovered, re-logged and re-analysed.

DMT was provided with a comprehensive set of historical reports and data which have been collated and used in conjunction with data collected more recently by NIO in order to estimate and report Mineral Resources for the Blötberget Project in accordance with JORC standards.

In the resource development programme of 2012 and 2014 NIO completed industry standard QA/QC programs to ensure the data is reliable and suitable for resource estimation. The drill density of the resource is adequate for the purpose and is reflected in the JORC compliant resource category classifications of Measured, Indicated and Inferred Mineral Resource.

DMT has relied heavily upon the information provided by NIO, however DMT has, where possible, verified data provided independently during the site visits.

DMT was able to overlay licence information on the Mineral Resource estimate area to confirm that the deposit is within NIO's license. DMT has not undertaken a legal review of the licences and assume that all the required licences are in place.

The geology of the deposit is fairly well understood and DMT has constructed a wireframe geological model for the Blötberget deposit based upon a combination of logged lithologies and analytical and SATMAGAN magnetite results. This has allowed the splitting of the deposit into geological domains comprising, magnetite-rich material of KALV and hematite-rich material of HUGFLY and SAND.

DMT has undertaken a statistical study of the data, which demonstrates adequate splitting of the data into single iron population domains, and undertaken a geostatistical study to investigate the grade continuity and to provide grade estimation parameters for Ordinary Kriging.

A Surpac block model using all the available geological and sample analytical test data has defined an iron ore resource. At this stage of the investigation most of the mineral resources of Blötberget have been classified into the Measured and Indicated categories.

As a result of the site visits, data base verification and validation and the geological and model generated therefrom, DMT has estimated the total Measured and Indicated Resources for the Blötberget Project as 47.8 Mt at a grade of 41.5 % Fe (Total) and 0.5 % P at preliminary COG of 25 % Fe. Of the total estimated contained Fe, the magnetite-hematite ratio is estimated at 62:38.

13.13 Recommendations

13.13.1 Further Drilling

For the current Resource Estimate it is considered that there is only limited additional geological information that can be gained from further, expensive, surface drilling programmes. The bulk of the upper levels of the Blötberget deposit that have been identified as part of the proposed mine plan are within the Measured Resource category. However, surface drilling for rock mechanical/structural and or metallurgical information for detailed mine planning should be considered.

Definition and Grade Control drilling should commence as soon as there is access to the underground areas after dewatering. This close spaced drilling is required to support the transfer of Measured Resources into (Proven) Reserves. The underground drilling should follow a similar drill approach to that used historically, with fan pattern of close spaced drilling into the mine blocks , typically at 35-45 m centres, with wider spaced (100 m) deeper down dip drilling to provide increased confidence in the Indicated area of the resources.

13.13.2 Further Studies

Additional hydro-geological investigations on existing drillholes should be undertaken, as DMT considers that insufficient data exists on the hydrological and hydrogeological conditions for underground mining.

13.13.3 Sampling

There was no use of check samples in the historic core re-assay (BGU), this should be addressed as a partial re-run with standards inserted.

The blank samples assayed to date have indicated between 1 % and 2 % Fe. Prior to further analysis being undertaken, the preparation of suitable blanks for insertion into future sample streams should be addressed by NIO.

Check standards have slightly (but consistently) undervalued the results, this should also be corrected ahead of the next phase of core sampling, which will likely be from underground drill locations.

NIO should continue to source historical data and drill core for the purposes of re-assaying, re-logging and integration into the current database.

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http://www.indexmundi.com

Competent Person’s Consent Form

DMT Consulting Limited
International Mining Consultants

Pure Offices
Lake View Drive, Sherwood Park
Nottingham, NG15 0DT
United Kingdom

Email: UK@dmt-group.com
Web: www.dmt-group.com

Competent Person's Consent Form

Pursuant to the requirements of ASX Listing Rules 5.6, 5.22 and 5.24 and Clause 9 of the JORC Code 2012 Edition (Written Consent Statement)

Report name

Mineral Resource Estimate – Blötberget Iron Ore Project

(insert name or heading of Report to be publicly released ("Report"))

DMT Consulting Limited

(insert name of company releasing the Report)

Blötberget Iron Ore Project

(insert name of the deposit to which the Report refers)

10th April 2015

(Date of Report)

DMT Consulting Limited
International Mining Consultants

Pure Offices
Lake View Drive, Sherwood Park
Nottingham, NG15 0DT
United Kingdom

Email: UK@dmt-group.com
Web: www.dmt-group.com

Statement

I/~~We~~

Florian Lowicki

(insert full name(s))

confirm that I am the Competent Person for the Report and:

- I have read and understood the requirements of the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012 Edition).
- I am a Competent Person as defined by the JORC Code 2012 Edition, having five years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
- I am a Member or Fellow of The Australasian Institute of Mining and Metallurgy or the Australian Institute of Geoscientists or a 'Recognised Professional Organisation' (RPO) included in a list promulgated by ASX from time to time.
- I have reviewed the Report to which this Consent Statement applies.

I/~~We~~ am a full time employee of

DMT

(insert company name)

and have been engaged by

Nordic Iron Ore

(insert company name)

To prepare the documentation for

Blötberget Iron Ore Project

(insert deposit name)

On which the Report is based, for the period ended

30th January 2015

(Insert date of Resource statement)

I have disclosed to the reporting company the full nature of the relationship between myself and the company, including any issue that could be perceived by investors as a conflict of interest.

I verify that the Report is based on and fairly and accurately reflects in the form and context in which it appears, the information in my supporting documentation relating to Mineral Resources.

DMT Consulting Limited
International Mining Consultants

Pure Offices
Lake View Drive, Sherwood Park
Nottingham, NG15 0DT
United Kingdom

Email: UK@dmt-group.com
Web: www.dmt-group.com

Consent

I consent to the release of the Report and this Consent Statement by the directors of:

DMT Consulting Limited

(insert reporting company name)

10th April 2015

Signature of Competent Person

Date

Pr.Sci.Nat Geol. (SACNASP)

400425/13

Professional Membership
(insert organisation name)

Membership Number

Tim Horner – Nottingham, UK

Signature of Witness

Print Witness Name and Residence
(e.g. town/suburb)

Appendix A
Sampling Procedure Manual

Procedure manual

Note: As a prerequisite to carrying these tasks you must firstly have read the specific machinery manuals and procedures from the manufacturer and secondly you must have sign off by a competent trainer before undertaking any task.

1.1 Core handling

1.1.1 Collection of the core from the drilling team.

1.1.2 Make a quick measurement to ensure all core boxes are labelled correctly and that all core blocks are in the correct place.

1.1.3 Make sure all boxes are lifted in a correct way, using two people at the time.

1.1.4 When transporting the core, make sure to use ratchet straps!

1.1.5 Core is to be stored in Grängesberg at Inbox wall.

1.1.6 When geologist starts working with a drill hole, update the DDH-info onto the whiteboard.

1.1.7 Place boxes onto rollers in order i.e. 0.00 tray first. Remember to use wooden core stop at each end of the roller.

1.1.8 Prepare to measure meter marks on core and hole consistency checks.

- You need: Tape measure and a black permanent marker

1.1.9 Meter measuring

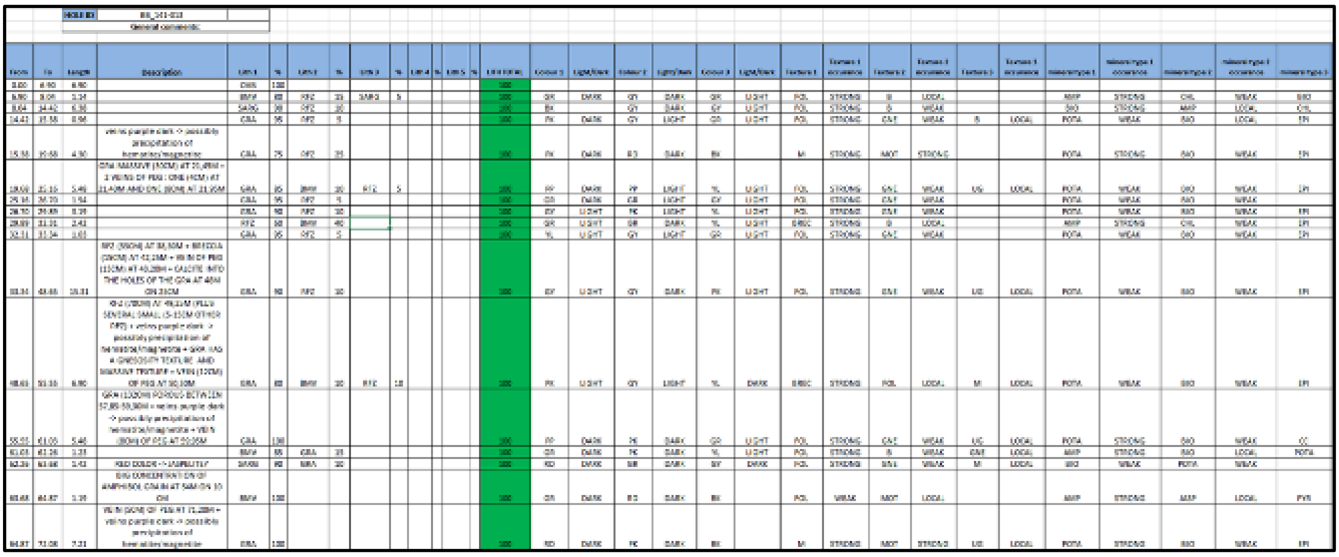
- Push all core to the left and match fractures/breakages while measuring metre marks
- Check each box for labelling consistency i.e. core blocks, hole ID, tray number.
- Check that core blocks match tray meterage.
- If there are any issues with this: measure further on and see if it fixes itself, refer to daily drilling protocols then refer to drillers ASAP.
- Check that all markings are legible.
- If mistakes are discovered core boxes and core blocks must be sanded off and relabelled and a note made on the logging of the error. This must then be fed back to the drilling contractor as we could be over or under charged. It is essential that this data is captured otherwise our logging and sampling will be incorrect.

1.2 Logging

1.2.1 Prepare for geological log and sampling.

- Tape measure
- Magnetic pen
- Scribe
- Biro pen
- Hand lens
- Old log/protocol
- Surpac section
- Notepad
- Sample tag book
- Water bottle
- Acid bottle and protective eyewear
- UV-lamp and black towel
- Handheld XRF

1.2.2 Log using the drop down logging template located on the server. Rename before saving as this will overwrite the blank version. Place template under the respective deposits folder in KARTERINGSLOGG.



1.2.3 Logging should have a 1cm accuracy to boundaries. Core over **1m** should be logged as separate lithologies if there is a change, unless it is ore which should be greater than **30cm**. **If the magnetite or hematite content is 15% or more, always put it as lithology 1.**

1.2.4 Firstly walk along the whole core and note down lithological changes, colour changes, take a mag pen with you and run across as you do so and note magnetic sections. Wet core wherever needed to assist with logging.

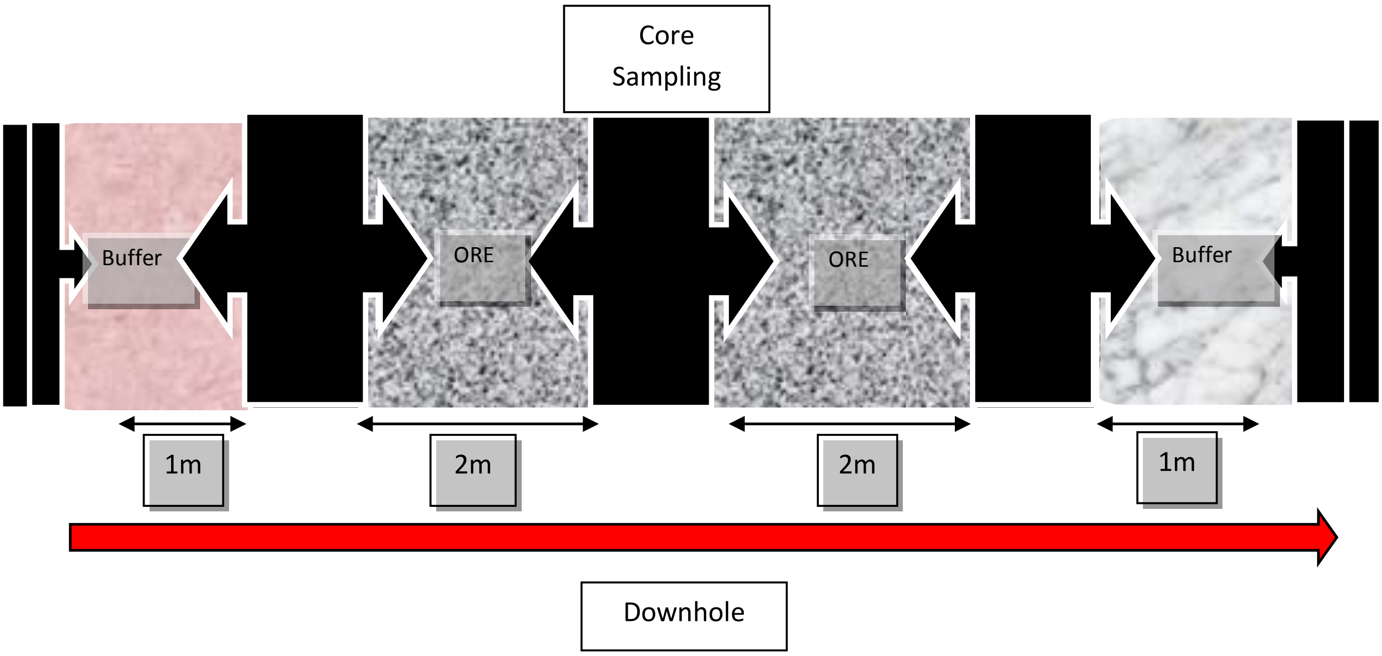
1.2.5 Before starting with the XRF and Magnetic Susceptibility, ensure that you have completed training with an experienced and competent person. For the XRF that means a person with a certificate signed by the manufacturer. Mark the test points for XRF and Magnetic Susceptibility with a biro pen**. Make sure the six XRF points are in the same spots as the six Magnetic Susceptibility points**. Use about 10 seconds per XRF sample point, or however long it takes to get a reasonably constant Fe-value. If unsure seek assistance from the NIO geologist on site.

1.2.6 When using the XRF for mineral identification, make sure to have the correct settings and measuring time.

1.2.7 When using the UV lamp, make sure it is set to A/C so you can see both short- and long-wave responsive minerals. **Never look directly into the UV-lamp.**

1.3 Sampling of core

1.3.1 Sample all material with Fe over 10%, greater than 50cm in length and **within a mineralised section**. If unsure seek assistance from the NIO geologist on site.



1.3.2 Use lithological boundaries to section the mineralised core. Samples should preferably be <>2m length.

1.3.3 Sample 1m before and 1m after mineralised zone and create these as separate samples.

1.3.4 Mark sample intervals on the core and on the boxes with black permanent marker (see figure above).

1.3.5 Write sample depths with biro pen at start and end of sample intervals on core boxes.

1.3.6 If necessary draw a saw cut line with permanent marker on core.

1.4 Allocating sample ID

1.4.1 Check last sample book used for first sample number and QA/QC insertion type and sequence.

1.4.2 Allocate sample id into sampling tab on the logging spread sheet

1.4.3 Cross check core marking and sample tags for errors

1.4.4 Insert the numbered sample tags into the core box next to the start of the section to be sampled – write hole name on new sample booklet and the sample series used– write date – hole id – total sample length of section- sampler name

- Insert certified standards, blanks and duplicates into the sample train at the rate of one per every 5 samples. Make sure high grade material gets a high grade standard and that the name of the standard used is recorded in the sampling sheet.

1.4.5 Enter data into logging spread sheet and save file under correct directory on server. Update the whiteboard to reflect completion of sampling and add to sawing list under the GEO tab.

1.5 DCSO and RQD

1.5.1 Start with moving the core over to the v-rail while matching fractures/breakages along the core. Locate the orientation line starting point made by the drillers. You might need to have several runs in the rail at the same time to make a match between two starting points. Use the edge of the v-rail and a red permanent marker to draw the orientation line. If you have any problems with matching the fractures/breakages and the foliation of the rock doesn't give you confidence enough, then skip the orientation line in this section. No data is better than false data.

1.5.2 As you start moving the core back into the box take a careful look at all fractures and separate the manmade fractures from the natural ones. All manmade fractures are marked with a black X. The midpoint of the X should be over both the fracture and the orientation line. If you are unsure leave the fracture as natural.

1.5.3 All natural fractures need to be measured with help of the DCSO tool. Measure the alpha and beta angles as seen in the description from Petro Team Engineering.

1.5.4 While you are holding the core, also estimate the J_R and J_A value for all natural fractures. The estimation is made according to Barton's Q classification chart and added into the same DCSO logging sheet as the information above. Update the whiteboard to reflect completion of DCSO measurements.

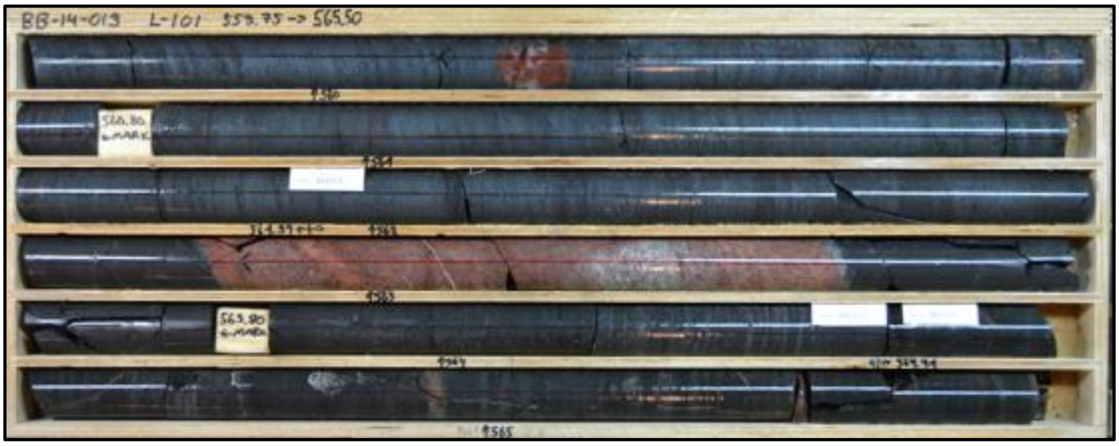
1.5.5 When all core is back in the box, do RQD. Start with dividing the core into intervals of similar RQD, zones that have the same RQD value. Mark the edges of each interval with a **+** using a green permanent marker. Note that all core loss should be recorded in the RQD measurement as RQD 10. If the RQD is the same as the previous zone then it should be merged as one zone and included in the RQD measurement.

- Measure and note each RQD interval
- Measure all core pieces over 10 cm in length. The total length of these pieces divided by the total length of the interval will give you a value that if multiplied with 100 is the RQD value for the interval. All RQD values below 10 are

recorded as 10! Enter the RQD value in the same software as for DCSO and J_R, J_A. Make sure each row is populated with the corresponding RQD value.

- For core loss or other intervals with RQD 10 make sure they are entered separately into the program. One row for the starting depth and one row for the finishing depth. Make sure to copy the comments from the first to the second row so anyone could easily see that it's the same RQD zone.
- Update the whiteboard to reflect completion of RQD.

1.6 Photographing core



Wet core photograph example

Dry core photograph example

1.6.1 Prepare camera and check if battery is charged (there are two)

1.6.2 Take pictures of each box individually dry and then wet with the Nikon D3100. Check the first few pictures to ensure they have come out correctly i.e. in focus, straight, not too dark or light and no obstructions. Once complete name these photos using the naming convention: VAS_12-011_L1_0.00-13.80_dry or VAS_12-011_L2_13.80-24,70_wet.

1.6.3 Ensure that the orientation line and sampling tags are visible in the picture.

1.6.4 Ensure all photos are then loaded onto the server under their respective folder in G drive before moving onto the next hole. Also ensure to remove your pictures from the camera memory.

- Ensure camera is turned off once finished. Update the whiteboard to reflect completion.

1.7 Point load

1.7.1 Open the point load template from the G: drive, rename the file to the current hole name.
1.7.2 Ensure you have the geological logging file
1.7.3 Copy the primary logging codes and interval lengths.
1.7.4 If the rock has foliation take six measurements – three parallel and three perpendicular within 15-20cm - close to each other to ensure that the measurements are taken on the “same” rock.
1.7.5 **Select a homogenous piece of core that represents the rock type**. If there is a lot of variation in the core take more readings. If there is a discrepancy and differing rock types haven’t been recorded then this needs to be fed back to the geologist.
1.7.6 Perpendicular is the preferred point load test to take if there is nothing else to choose from. If the rock is badly fractured then point load cannot be taken and should be skipped.
1.7.7 Make sure to calibrate the point load each day before taking any measurements. To calibrate; pump until the cones meet and then press the lower cone back to the bottom. Repeat three times. Make sure to use the pumping stick as much as possible when lowering the cone, don’t press the arm. When this is done set the display to *ZERO* and *PEAK.*
1.7.8 Between each measurement
- Clean the sides of the cone
- Open the valve
- Press the cone down to at least 7
- Close the valve
- Pump three times
- Set to *ZERO*

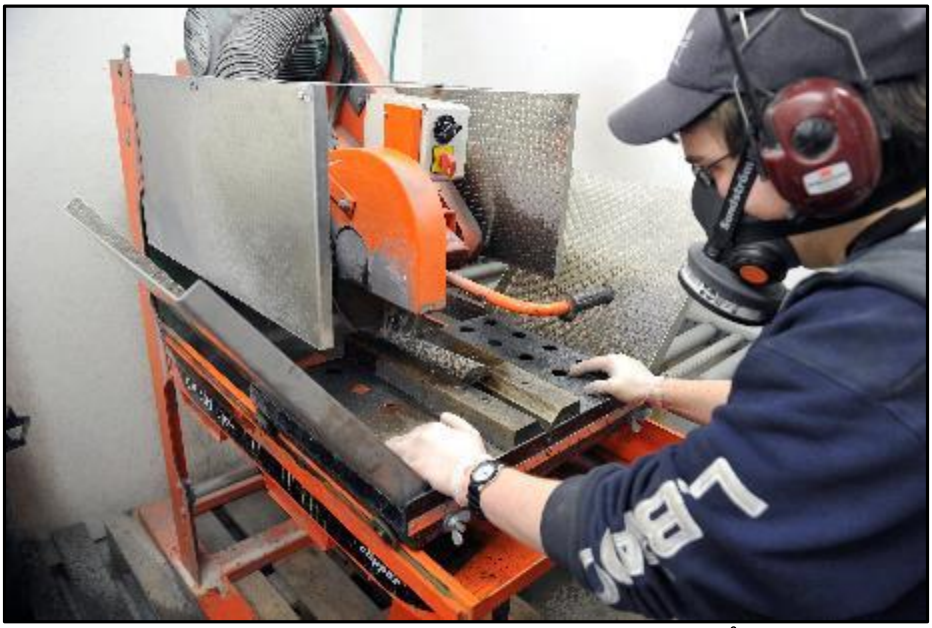
1.7.9 Photograph if necessary.

1.7.10 Make sure the file is saved with the right naming convention under *Geotechnical* log -> *Point load* folders.

1.7.11 Once point load is complete ensure hole is stored as per Core Storage protocols.

- Pallets containing non-sampled core shall be put in the temporary area. The technician then labels the pallet and moves it to its right place in the storage area.
- Pallets containing sampled core shall be put; near the exit roller by the sawing room, while the actual samples for sawing are temporarily put on a pallet under the “electrical box”. After sawing it, the technician returns the sampled pallets onto the pallet containing the remnants of the hole in order and moves completed hole to storage area labeling pallet.
- 22mm core is packed 3*15 boxes for one pallet. 38mm core is packed 3*10 boxes per pallet. 63,5mm core is packed 3*8 boxes per pallet. 51mm core is packed 3*7 boxes per pallet.

1.8 Sawing



1.8.1 Remove the sampling from GEO tab and place it under SÅG tab, print the sawing list.

1.8.2 Read sawing list then take sample bag and mark the bag with the sample number using a black permanent marker

1.8.3 Before you start sawing:

- No personnel is authorized to operate saw without training and sign off by authorized operator.
- Turn the fan on, located under the camera stand.
- Turn the water on, located next to the sink behind the sawing both. One handle for warm water and one handle for cold water.
- Make sure you have the appropriate safety gear on (see the signs next to the door in to the sawing both
- Make sure the size of the core guide matches the size of the core you are about to saw.
- Inspect the blade for wear and tear and make sure it isn’t skewed.
- Turn the main switch on and turn the tachometer to 400.

- When turning everything off, repeat steps in reverse.

1.8.4 Technician - saw a third of the core along the saw cut line (if present or use orientation line) and insert the samples into the sample bags with corresponding sample numbers, and put in the correct sample number in the bag with the sample then update whiteboard.

1.8.5 Load plastic bags into oven for drying overnight minimum 12 hours drying – check moisture content of bags before completing density.

1.9 Density

1.9.1 Note: density measuring is conducted on all drill core samples

1.9.2 Make sure that the water is room temperature by filling it up a day in advance

1.9.3 Make sure that the correct samples and lists are provided by geologist and make sure that the samples all match the list provided.

1.9.4 Ensure that you have completed density training with an experienced and competent person before completing task and if any problems or discrepancies arise during measuring – talk with your supervisor.

1.9.5 Calibrate both scales before use and every 40 samples.

1.9.6 Weigh the sample in the metal bowl try to get at least1.1kg of sample or take the entire sample (you can only weigh what you have) – **Make sure the scale stops before entering value in the density list.**

1.9.7 Weighing in water. Take the first metal bowl and pour the drill core into the colander.

1.9.8 Ensure all material from the metal bowl enters the colander, bang all sample out and clean with a dry cloth and put material into colander as well.

1.9.9 Clean metal bowl with a lightly moistened cloth and allow drying then reuse.

1.9.10 Lower the colander into the water slowly. **Make sure the scale stops before taking reading – this can take a while as the scale is extremely sensitive 0.05g so walking or doors closing can change the result.** Enter the whole value into list.

1.9.11 Lift the colander out of the water, and put the drill core back into the sample bag (leave bag open and place ready to go to the lab.

1.9.12 Go back to step 1.9.5 and repeat.

1.9.13 Check all data for errors

1.10 Sample dispatch

1.10.1 Fill in the sample batch and save a copy under correct directory on server.

1.10.2 Remove air from sample bag, staple and load into cardboard box. Label the box with "*Nordic Iron Ore Sample no: X-X*".

1.10.3 Maximum weight for each box is 35kg as this is the weight limit of Bussgods.

1.10.4 Send samples to a certified laboratory together with the sample batch.

1.11 Sample Received

1.11.1 Place the "fraktsedel"- paper in the folder called "mottaget gods".

1.11.2 Update the whiteboard

1.12 Magnetic Susceptibility

1.12.1 Taking the measurements

- Make sure to not place the bag on top of a metallic surface during measuring.
- Shake bag to hand homogenise.
- Take measurements on each side and then the bottom, recording results in the spreadsheet Magsus_Satmagan placed on G: drive.

1.13 Satmagan

1.13.1 To prepare the samples –
- Select correct sample and add to ampule ensuring ampule is clean.
- Insert certified standards, blanks and duplicates into the sample train at the rate of one per every 5 samples.

1.13.2 Starting the machine
- Turn power on and let machine warm up for 20 minutes prior to use.
- Run through all ten calibration samples and record in Magsus_satmagan spreadsheet – See training manual.

1.13.3 Taking the measurements
- Fill data into the file called Magsus_Satmagan on G drive.

1.13.4 Update whiteboard to reflect on completion

- Clean all ampules before use ensuring you wear your PPE due to dust:

Appendix B

List of Historical Geological Maps & Sections

Deposit	old level m	new level m	horizontal	Comments	deposit	old level m	new level m	horizontal
Hugget-Sandell	0	40	Y		Kalvgruvan	0	40	Y
Hugget-Sandell	80	120	Y		Kalvgruvan	10	50	Y
Hugget-Sandell	120	160	Y		Kalvgruvan	20	60	Y
Hugget-Sandell	160	200	Y		Kalvgruvan	25	65	Y
Hugget-Sandell	200	240	Y		Kalvgruvan	30	70	Y
Hugget-Sandell	240	280	Y		Kalvgruvan	45	85	Y
Hugget-Sandell	280-1	320	Y		Kalvgruvan	50	90	Y
Fredmundsaberget	280-2	320	Y	exploration drift to the East	Kalvgruvan	60	100	Y
Fredmundsaberget	280-3	320	Y	exploration drift to the East	Kalvgruvan	70	110	Y
Fredmundsaberget	280-4	320	Y	exploration drift to the East	Kalvgruvan	80	120	Y
Fredmundsaberget	280-5	320	Y	exploration drift to the East	Kalvgruvan	90	130	Y
Hugget-Sandell	330	370	Y		Kalvgruvan	100	140	Y
Hugget-Sandell	380	420	Y		Kalvgruvan	140	180	Y
Hugget-Sandell	430	470	Y		Kalvgruvan	160	200	Y
Hugget-Sandell	480	520	Y		Kalvgruvan	180	220	Y
Hugget-Sandell	530	570	Y		Kalvgruvan	200	240	Y
Vertical section_E1				Overview of vertical section 18-39	Kalvgruvan	220	260	Y
Vertical section_10					Kalvgruvan	260	300	Y
Vertical section_11					Kalvgruvan	280	320	Y
Vertical section_12					Kalvgruvan	300	340	Y
Vertical section_13					Kalvgruvan	320	360	Y
Vertical section_14					Kalvgruvan	340	380	Y
Vertical section_15					Kalvgruvan	360	400	Y

Deposit	old level m	new level m	horizontal	Comments	deposit	old level m	new level m	horizontal
Vertical section_16					Vertical section_1			
Vertical section_17					Vertical section_2			
Vertical section_18					Vertical section_3			
Vertical section_19					Vertical section_4			
Vertical section_20					Vertical section_5			
Vertical section_21					Vertical section_6a			
Vertical section_22					Vertical section_6b			
Vertical section_23					Vertical section_7			
Vertical section_24					Vertical section_8			
Vertical section_25					Profile 1-9 C-D			
Vertical section_26					Profile 1-9 E-F			
Vertical section_27					Profile 3-5 A-B			
Vertical section_28								
Vertical section_29								
Vertical section_30								
Vertical section_31								
Vertical section_32								
Vertical section_33								
Vertical section_34								
Vertical section_35								
Vertical section_36								
Vertical section_37								
Vertical section_38								

Deposit	old level m	new level m	horizontal	Comments	deposit	old level m	new level m	horizontal
Vertical section_39								
Vertical section_40								
Vertical section_41								
Vertical section_42								

Appendix F

GEOTECHNICAL REPORTS

Rock mechanics evaluation

NORDIC IRON ORE AB

Anläggningsprojektet Ludvika gruvor, Blötberget

Göteborg 2015-03-25

Anläggningsprojektet Ludvika gruvor, Blötberget

Rock mechanics evaluation

Date	2015-03-25
Project number	1320001429
Status	Final report

Per-Erik Söder	Linn Karlsson Mikaela Bäuml	Thomas Andersson
Uppdragsledare	Handläggare	Granskare

Revision 2015-03-25

All fracture orientations are now labeled according to the right hand rule. For example, the ore orientation is now N090E instead of ~~N270E~~ used in the previous version.

Revision 2014-12-16

Changes from the report posted on Byggnet 2014-12-02:

- Boreholes from 2014 are added to Figure 2.
- Figure 13 showing RQD for the ore has been updated with mapping of drill cores from 2014.
- Chapter 2.3 (Hydrogeological tests) is updated with tables and figures.
- Appendix 6 has been added.
- Chapter 2.4 (structures from old mine maps) has been updated with a new figure (Figure 22) showing the joint orientations for the 2014 drill cores.
- Background, Conclusion and Recommendations have been slightly altered to reflect the above mentioned revisions.
- All analyses have not been updated with data from the 2014 and 2012 drill cores, since the rock quality and fracture orientations are similar in general. In addition, the drill cores from 2012 and 2014 are not entirely mapped and more information can be collected from more detailed studies of the cores. All analyses are recommended to be updated with all mapped data when all drill cores are entirely mapped in order to create a more detailed evaluation of the Blötberget mines.

Table of contents

Appendix

1. Background

This study is made as part of the technical studies for the proposed mining in Blötberget, Ludvika. This report is a compilation of geotechnical work based mainly on RQD loggings of drill cores performed in 2012, old mine maps and previously conducted geological studies. Still, some material from both the 2012 and 2014 drilling operations is missing in order to get a full geotechnical model of the Blötberget mines.

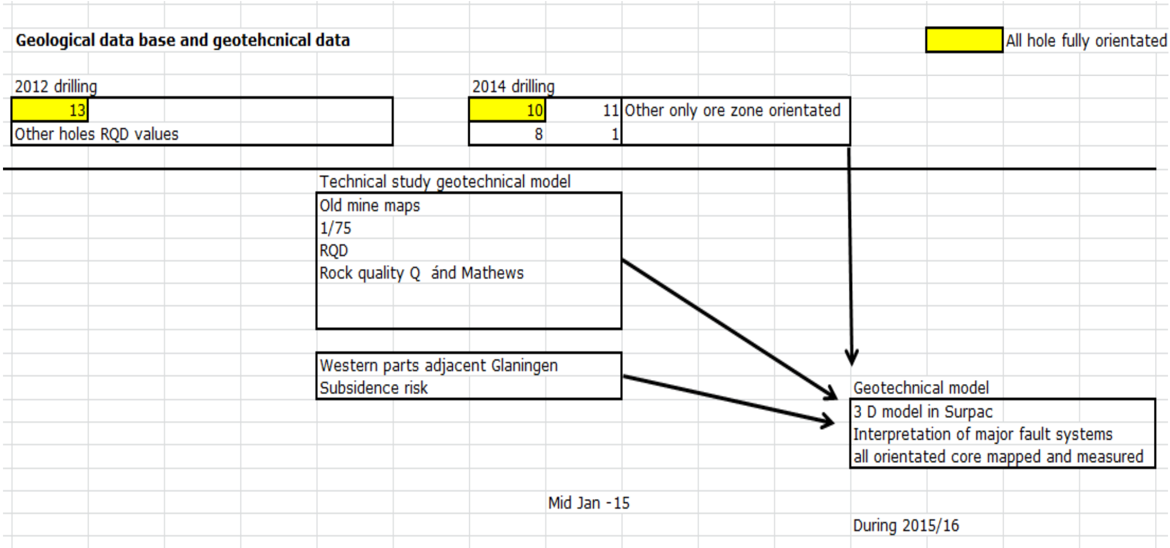


Figure 1. The data in this study is based on parts of the drilling program material available. The material is not yet mapped in full and put into the geological data base for the mine. Only two fully orientated drill cores are included in the material treated in this report; borehole 2012013 and 2014010.

In the Blötberget mines, sublevel caving was conducted to some extent up to the finalisation of mining in the early 1960's for the Kalv/Fly mine and in 1976 for the Hugget/Betsta mine. Open stoping method was also tested in the Betsta orebody. This may have some implications on planned new mining operations, and, thus, a mapping of the old mine workings is needed to determine the adjacent mining area's stability.

2. Geotechnical compilation

The geotechnical work is a compilation of base data such as

- RQD
- Point load tests
- Water pressure tests

- Structural mapping from old mine maps. Fracture orientations are according to the right hand rule if not stated otherwise.

2.1 RQD

A compilation of mapping data such as RQD values were made on boreholes BB12001, BB12002, BB12004, BB12005, BB12006, BB12007, BB12008, BB12013, BB14001, BB14002, BB14003, BB14004, BB14005, BB14006, BB14007, BB14008, BB14009, BB14010, BB14011, BB14012, BB14013 and BB1/75, see Figure 2. The aim was to determine variation in fracturing within and around the ore body. Drill cores are located in connection to the Kalv/Fly and Hugget/Betsta mines, see Figure 3 and Figure 4, and they are drilled from hanging wall to footwall (HW→ore→FW).

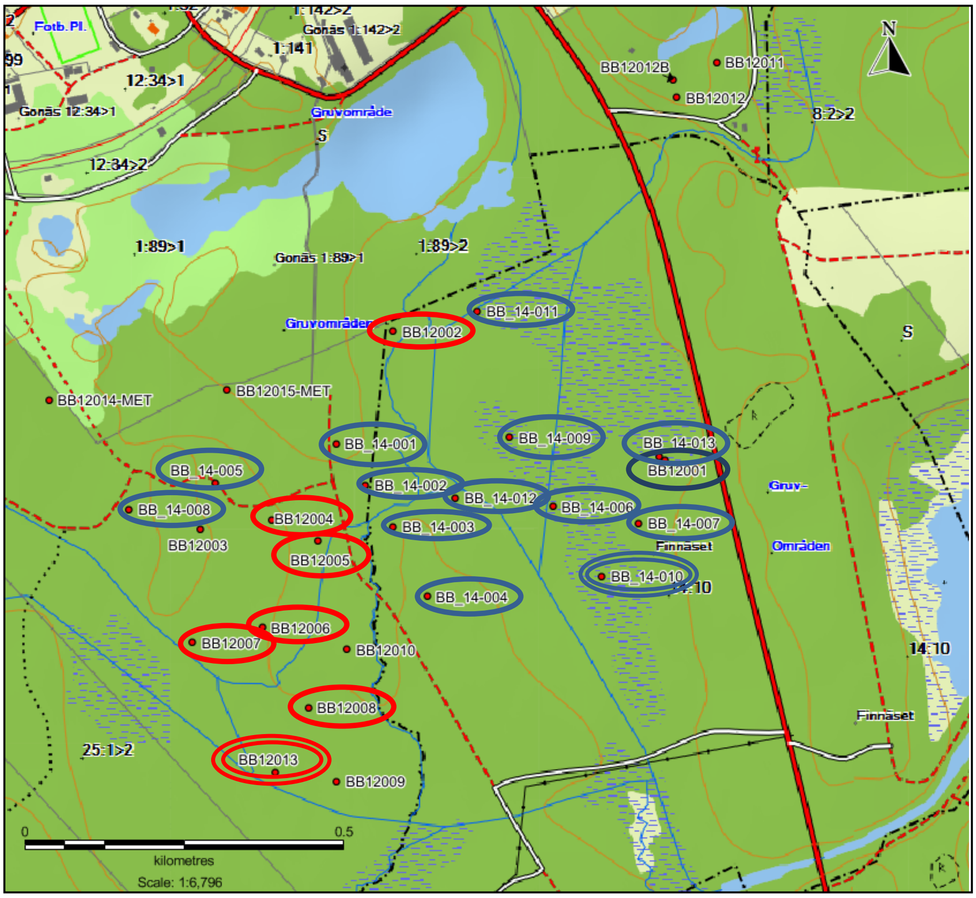


Figure 2. Locations of main boreholes used in analyses. Boreholes from 2012 are marked with red and from 2014 with blue. Boreholes marked with two circles are entirely mapped and oriented.

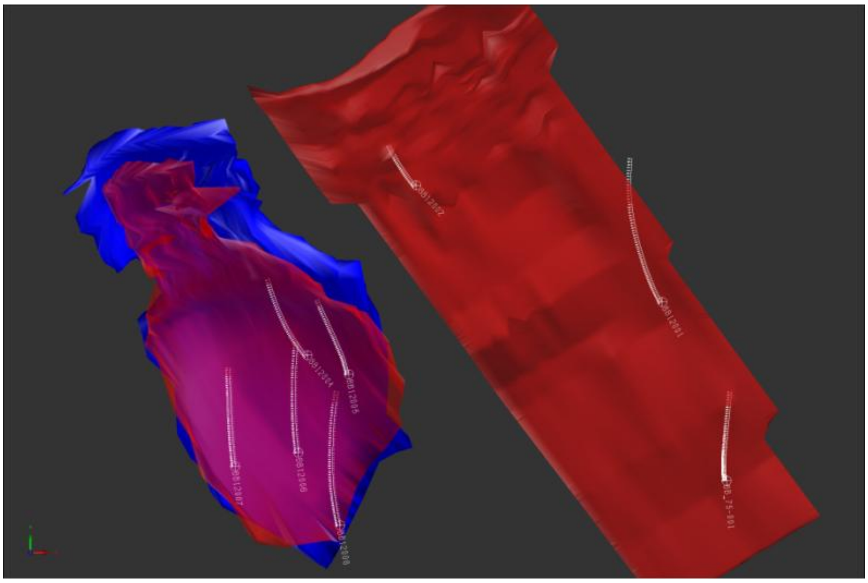


Figure 3. View of ore bodies and location of studied cores (2012), where the Kalv/Fly mine is purple/blue and Hugget/Betsta is red.

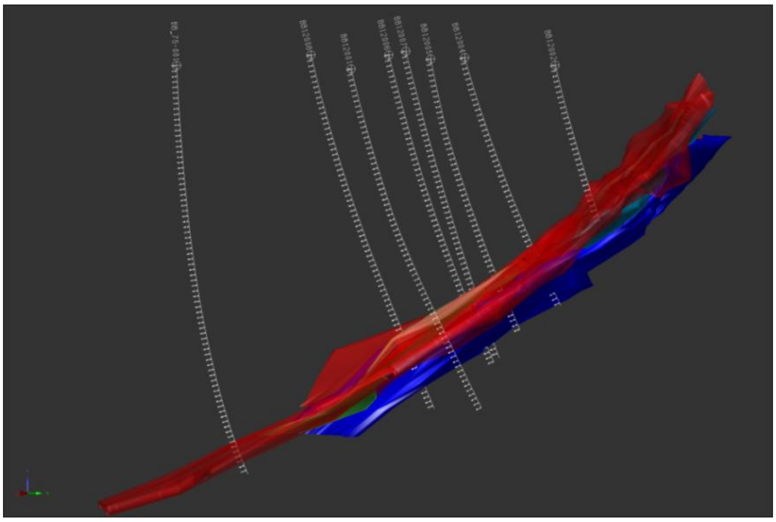


Figure 4. Cross section of ore body and location of drill cores (2012), starting from the ground surface.

The compilation of RQD (Figure 5–Figure 8) is based on the average value of approximately 50 m of the footwall and 50 m of the hanging wall. The average value for the ore body is made for rock containing more than 20% of iron (mineralisation>20%). The hanging wall is divided into the adjacent hanging wall (0–25 m from the ore body) and the distant hanging wall (25–50 m from the ore body).

Figure 5 shows RQD for each borehole at 300–800 m depth. The RQD-values are mainly in the range 80-100, with some lower values, down to 50, in one borehole. Little or no variation was detected at depth.

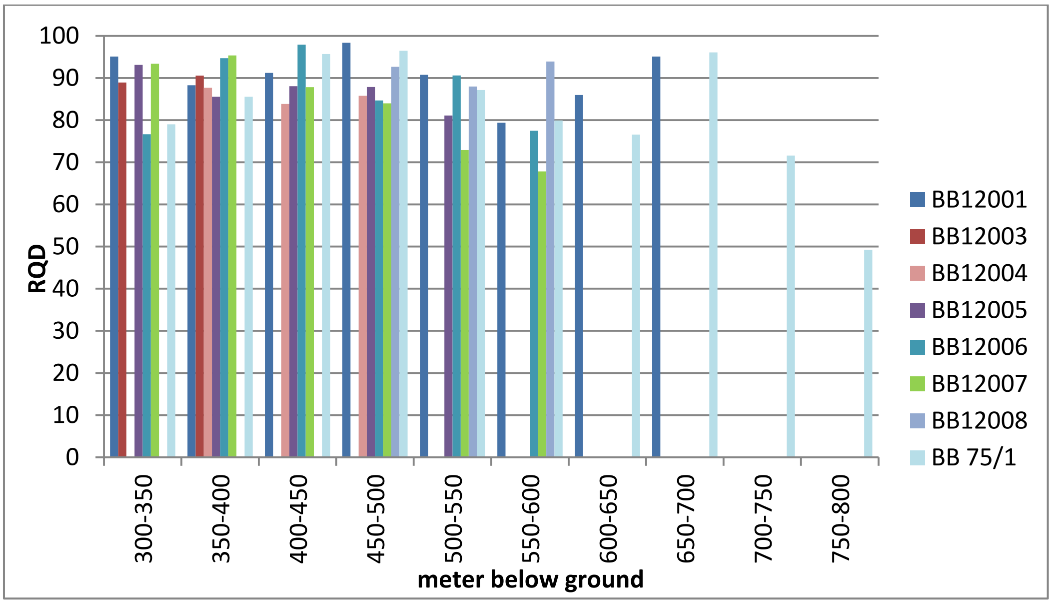


Figure 5. Distribution of RQD at different depths for boreholes BB12001, BB12003-8 and BB 75/1.

Figure 6 shows the RQD values in the hanging wall divided into the adjacent hanging wall (0-25 m from the ore body) and the distant hanging wall (25-50 m from the ore body). The cores show generally small variations in RQD-values and are mostly in the range of 70-90. Similar ranges can be seen for the ore body and the footwall, see Figure 7 and Figure 8.

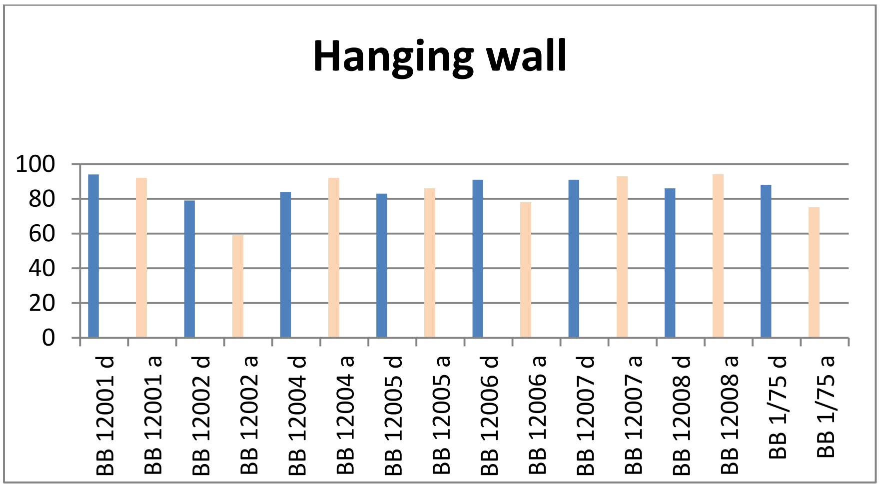


Figure 6. RQD values in the distant hanging wall (25-50 m) and the adjacent hanging wall (0-25 m) for boreholes BB12001-2, BB12004-8 and BB 75/1.

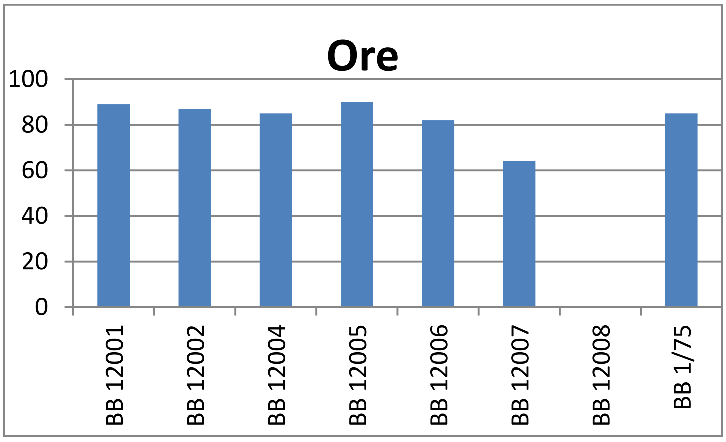


Figure 7. RQD values for the ore body for boreholes BB12001-2, BB12004-8 and BB 75/1.

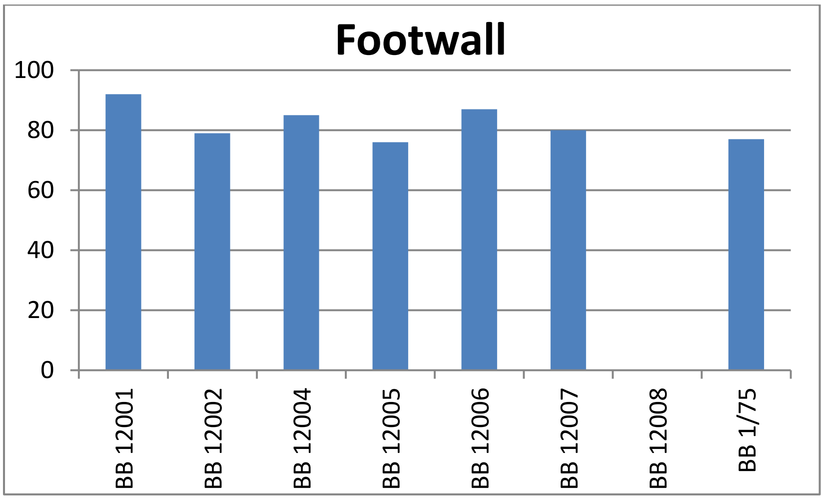


Figure 8. RQD values for the foot wall in borehole BB12001-2, BB12004-8 and BB 75/1.

Figure 9 to Figure 12 show some images of the drill cores for sections with more fractured rock, giving lower RQD-values.

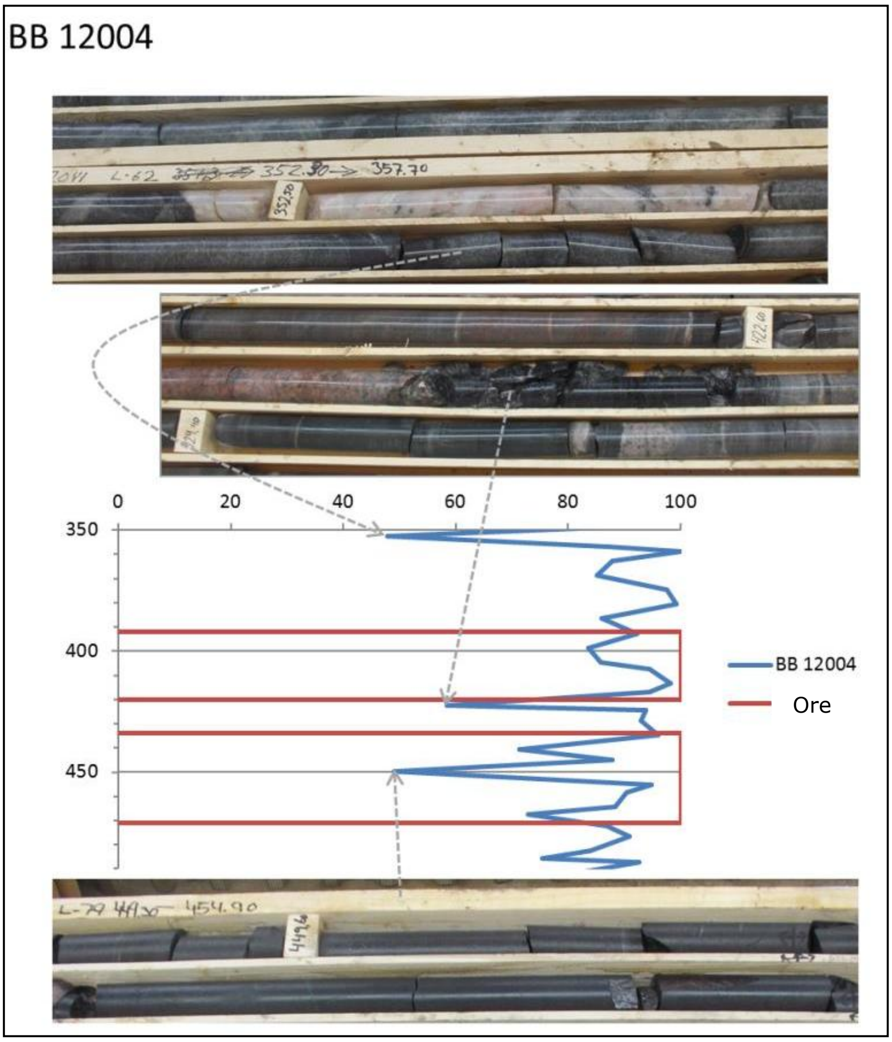


Figure 9. Core log of BB 12004, showing the RQD and core photos in sections with higher fracture frequency in the mineralized zone.

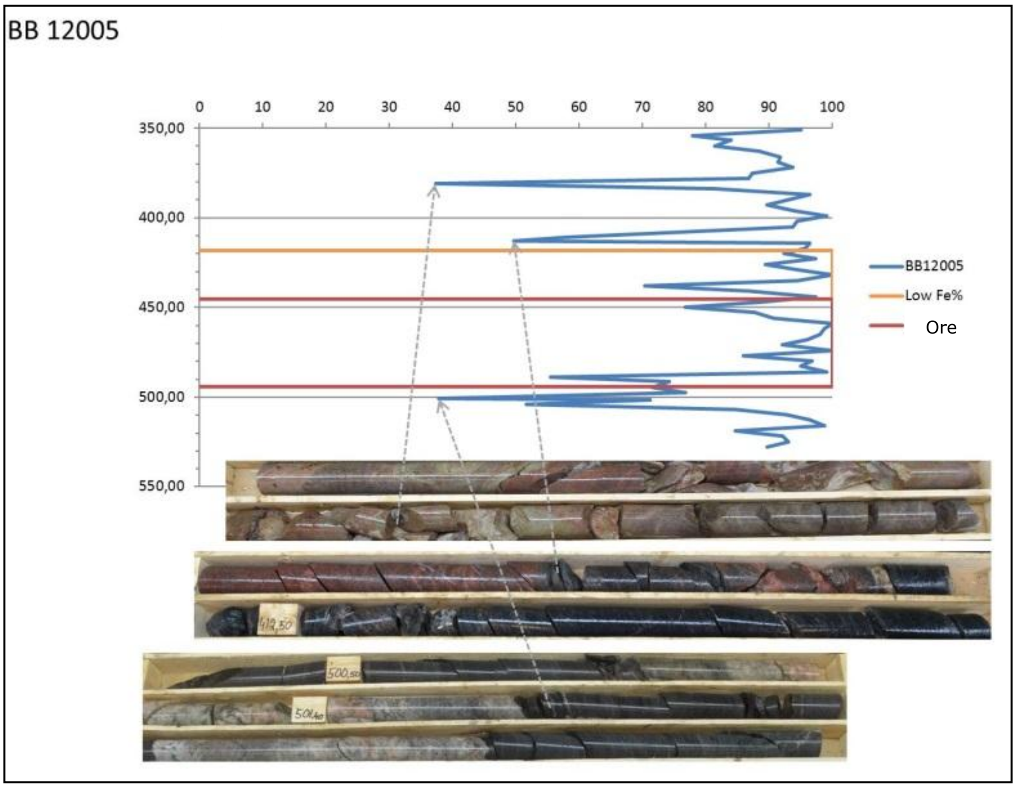


Figure 10. Core log of BB 12005, showing the RQD and core photos in sections with higher fracture frequency in the mineralized zone.

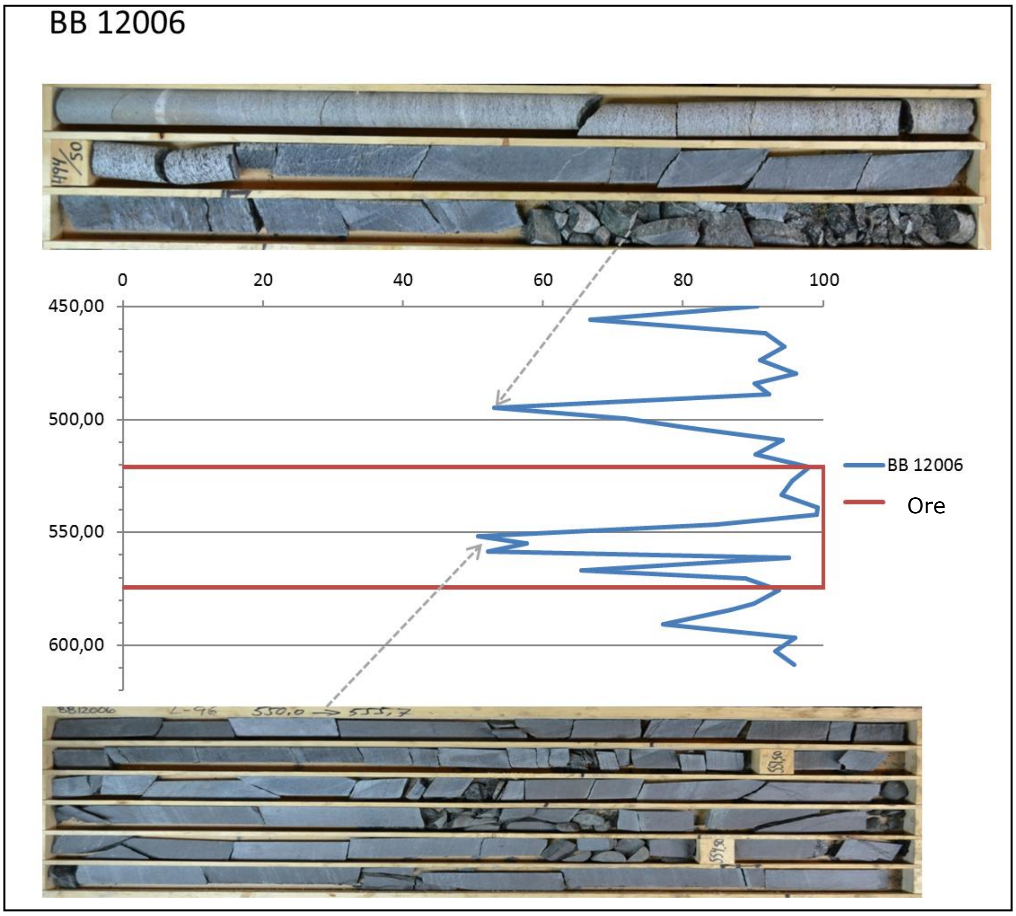


Figure 11. Core log of BB 12006, showing the RQD and core photos in sections with higher fracture frequency in the mineralized zone.

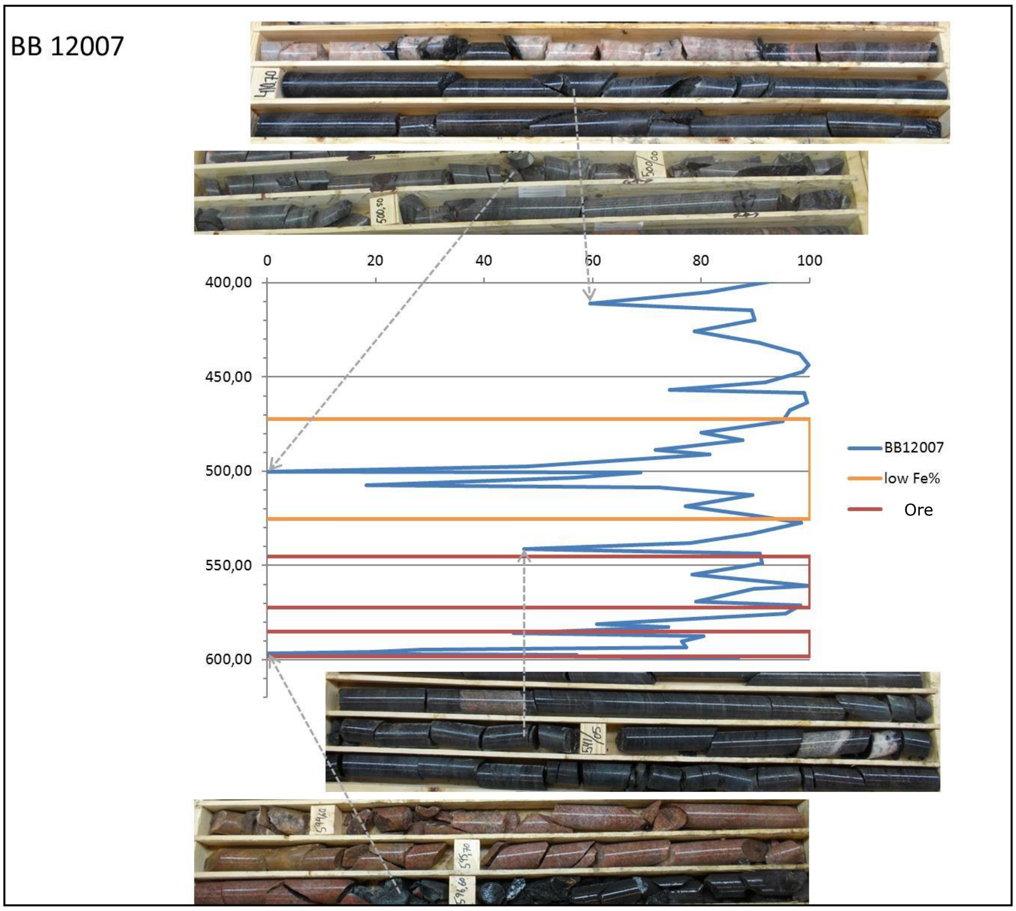


Figure 12. Core log of BB 12007, showing the RQD and core photos in sections with higher fracture frequency in the mineralized zone.

When studying the rock mass quality, no major difference between the different boreholes or between the different parts of the ore (hanging wall, footwall, ore body) have been detected, see Figure 13. Mapping of the drill cores from 2014 indicates the same rock qualities as previous drill core studies. There are differences in RQD within each borehole, but in general the boreholes show high values of RQD values. In general, the cores show a relatively low to moderate joint frequency.

	RQD												Level, ore	
	HW	ORE body										FW		
	HW-5 m	Start ore	1-5 m	5-10 m	10-20 m	20-30 m	30-40 m	40-50 m	50-60 m	60-70 m	Last 5 m	FW- 5 m	Start	Stop
BB12001*	95	91	88	-	-	-	-	-	-	-	-	93	378	383
	95	90	94	99	68-100^	-	-	-	-	-	-	94	426	443
	99	100	92	95	98	84	44-77^	-	-	-	-	70	514	555
BB12002	85	87	92	-	-	-	-	-	-	-	-	100	77	84
	21-81^	41-81^	41-89^	84	91	82	94	-	-	-	-	89	226	263
BB12004*	86	89	92	84	90	96	-	-	-	-	-	58-94^	392	420
	93	94	96	71	49-88^	93	81	-	-	-	-	89	434	471
BB12005	96	94	95	93	90	86	85	98	93	97	65	74	419	495
BB12006	82	67	53	76	87	94	97	-	-	-	-	94	495	532
	99	99	92	51-85^	52-95^	83	-	-	-	-	-	94	544	575
BB12007	88	72	77	48-81^	18-72^	81	-	-	-	-	-	83	489	520
	47-78^	47-91^	91	91	89	-	-	-	-	-	90	89	544	569
BB12008*	'	'	'	'	'	'	'	'	'	'	'	'	579	618
	'	'	'	'	'	'	'	'	'	'	'	'	644	664
BB1/75	71	69	77	100	-	-	-	-	-	-	-	100	593	600
	90	99	98	21-95^	62-100^	-	-	-	-	-	97	98	634	656
	97	98	95	99	94^	99^	100^	-	-	-	81^	12-100^	705	751
BB14001*	85	100	98	100	50-100^	40-100^	50-100^	-	-	-	96	100	348	393
BB14002*	'	95	95	-	-	-	-	-	-	-	95	93	388	401
	94	95	94	95	95	92	-	-	-	-	-	85	409	440
BB14003*	93	85	86	10-90^	10-95^	-	-	-	-	-	88	83	454	479
BB14004*	95	95	10-95^	93	91	-	-	-	-	-	95	95	523	547
BB14005*	90	90	90	-	-	-	-	-	-	-	92	95	382	389
	10-95^	10-90^	90	20-90^	50-90^	74	-	-	-	-	-	10-75^	413	443
BB14006*	90	90	90	90	86	-	-	-	-	-	-	85	500	521
BB14007*	100	100	96	91	-	-	-	-	-	-	92	99	570	585
	96	97	97	97	-	-	-	-	-	-	-	10-97^	609	619
BB14008*	95	95	95	95	10-95^	88	85	90	-	-	-	10-90^	323	371
	10-90^	85	85	93	99	-	-	-	-	-	68-99^	68	390	417
BB14009*	100	100	99	95	-	-	-	-	-	-	88	99	437	453
BB14010*		- no ore in larger pieces, but only parts of it mapped -												
BB14011*	'	98	98	-	-	-	-	-	-	-	98	98	307	319
the only borehole drilled FW->ore->HW	94	100	44-96^	98	95	-	-	-	-	-	10-98^	100	341	368
	100	10-98^	92	99	-	-	-	-	-	-	93	99	484	501
BB14012*	'	'	'	'	'	'	'	'	'	'	'	'		
BB14013*	94	90	90	81	80	80	-	-	-	-	77	80	543	580

*No sampling; values are based on Ramböll's evaluation of possible ore location.
^Value is much lower than adjacent values or some values within the range of values are significantly lower than others, which may be misrepresentative.
'No data

Legend	
	RQD>80
	80>RQD>60
	60>RQD>40
	40>RQD>20
	Interval of values

Values should be weighed in accordance with length of sample and adjacent values (dip of 50 in RQD within 2 m is unlikely?).

Figure 13. RQD for the first 5 m of hanging wall (HW), ore body and first 5 m for the footwall (FW) in the boreholes, where colours show different RQD-levels, see legend.

2.2 Rock material strength testing

The rock material strength is analysed through point load tests on drill cores BB12011, BB12012 and BB12013. The tests are documented as Point Load massive, Point Load parallel, or Point Load Perpendicular. All values are adjusted to Is_{50}.

Figure 14 shows a compilation of point load test results for different rock types. The results show a variation in strength between 2 and 14 MPa, which corresponds to 40–280 MPa as UCS. No UCS tests have been made on the material. The tests were performed by Petroteam (2012) and the highest presented point load value was 14 MPa on these tests. The reason for this maximum load is not given.

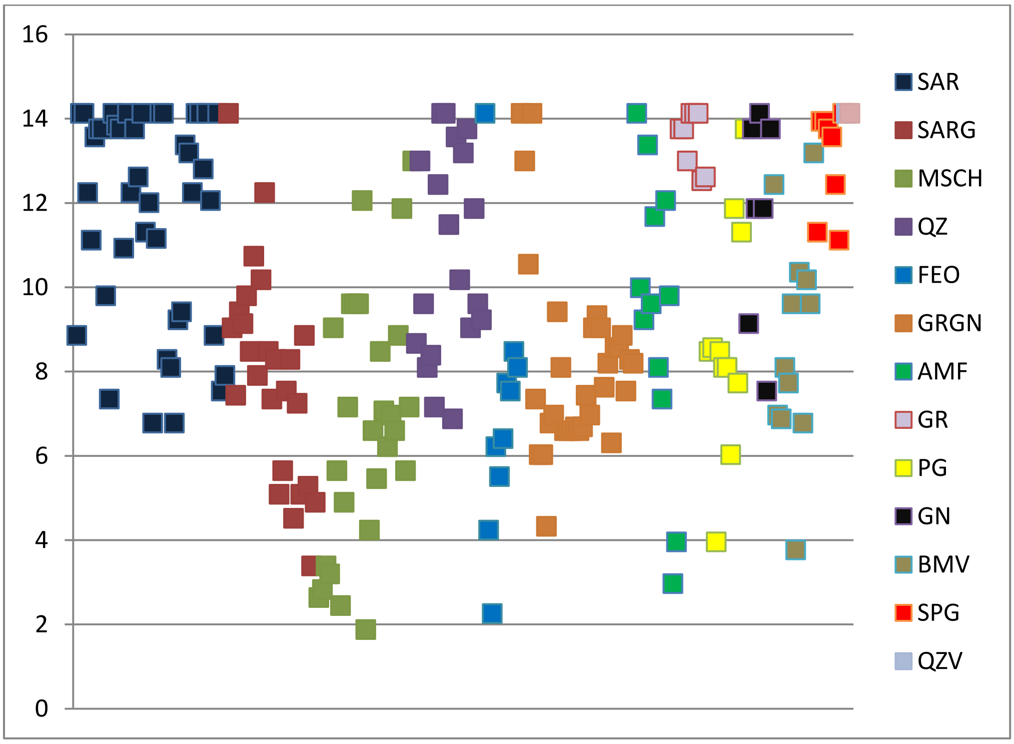


Figure 14. Point load test results for the different rock types.

The point load tests show no significant variation between different rock types in the side rock. However, the rock types SARG and MSCH seem to show a greater amount of test results with lower strength, 2-6 MPa.

The tested iron mineralization showed a significantly lower strength than the side rock with an average of about 6 MPa (USC=120 MPa). Please note that only 8 tests were carried out on ore specimens.

2.3 Hydrogeological tests

A number of water pressure tests have been performed in some of the BB-14 boreholes. The water consumption has been measured between a packer and the bottom of the hole at several depths. The length of the measured sections varies between approximately 20 and 100 m. The measurements are usually performed with a series of pressures 0,5, 1 and 0,5 MPa, each measurement during 5 or 10 minutes.

Table 1 shows some sections with the highest water losses, measured in the BB-14 boreholes. The measured section where the ore body is located is also shown for comparison of depths and water losses. See also Appendix 6, which shows the water consumption tests for borehole BB 14-005, BB 14-008 and BB 14-010 with added calculations of Lugeon values and hydraulic conductivity.

Table 1. Selected sections with water pressure tests in the BB-14 boreholes.

Hole no./ section (m)	Measured section (m)	Total water loss (litre) at 0,5,1,0,5 MPa	Lugeon (l/min, m, MPa)	Geology/description
BB 14-005				
326,4-395,6 Adjacent HW and ore body	69,2	126,0 159,8 113,9	0,73 0,46 0,66	Ore in adjacent HW 352,1-363,5 m Ore body 373,7-394,3 m Small crush zone at 331-332 and 335 m
395,1-464,3 Ore body and FW	69,2	3,6 4,4 2,7	0,02 0,01 0,02	Ore body 395,1-447,6 m FW 447,6-464,3 m
BB 14-008				
233,6-293,8 Distant HW	60,2	142,5 413,0 195,5	0,95 1,37 1,30	Fractured SAR at 251,7-253,8 m Crush zone at 280,8-281,1m and 290,5-291,3 m with core loss, see Figure 15.
293,2-359,5 Distant and adjacent HW	66,3	4,2 5,8 3,0	0,03 0,02 0,02	
356,6-455,8 Ore body and FW	99,2	15,1 41,3 19,6	0,06 0,08 0,08	Ore body 364,5-415,0 m FW 415,0-455,8 m
BB 14-010				
326,9-399,3 Distant HW	66,4	213,6 303,3 194,7	1,18 0,84 1,08	Fractured rock 333-336 m. Crush zones and highly fractured rock 356-374 m, see Figure 16.
503,6-578,4 Distant and adjacent HW	75,3	0,9 11,6 1,0	0,00 0,03 0,01	Ore in adjacent HW 559,9-574,6 m
575,8-651,1 Ore body and FW	75,3	5,3 7,4 1,5	0,03 0,02 0,01	Ore body 576,7-604,4 m FW 604,4-651,1 m

2.3.1 **Lugeon values and hydraulic conductivity**

Generally the water losses are relatively small in sections that compile the ore body and the adjacent rock in the hanging wall and the footwall. The larger water losses are concentrated to the rock mass in the hanging wall above the ore, mostly somewhere between 5 and 300 m depth. However, in aspects of calculated Lugeon values, the water loss is still very low even in sections with higher water consumption due to crush zones and core losses. The Lugeon values are generally below 0, which might be explained in the large distance between the packer and the bottom of the hole. The Lugeon values would probably be locally higher if the measurements could be divided in shorter sections with double packers to separate the conductive zones.

For instance, the maximum Lugeon value in BB 14-008 is around 1-1,4 (l/min, m, MPa) on the measured section that contains the crush zone and core loss. The measured section is 60 m between the packer and the hole bottom. According to core photos, the evaluated core length with affected rock mass is approximately 5 m, see Figure 15. Assuming that most of the water loss can be connected to this core section, the Lugeon would be approximately 15 (l/min, m, MPa) for this specific length. This is still a low to moderate Lugeon value, which indicates that the rock mass is not very conductive even in the fractured parts.

The corresponding maximum Lugeon values in BB 14-010 are approximately 1-1,2 (l/min, m, MPa) on a measured length of 72 m. The affected rock mass with crushed and highly fractured rock has a length of approximately 12 m, see Figure 16. This would give a Lugeon value of 5-7 (l/min, m, MPa).

Calculations of the hydraulic conductivity are shown in Appendix 6. In sections with the higher water losses due to fractured and crushed rock, the hydraulic conductivity is in the range of $1*10^{-7}$ to $3*10^{-7}$ m/s. This can be considered as a quite low to medium permeable rock mass in the weakness zones. The "normal" rock mass, close to the ore body, show a less hydraulic conductivity and is in the range of $1*10^{-9}$ to $2*10^{-8}$ m/s. This rock mass has a lower fracture frequency, often with RQD >80 %.

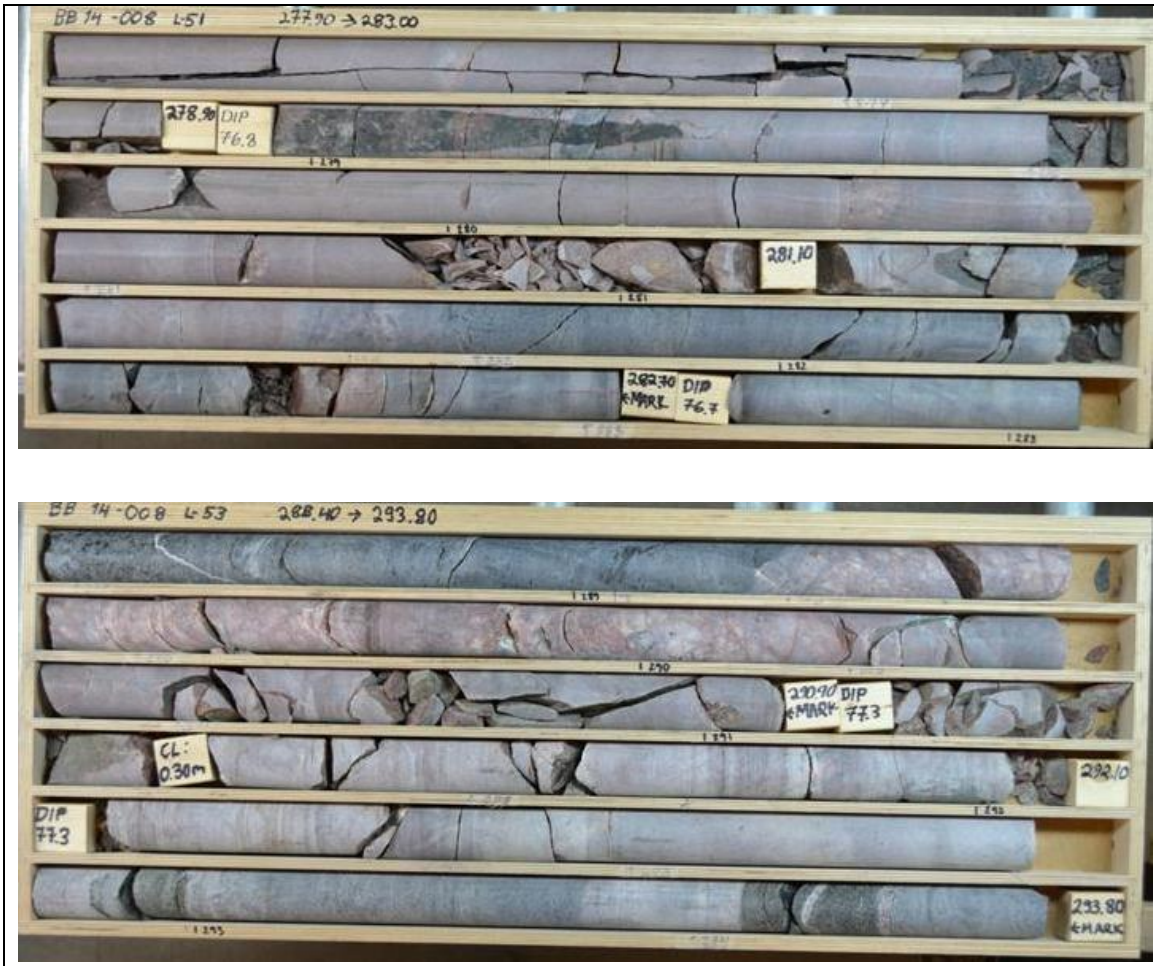


Figure 15. Sections with fractured rock and core loss in BB 14-008.

Figure 16. Sections with fractured and crushed rock in BB 14-010.

2.4 Structures from old mine maps

On historical mine maps (dating back to the 1800's), major structures and faults were marked. A compilation of the structures on these maps have been done. The strikes of the structures are measured directly on the mine maps, based on the coordinate systems given on the maps. In total 213 fractures and 103 faults were found on the maps (se used maps in appendix 1). These observations are compiled from the uppermost levels down to elevation 540 m. An example of an area with structures marked on the map is given in Figure 17.

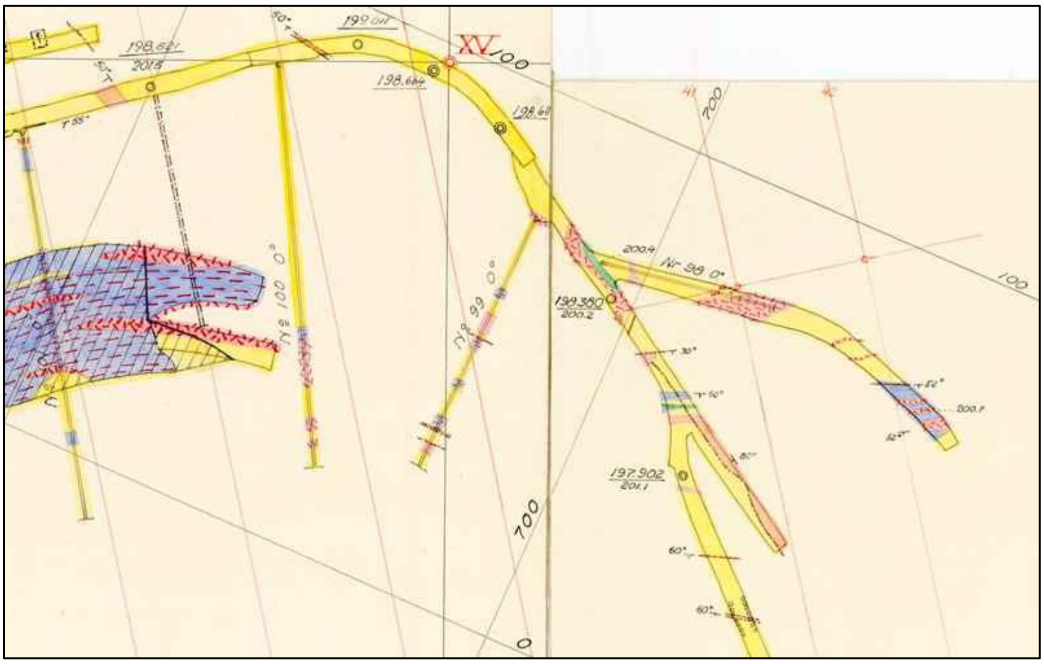


Figure 17. Part of mine map showing structures in drifts and boreholes (in boreholes only indicated, no measures possible).

On the maps the structures are marked as joints or faults, with two different markings. This makes it possible to interpret them either as joints or faults. Most of the structures are marked on drifts or on the side rock, and only a few are found in the ore body. Figure 18 shows one fault, supported with timber, marked in the drift (red circle). The support indicates that this fault is greater (maybe a wider zone of crushed rock) than other faults without support (blue circle) in the same area.

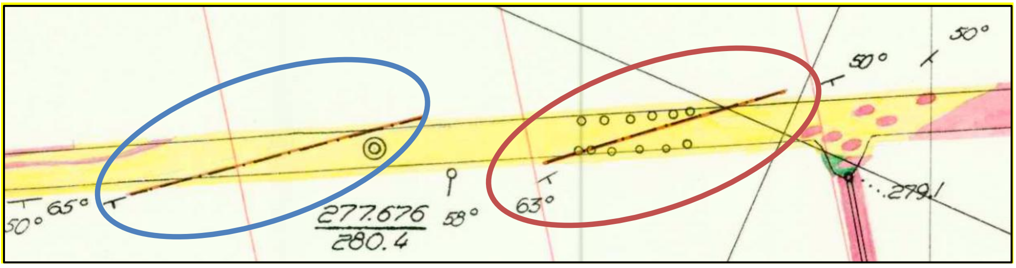


Figure 18. Example of faults, tunnel support and general markings showing the overall strike and dip of the structures.

The structures were compiled and divided into joints and faults independent of elevation, see Figure 19 and Figure 20. The structures show a uniform distribution throughout the mine independent of where they are located, both in horizontal and longitudinal level. The joints are mainly occurring in direction N050-080E. The main directions of the faults are N070E and N110-N160E. No faults can be traced on several sublevels.

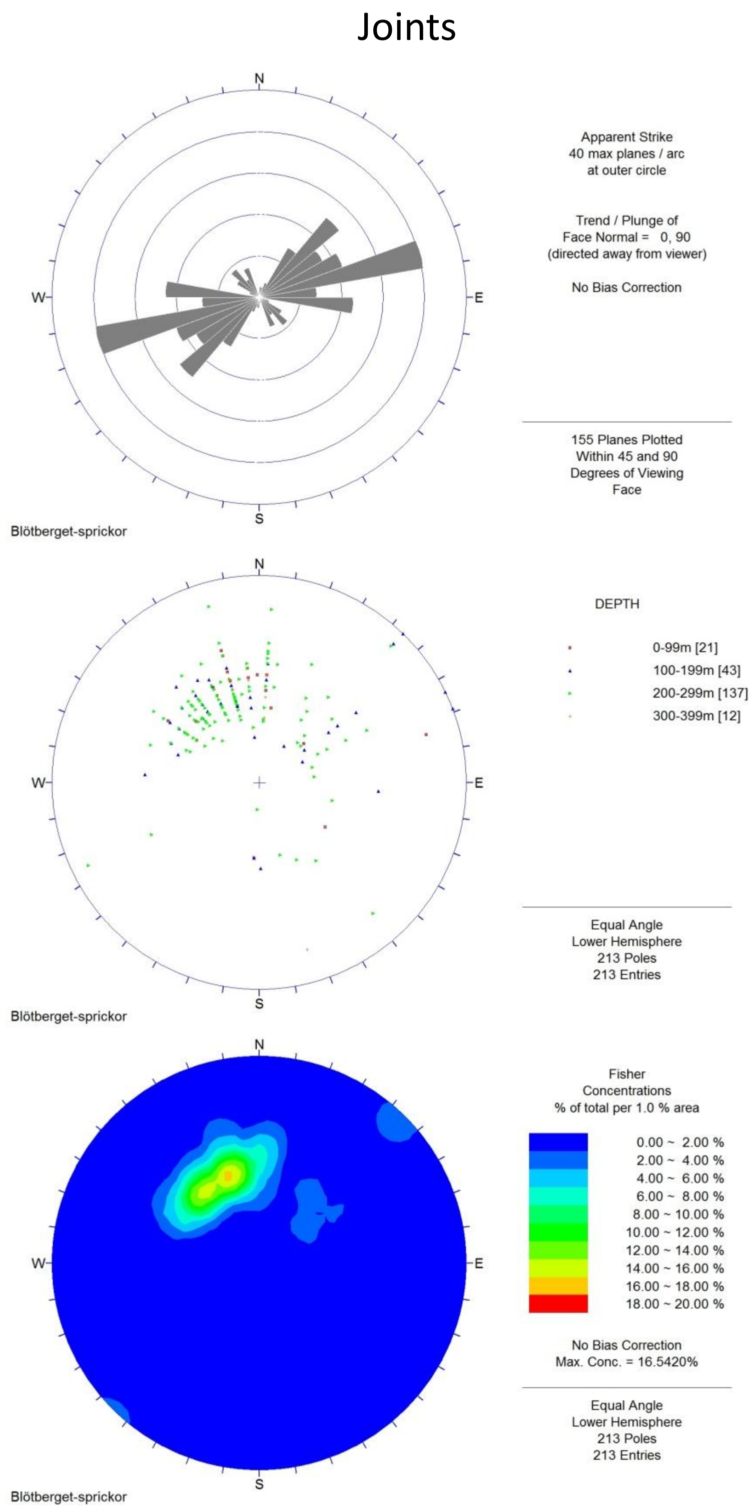


Figure 19. Joint orientation based on old maps of the Blötberget mines.

Faults

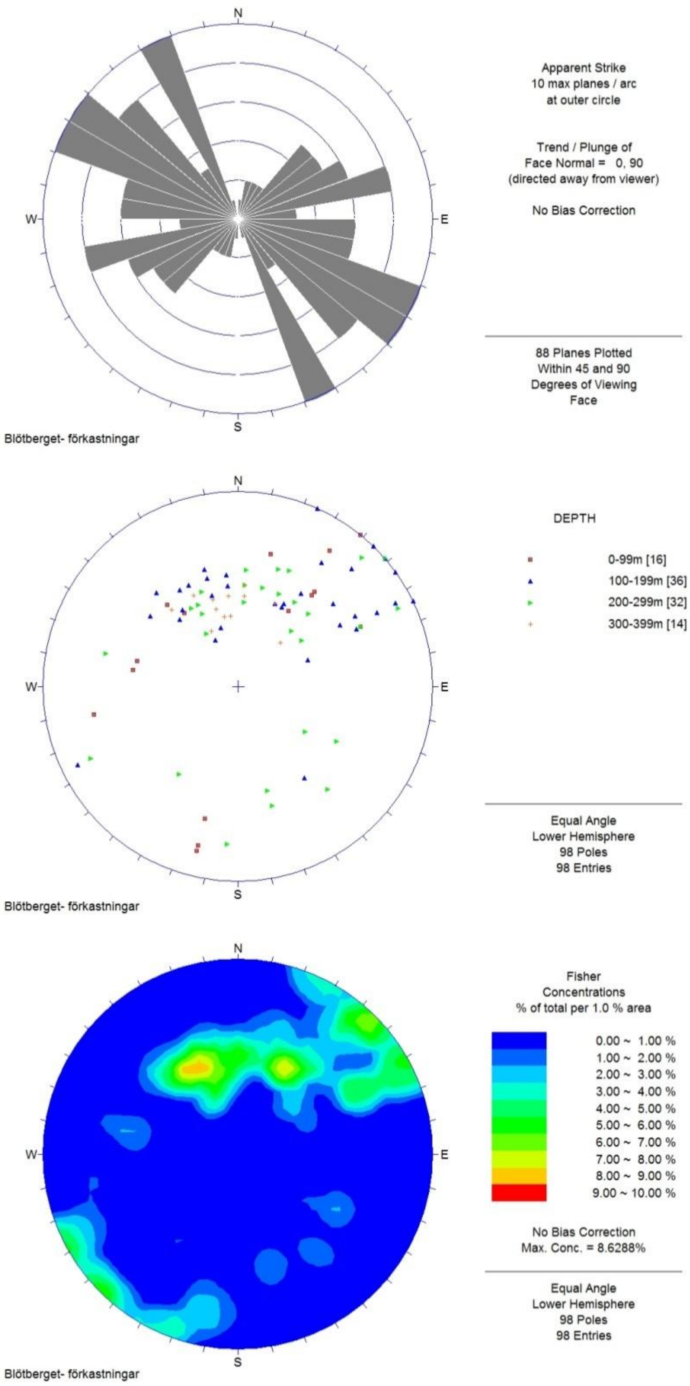


Figure 20. Orientation of fault zones based on old maps of the Blötberget mines.

The dip and strike of the structures are consistent with the results from core mapping conducted in 2012, see Figure 21.

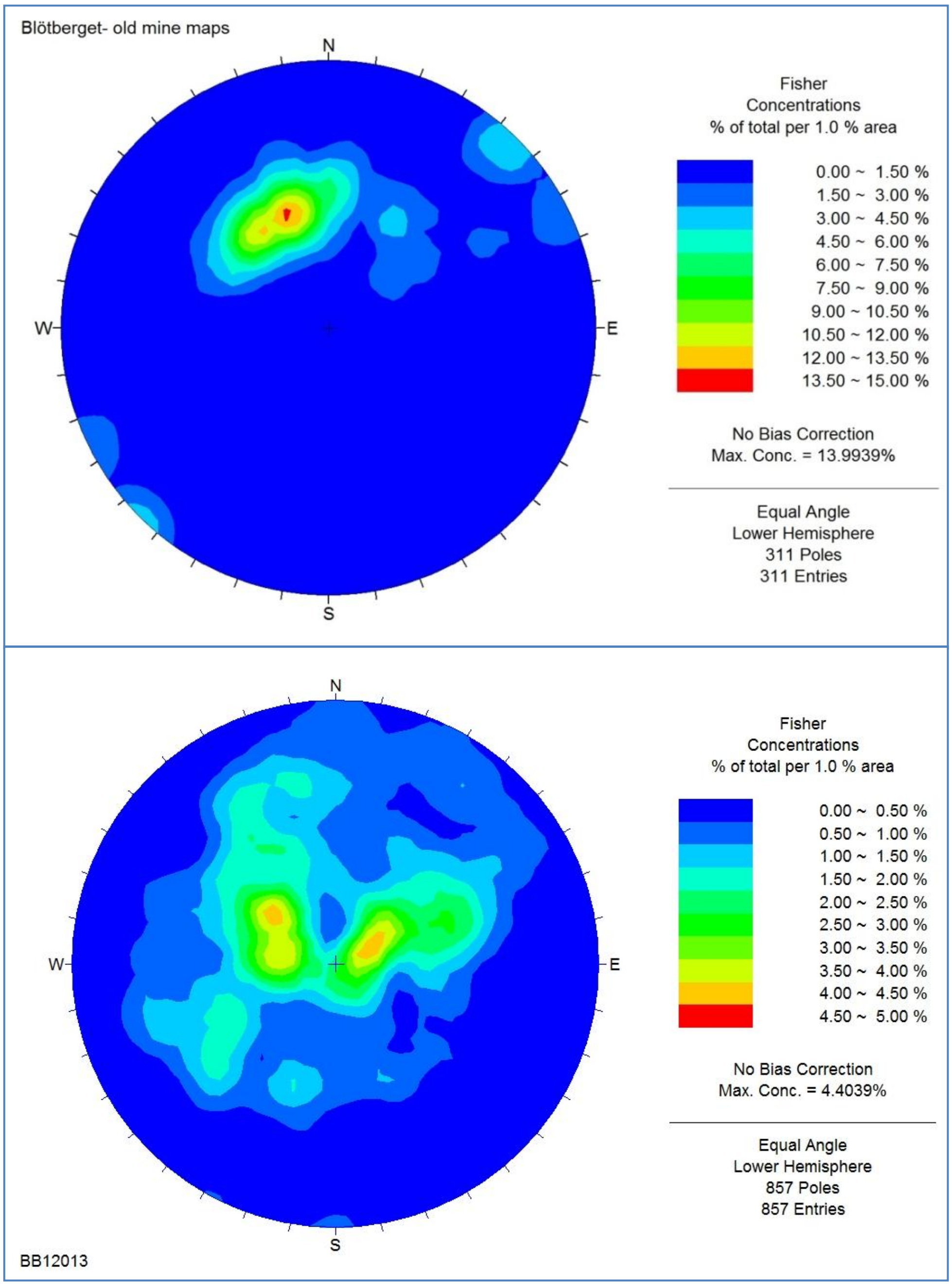


Figure 21. Correlation between faults and joints from old mine maps (upper pictures) and structures from core mapping in 2012 (lower pictures).

In appendix 1, all input data for this evaluation is found together with a compilation of all structures, which are divided into 100 m elevations steps.

The drill cores made in 2014 show similar fracture orientations as the old mine maps and drill cores from 2012, see Figure 22.

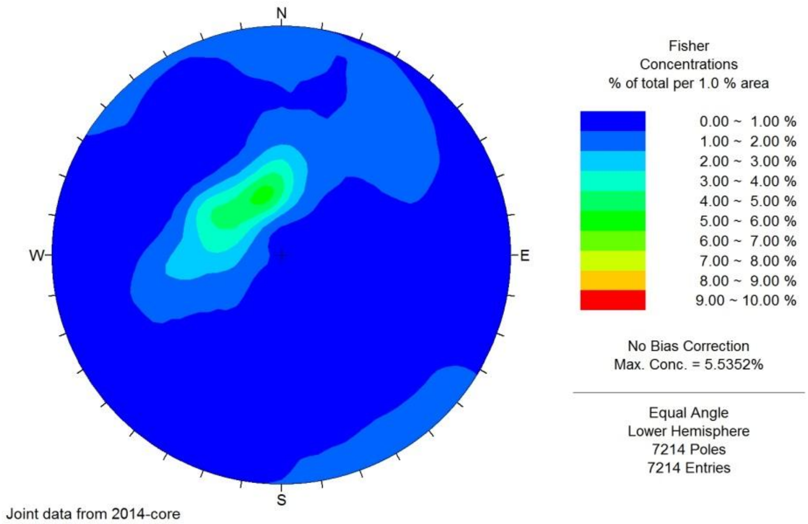


Figure 22. Fractures mapped from the 2014 drill cores.

3. Mathews stability graph method

The Mathews stability method is an empirical tool for predicting open-stope stability. The method is based on a stability graph relating two main factors; the shape factor (S) and the stability number (N), see Figure 23. The principal concept of the stability graph is that the size of an excavation surface can be related to the competence of the rock mass to indicate stability or instability of the open stope. The stability graph can present numerous excavation surfaces with a specified range of stabilities.

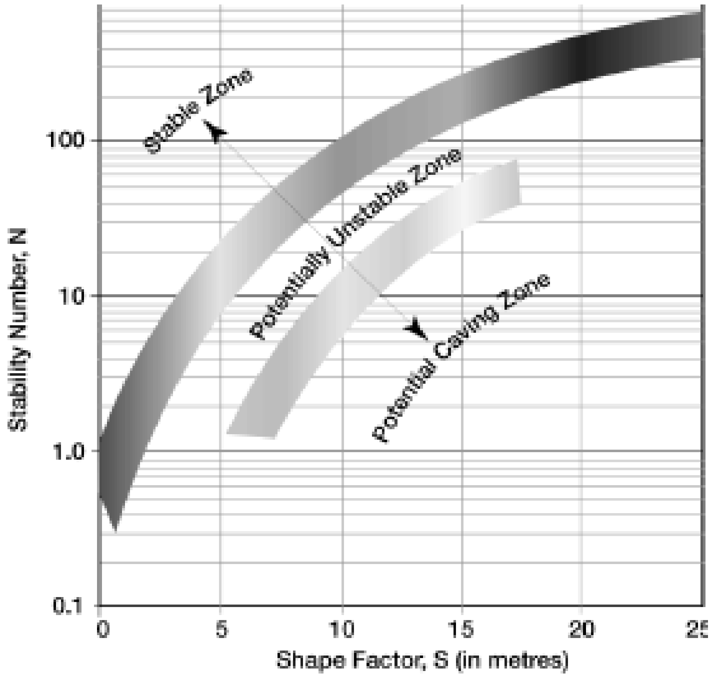


Figure 23. Three stability zones of the original Mathews stability graph.

For a typical rectangular excavation there will be five surfaces; four sidewalls and the crown/back. The stability graph deals with each of these individually, allowing the five surfaces to fall within different predicted stability zones.

The shape factor (S) accounts for the size and shape of the excavation surface and is defined as the area of the stope surface divided by the length of its perimeter.

The stability number represents the capability of the rock mass to withstand a given stress situation. It is defined as:

$$N = Q' \times A \times B \times C$$

Where

- Q′ (Q_{BASE}) is the factor of the rock mass quality
- A is the rock stress factor
- B is the joint orientation adjustment factor
- C is the gravity adjustment factor.

Q′ describes the rock mass quality without considering water parameters. It is defined as:

$$Q' = \frac{RQD}{J_n} \times \frac{J_r}{J_a}$$

The rock stress factor A is a measure of the ratio of intact rock strength to the surrounding stress. As the maximum compressive stress acting parallel to a free stope face approaches the uniaxial strength of the rock, factor A reflects the related instability due to rock yield.

The joint orientation factor B measures the relative orientation of the primary joint set in relation to the excavation surface. Joint planes almost parallel to the free face are most likely to form unstable blocks and slices. Joint planes more or less perpendicular to the free face generally have the least influence on the rock stability.

The gravity adjustment factor C, represents the potential influence of gravity on the stability of the face. The dip of the stope face and the dip of unfavourably oriented structures like joint planes and weakness zones will have an influence on the stability. The failures may occur either by gravity fall, slabbing or sliding due to the orientation of structures.

Figure 24 show the input factors to be evaluated for calculation of the stability number.

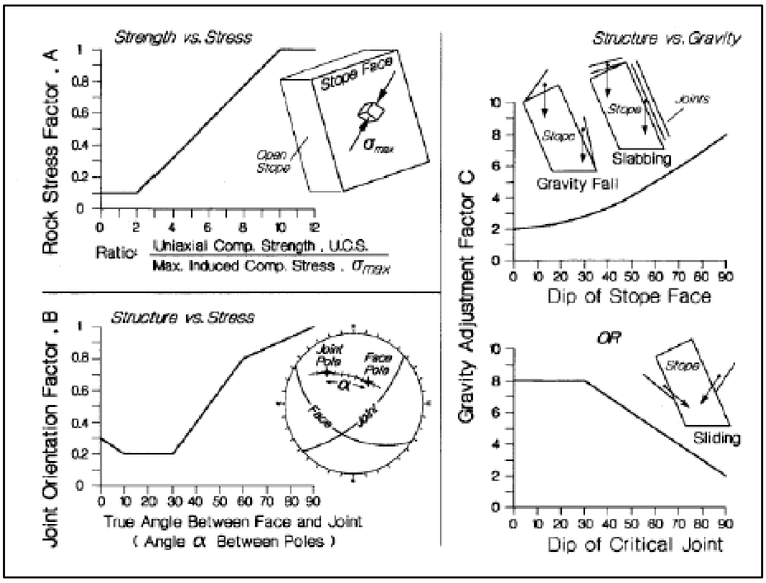


Figure 24. The principal of input factors for analysis of the stability number.

3.1 Applied stability analysis for the Blötberget mine

The Mathews stability chart has been used to evaluate the proposed mining method. The analysis is made with consideration of structural data, possible mining stopes and size of sublevel slices.

In 2011, Vattenfall published an investigation of the stability situation in the Blötberget mines based on the Mathews stability method; see Appendix 3-1 in the PEA study. For the stability estimation done now, some parameters were given the same values as in 2011. All input parameters for this Mathews stability graph are shown in Appendix 2.

Factors J_n (joint number), J_r (joint roughness) and J_a (joint alteration) are the same in this evaluation as in 2011. Factor A, regarding rock stresses, has been estimated from measurements done in the 1950's made in Grängesberg, some 10 km away, but in similar geological setting. These measurements show high horizontal stresses. Factor B, regarding joint orientation, is based on estimations made in 2011. For this study, factor B is assumed to be the same for all walls of the stope (hanging wall, footwall and side walls).

The remaining parameters have been estimated in this study. RQD was set to an average value at level 420 meters between the following boreholes; BB12001, BB12002, BB12004, BB12005, BB12006, BB12007, BB12008 and BB1/75. Factor C is based on the inclination of the ore body in relation to the ground surface for the hanging wall and footwall, which is 42–44°. For the side walls, the inclination is set to 90°.

In order to analyse the stability of Blötberget, several scenarios were studied. For each scenario, the side walls are assumed to be identical. The entire ore area was divided into several stopes, or spaces, of different sizes since the ore width varies, see Figure 25. Hence, the ore width is the width of the mining space.

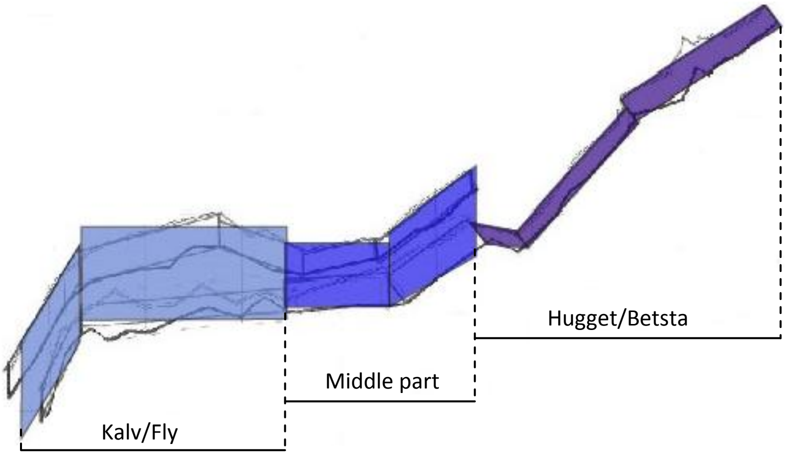


Figure 25. The different mining spaces (each rectangle is one space), where light blue represents the Kalv/Fly mine, darker blue is the middle part and purple is the Hugget/Betsta mine.

In the first scenario, each space in the figure is analysed individually. The height varies between 20 and 26 meters, which is the height of the mining fan. All input parameters of this scenario are shown in Appendix 2a.

In the second scenario, the stope is one sublevel fan of length 10 meters, which is a reasonable length to be able to take out rock. The height is 20-26 meters, which is equivalent to one fan height when mining. This means that the rock above the fan is intact. These parameters are shown in Appendix 2b.

In the third scenario, the stope is one sublevel fan of length 10 meters and height 420 meters. This scenario demonstrates the stability when the rock above is not intact and possible fallout of rock can go all the way to the ground surface, which is 420 meters above. These parameters are shown in Appendix 2c.

3.2 Results of Mathews stability method

The results of the first scenario, regarding the larger mining spaces, are shown in Figure 26. All spaces are stable and rock failure is unlikely.

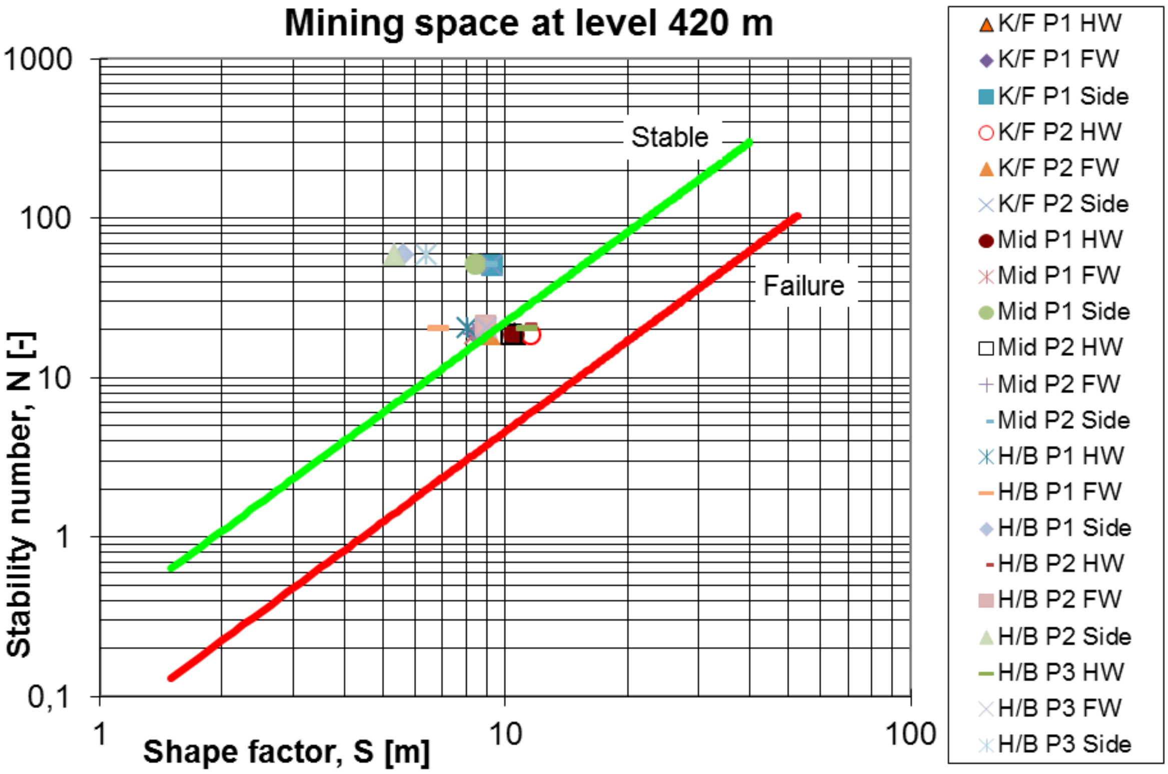


Figure 26. Relationship between stability and shape, which indicates stability or probable failure, for a mining space of height 20-26 meters at level 420 meter below ground level.

The results of the second scenario, with an opening length of 10 m, are shown in Figure 27. All are stable and rock failure is unlikely.

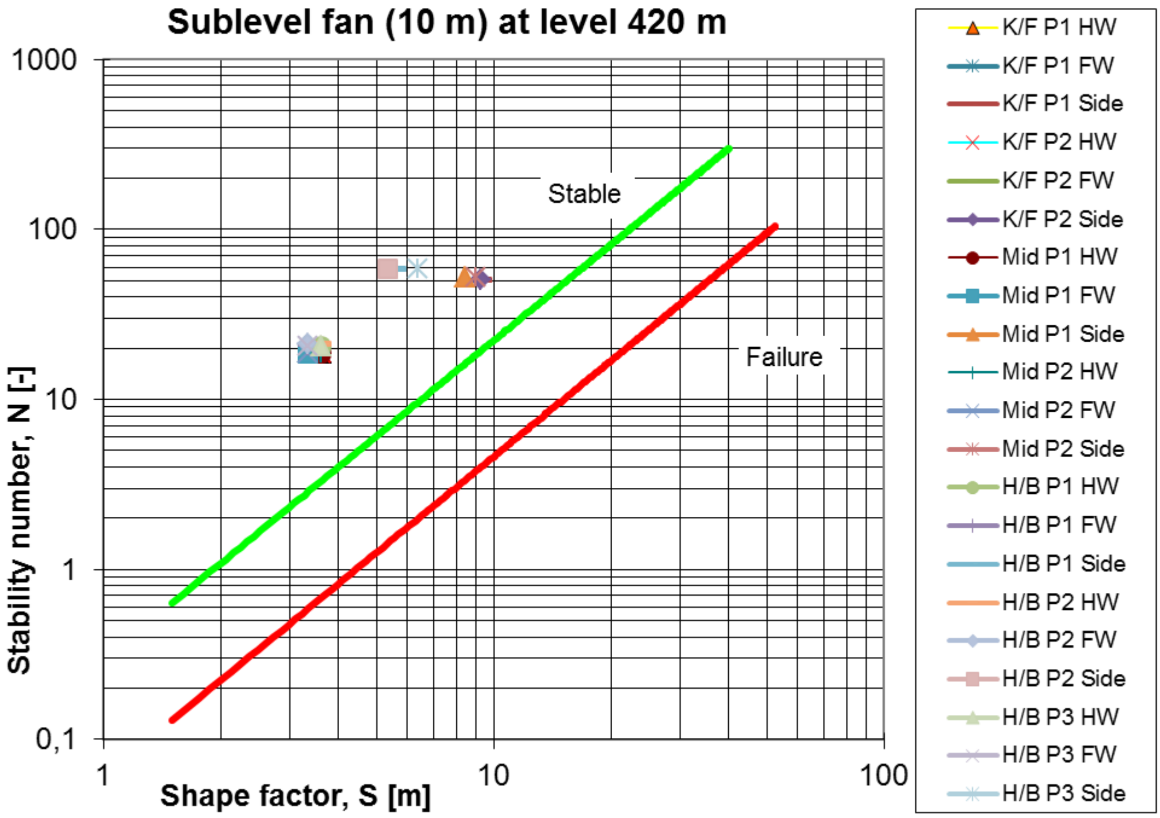


Figure 27. *Relationship between stability and shape, which indicates stability or probable failure, for the first sublevel fan of length 10 meters and height 20-26 meters, at level 420 meter below ground level.*

The results of the third scenario with an opening length of 10 m and height 420 m are shown in Figure 28. Most values are stable. The exceptions are the end walls of the Kalv/Fly mine and the middle part (K/F P1 Side, K/F P2 Side, Mid P1 Side, Mid P2 Side).

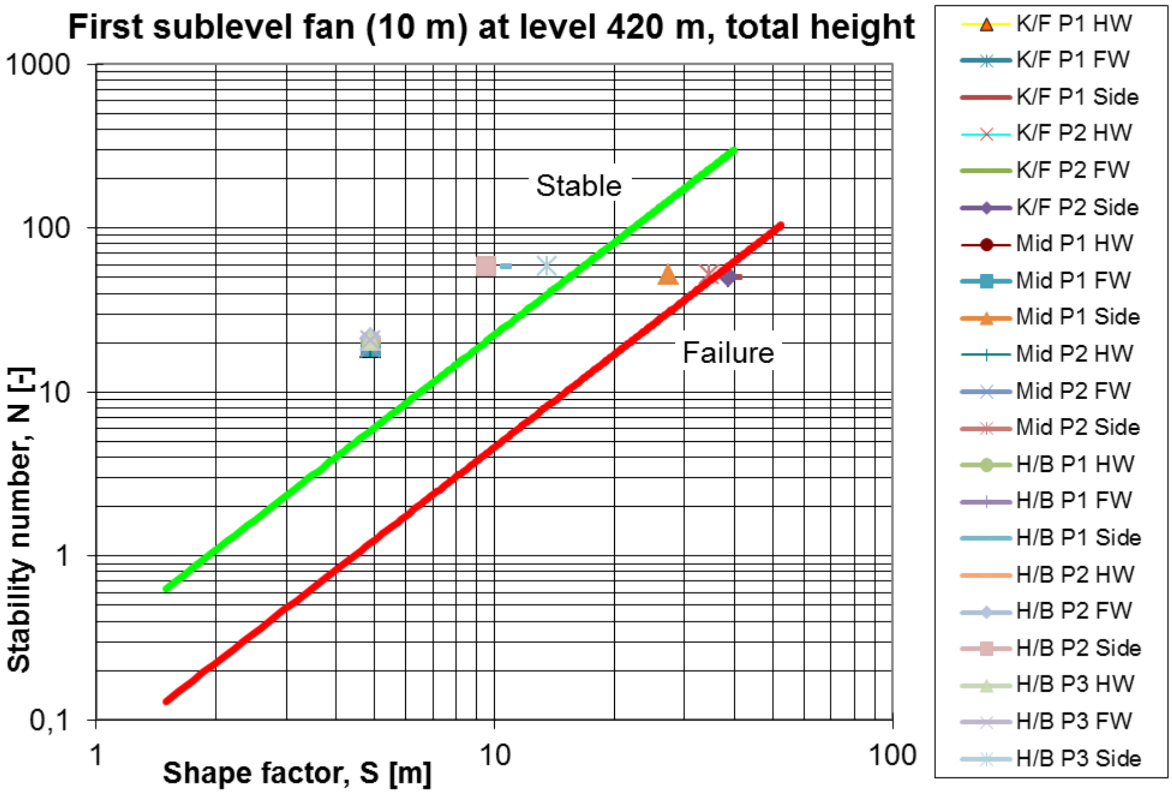


Figure 28. *Relationship between stability and shape, which indicates stability or probable failure, for one sublevel fan of length 10 meters and height 420 meters.*

For the rock cores examined in 2012 and 2014, the boreholes were drilled in the same direction (hanging wall → ore → footwall), leaving some of the joint sets undetected. However, the old mine maps indicate that three or more joint sets are probable, resulting in a higher joint set number, J_n, of 12. Using this value, the rock is more jointed. Thus, the stability graph is changed and more of the rock is within the unstable part of the graph, see Figure 29.

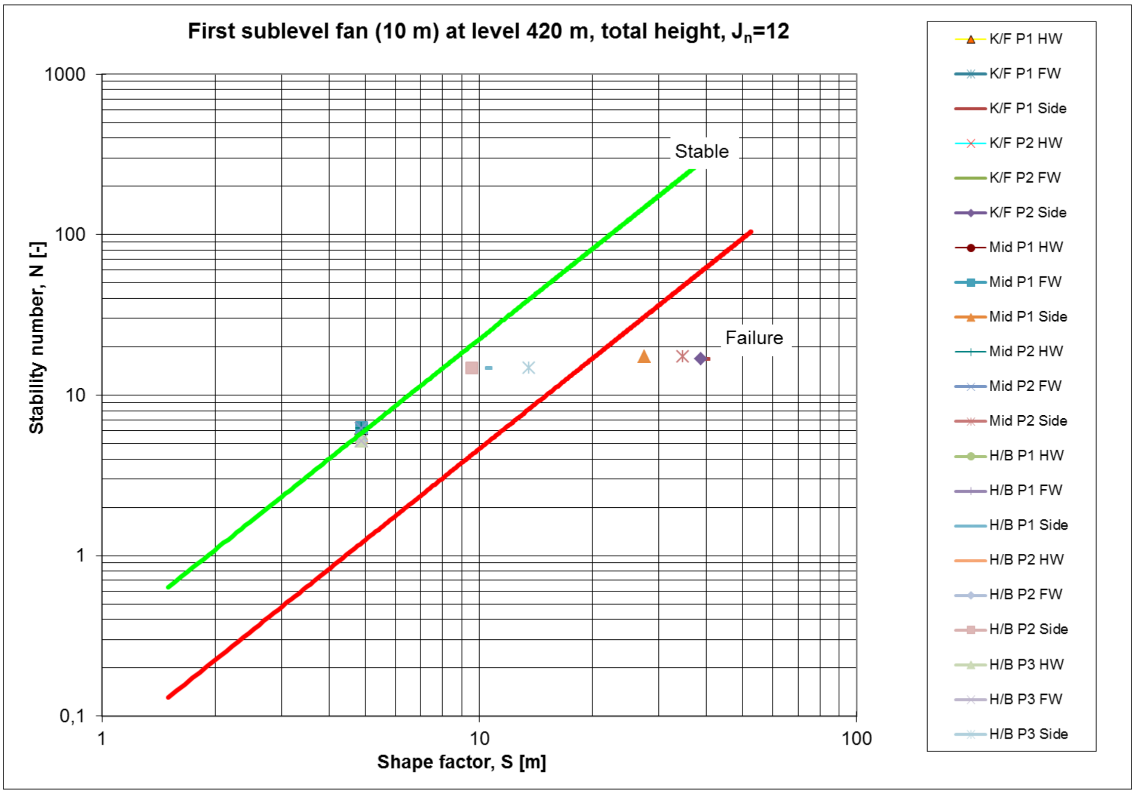


Figure 29. Mathews stability graph for a sublevel fan of 10m length, 420 m height and a joint set number of 12.

4. Sublevel caving stability

In sublevel caving (SLC), the side rock breaks and fills the voids created by mining of the ore in a controlled manner. To create a good SLC, a large enough open space must be created to enable the rock to fall out. Depending on the rock quality, the caving may be induced on the longitudinal side rock contact after just one fan is blasted, or after some 10–20 m of mining operation, see Figure 30. As the opening increases in size, instability is more likely. If the excavation interferes with old caving areas, the probability of a failure is likely to be high.

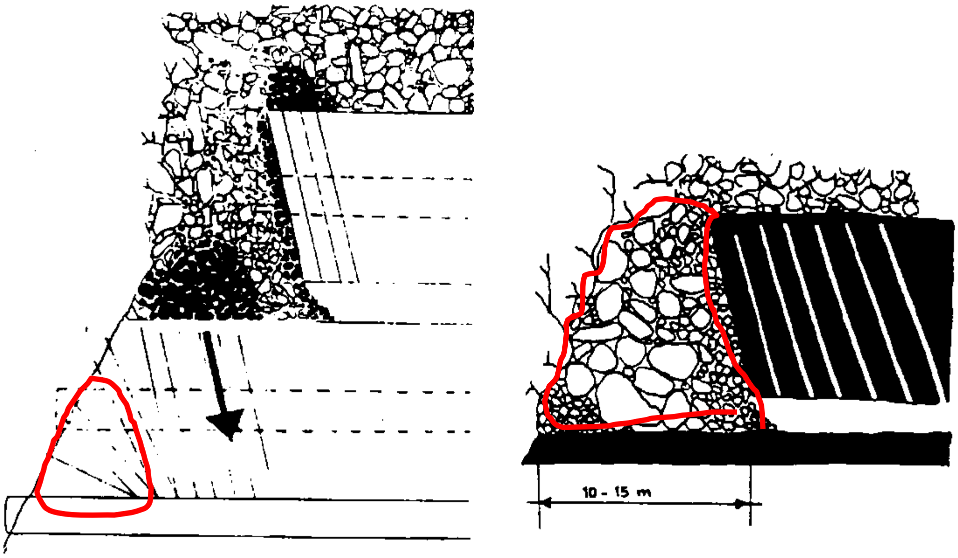


Figure 30. Longitudinal figure of the caving. Either caving starts almost directly after the level is opened up, or some larger fans or more sublevel fans have to be excavated in order to create the wanted excavation room.

The ore bodies in Blötberget are showing variations in geometry. For areas with more narrow ore width, mining will be conducted through one draft; meanwhile in areas with a wider ore body, several drifts are required, see Figure 31.

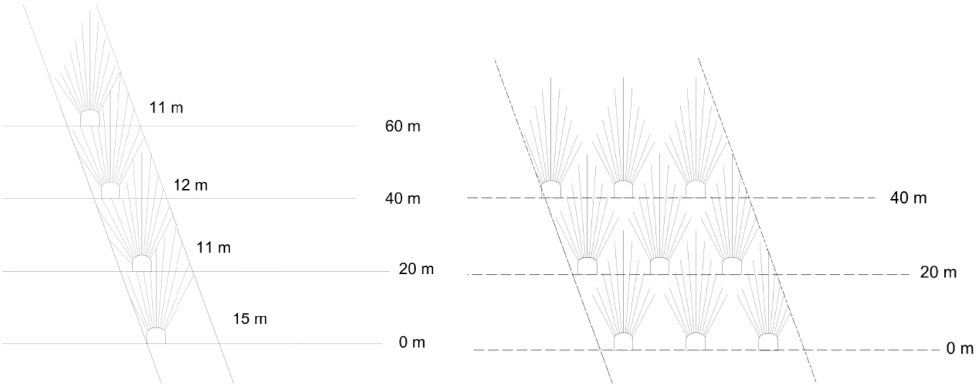


Figure 31. Cross section of one drift in a more narrow ore body (left) and several drifts in a wider ore body (right).

For this report, a general picture of the stability is given. A stability assessment of the adjacent sublevel is not studied, but this may have to be assessed to determine the mining sequence. For such an analysis, more data of the ore quality is required.

4.1 SLC stopes

In order to show the influence on the mined SLC stopes from the fracturing, the geometry of the joints in relation to the geometry of the stope has been analysed. The analysis consists of determining what wedges will form in one stope. In this analysis, only structure data from old mine maps is used, and since this dip and strike of the structures are consistent with the results from core mapping in 2012 and 2014, no update of the figures and analyses has been made. However, this should be updated when these drill cores are entirely mapped in order to get a more detailed overview of the area.

The structures were divided into three sets, see Figure 32. Please note that only the average value for each set was used in the analysis. These are shown in Table 2.

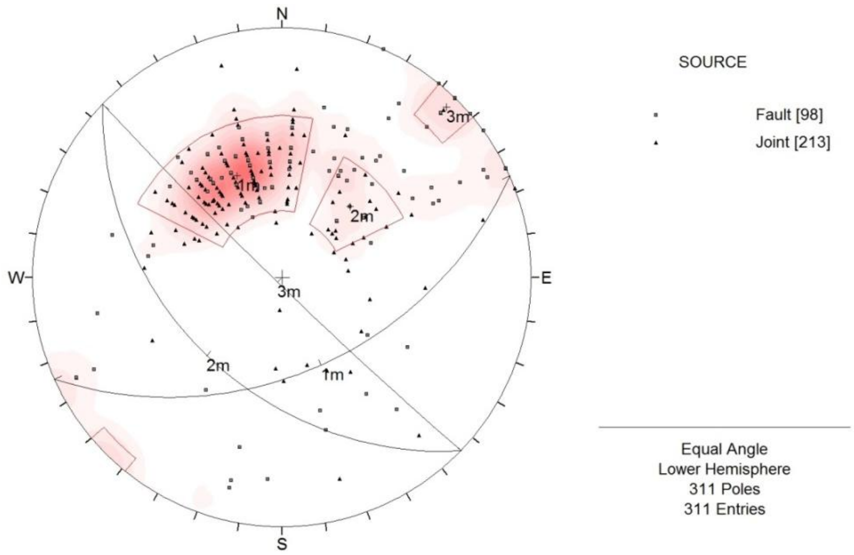


Figure 32. Joint sets of structures from old mine maps.

Table 2. Average values for each joint set.

Set	Dip	Strike
1	48	66
2	43	134
3	87	134

The stope was given a width of 10 meters and height 20 meters, see Figure 33. This is the geometry of one sublevel. The inclination of the ore is approximately 43° on level 420 meters.

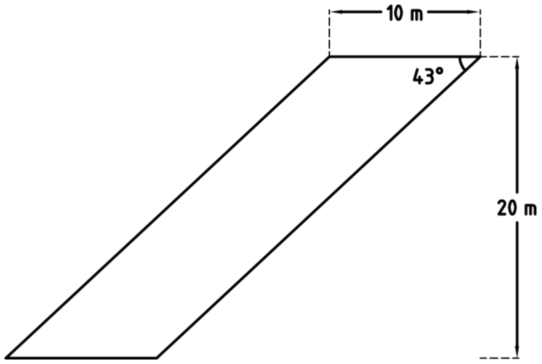


Figure 33. Geometry of the tunnel face when mining, for one sublevel.

The ore in Blötberget has two main directions; N090E and N035E, see Figure 34. Therefore, two analyses have been conducted in Unwedge (software that calculates the wedges that will be formed), one for each ore direction. A comparison of wedges on the hanging wall/foot wall and the end walls is show in Figure 35. Input data for this analysis and all images of the wedges are found in Appendix 3.

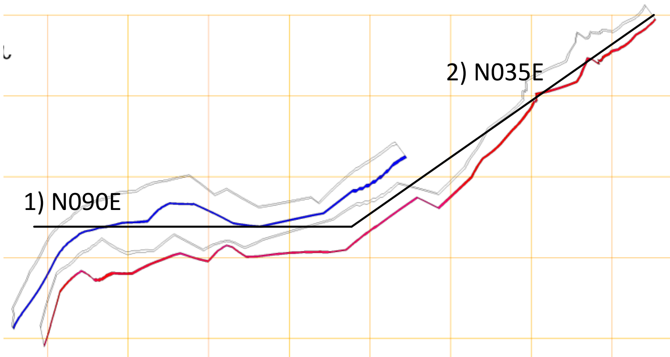


Figure 34. The two ore directions.

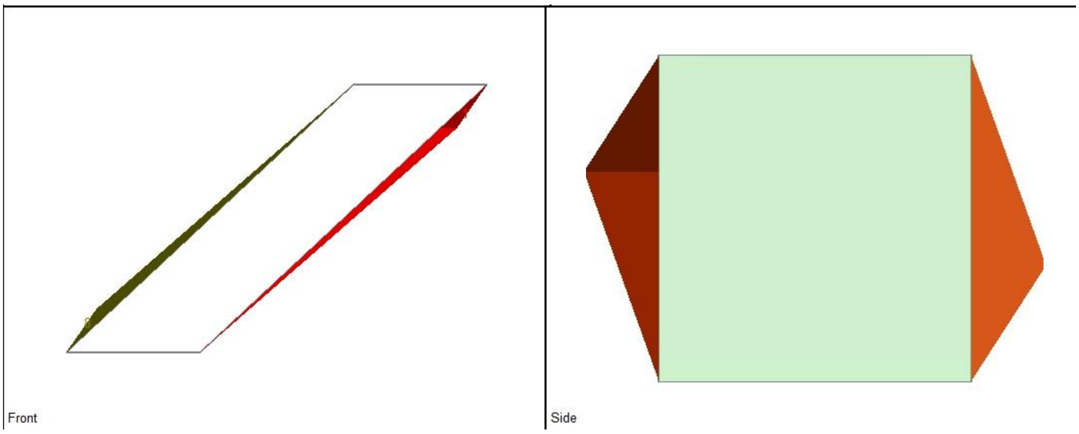


Figure 35. Wedges formed on the hanging wall and footwall (left) compared to the ones on the end walls (right) for direction 1 (N090E).

The wedges that are formed on the hanging wall and footwall are rather small. However, larger wedges are formed on the front and rear walls, which may cause problems in the mined area with large blocks

4.2 Sublevel geometry

The size of the wedges that can form depend on the geometry of the stope. For the sublevel geometry, the same joint sets used in chapter 4.1 are used here, see Figure 32. Analyses are done in Unwedge and all input data is found in appendix 4.

4.2.1 Comparing heights

On the old mine maps, each sublevel was around 5 meters apart. In order to see what wedges will form depending on the height of the stope, an analysis has been done of three alternative heights; 5, 10 and 20 meters, see Figure 36.

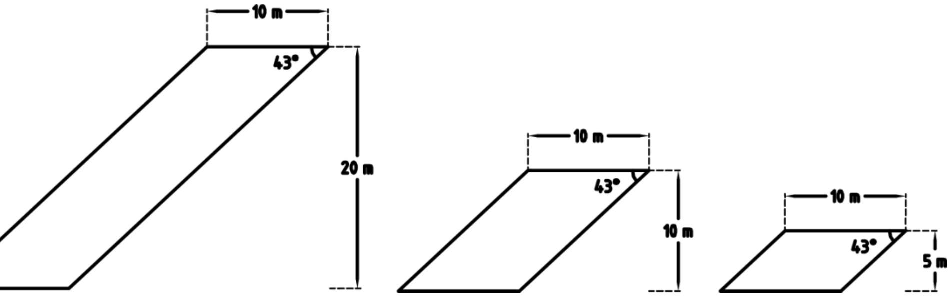


Figure 36. Stope geometry.

The resulting wedges for the two ore directions (see Figure 34) are compared in the diagram in Figure 37. Images of all wedges and the wedge input data for this analysis is found in appendix 4a.

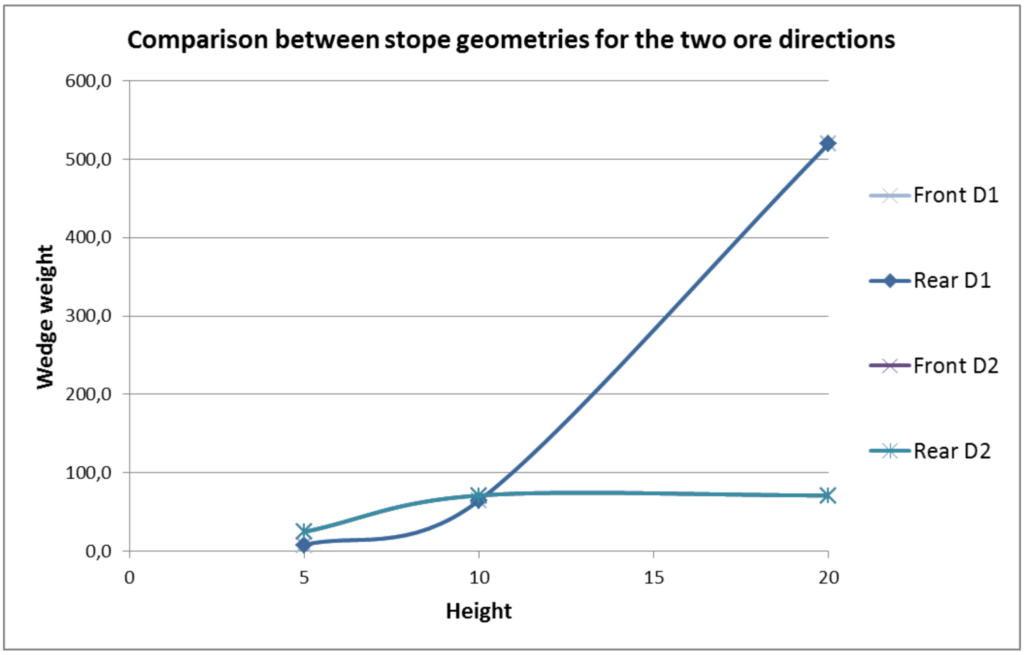


Figure 37. Wedges formed for the three alternative sublevel heights.

4.2.2 Comparing widths

Four different widths are compared, see Figure 38. The height is 20 meters for all cases.

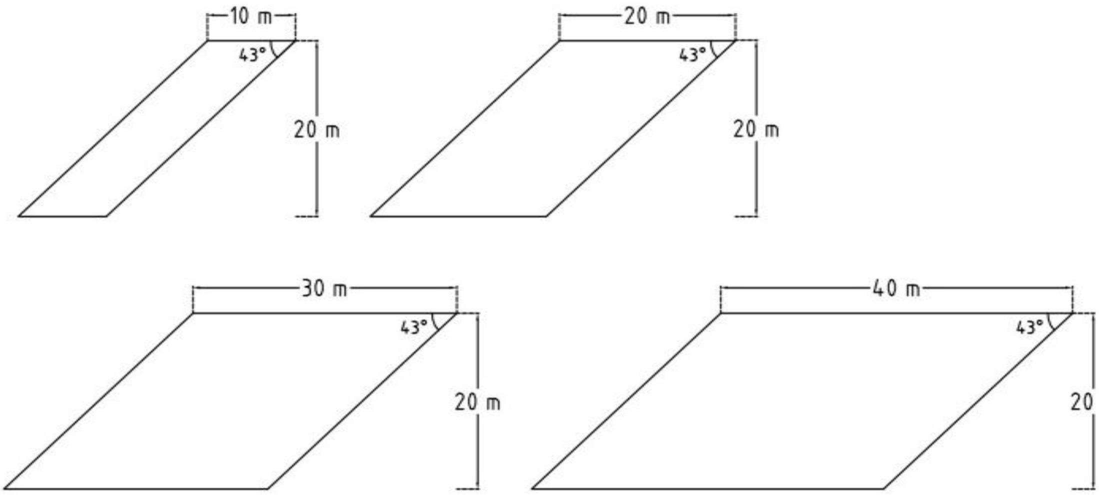


Figure 38. The different widths that are compared.

A comparison between the four cases for the two ore directions is shown in the diagram in Figure 39. Images of all wedges and the wedge input data for this analysis is found in appendix 4b.

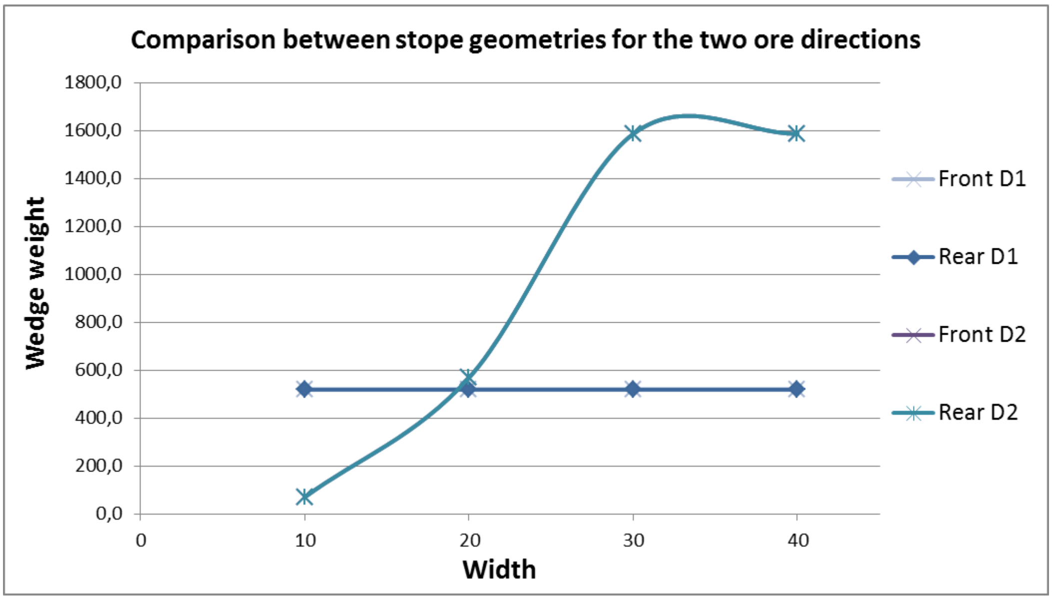


Figure 39. Comparison between widths for the two ore directions.

4.3 Ramp design

The ramp should be designed to follow the main ore direction, 45° off the ore direction or vertical to the ore direction. Therefore, these three options are analysed with regards to joints and the blocks that can be formed. Ore direction 1 (N090E) has been set as the main direction of the ore. Thus, the ramp system will be designed with a main direction parallel to the main ore direction (N090E), 45° off the ore direction (N135E) or perpendicular to the ore direction (N180E).

In this analysis, the same joint sets used in chapter 4.1 are used, see Figure 32.

In order to decide which direction is most optimal, the three options are analysed in Unwedge. The diagram in Figure 40 shows a comparison between the different wedges that are formed. The locations of the possible wedges are displayed to the right (block 5 is not formed in any of these cases and is thus not included in the diagram). The wedges for each direction and the wedge data for this analysis are found in appendix 5.

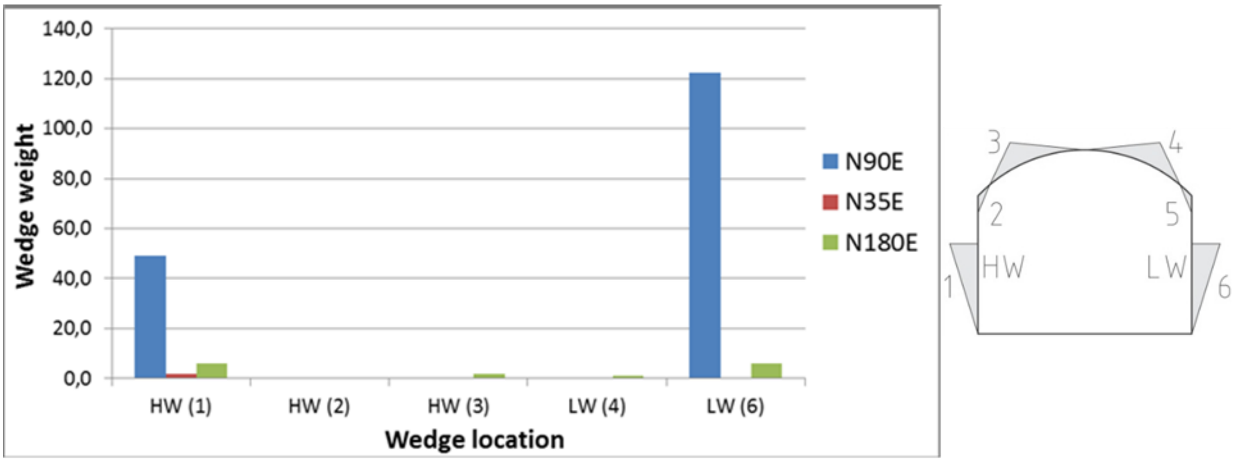


Figure 40. Diagram of the different wedges. Wedge locations are shown to the right.

Designing the ramp in the same direction as the ore will create the largest blocks. Choosing one of the other options would be more optimal regarding block size.

5. Blend of waste rock and ore

When mining, some of the side rock/waste rock will break and fall out together with the ore. This will result in a blend of ore and waste rock. This blend was analysed in Unwedge for a stope of size H20 x W10 x L4 meters. The result is shown in Table 3.

Table 3. Blend of waste rock and ore when mining. The percentage shows the amount of ore in the ore/waste rock blend.

	Volume [m³]	**Unit weight [t/m³]**	**Weight [t]**
Ore	800	5.2	4160
Waste rock			
Dir N090E	2.7	2.6	7
Dir N135E	19.6	2.6	51
Blend		Percentage	
Dir N090E		99.7%	
Dir N135E		97.6%	

The amount of waste rock is small for a stope of this size.

6. Conclusion

No major differences regarding RQD are found between the different boreholes or between the different parts of the ore. There are differences in RQD within each borehole, but in general the boreholes show high values of RQD values, mainly 80–100.

Generally the water losses are relatively small in sections that compile the ore body and the adjacent rock in the hanging wall and the footwall.

From the old mine maps, the dominant strike of the fractures is almost parallel to the mineralization, east-northeast direction (N070E), and the dominant strike of the faults is in a southeast direction (N110-160E). This dip and strike of the structures are consistent with the results from core mapping in 2012 and 2014.

The Mathews stability analysis shows that the rock is mainly in the stable zone of the graph. However, since the boreholes were drilled in the same direction, some of the joint sets may be undetected. Hence, one analysis was made with the assumption that the rock has more fractures, which resulted in more unstable rock. Sublevel caving is an option for Blötberget. Parallel drifts can be mined independently of each other. However, front and rear wedges may cause problems.

The sublevel geometry is almost insignificant for the ore in direction 2 (N135E), since the block size does not change immensely when changing neither the height nor width. For direction 1 (N090E) however, the block size changes exponentially and the geometry of the sublevel has importance.

If the ramp design follows the main ore direction (N090E), the largest wedges will be created, out of the three optional directions. Designing the ramp in one of the other directions (N135E or N180E) generates much smaller wedge sizes.

For a stope of size H20 x W10 x L4 meters, the amount of waste rock blended with the ore will be small (<3%).

7. Recommendation

For this report, all fractures have been analysed together and no distinction between hanging wall, ore and footwall has been done. The characteristics for the ore should be further evaluated, so that differences between different ore parts can be studied. That has been considered a question for the detailed mine planning and not been studied in detail at this stage.

Even though the material available so far gives a quite consistent picture, we recommend, in order to create a more holistic picture, that full mapping and orientation of all cores should be conducted. This report should then be updated with the new data. The RQD analysis has been updated with data from all mapping that has been done so far, but the Mathews stability graphs and the Unwedge analyses are only based on average data from boreholes mapped in 2012. Data from mapping of drill cores in 2014 have not been taken into consideration in these analyses, since they show the same values in general. Both the 2012 and 2014 drill cores are recommended to be entirely mapped and then all analyses should be updated, if a more detailed evaluation should be produced.

Two boreholes in 2012 years program were stopped in fractured rock in the western fringe of Blötberget. These two holes should be studied further in order to show whether these are drilled in true crushed zones or if the more fractured rock has more correlation to the boudinage structure of the mineralization.

Stress analyse Glaningen

NORDIC IRON ORE AB

Technical report

Stockholm

Technical report

Stress analyse Glaningen

Datum
Uppdragsnummer 1320011785
Utgåva/Status

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Summary
Mining in Blötberget will induce stress re-distributions and deformations in the rock mass adjacent to the mine. A numerical analysis has been performed to assess the extent of these changes in the rock mass. It has been found that there is a potential for increased conductivity in the rock mass between the lake and the mine. The re-distribution of stresses and the displacements might influence the existing fracture system in the rock mass and thereby increase the conductivity. The changes in the rock mass will probably have a low impact on the conductivity as long as the mining activities occur above the 450 m level. It is recommended that fracture orientations in the upper part of the rock mass are further evaluated.

Innehållsförteckning

RAMBØLL

Technical report

1. Introduction

The mining at Blötberget will change the rock stress situation in a fairly large volume around the ore bodies. Lake Glaningen is located at a horizontal distance of approximately 500 m from the ore bodies. One concern with the deepening of the mine is that the stress re-distributions might influence the flow paths from Glaningen to the mine. If the flow paths are widened or new ones created the lake might be drained into the mine. This will cause the need for increased pumping capacity in the mine as well as a unwanted impact on the environment in the area.

2. Objective

The objective with the work presented in this report is to evaluate the magnitude of the stress re-distribution, the influence area of the stress re-distribution, deformations and discuss the possible effects on the rock mass and the flow paths between Glaningen and the mine.

3. Scope of work

The stresses where modelled using the three dimensional code FLAC3D (ITASCA). The steps involved in the modelling and analyses of the results are as below listed.

1. Ore and lake geometries retrieved from NIO. For the lake only the shore line is known and the depth is assessed to approximately 30 m.
2. 3D modelling of the rock mass and ore bodies.
3. Analyse and interpretation of results.

4. Geotechnical setting and input data

4.1 Fractures and fracture sets

A compilation of the fracturing of the rock mass in Blötberget is presented in (Anläggningsprojektet Ludvika gruvor, Blötberget. Per-Erik Söder, Ramböll, 2014). The report indicates a good rock mass with typical RQD values in the range of 80-100. Old mine maps have in the report been used in the report to compile the orientation of fractures and faults. Three sets were found with the orientations presented in Table 1. Two sets are moderately dipping and striking NE and SE respectively. An additional sub-vertical set strikes to the SE as well. Set 1 includes the vast majority of the mapped fractures. It should be noted that these fractures are mapped as some depth. It is at this time not known if these orientations are representative close to the surface.

Table 1. Joint sets derived from old mine maps (Söder, 2014).

Set #	**Dip**	**Strike**
1	48	66
2	43	134
3	87	134

4.2 In Situ stress

The in situ stress situation in the area is poorly known. The most recent stress measurements were made in the 1950s. For this work the same in situ stress field is used as the one in the report "Bilaga 4-3 Rasvinkel Blötberget, Vattenfall 2011." The stress field adopted is listed below.

σ_v= 0.027z Mpa
σ_h= 2 + 0.025z MPa
σ_H= 5 + 0.040z MPa

The orientation of the major principal stress is set perpendicular to the ore body.

The above in situ stress values gives a reasonable agreement with the old 1950s reports on the stress measurements. The stress values must be treated as preliminary since the comparison with the old report is made at a high level.

5. Numerical model

The numerical model was made as a block with sides approximately 10 km long. The depth of the block is approximately 3 km. The large block will remove any boundary effects on the stress situation around the ore body. The ground surface is modelled according to the 3D map of the area. The area is however quite flat and the model could just as well have used a flat top surface. The block/element size close to the ore body is approximately 10 m and at the model boundaries approximately 100 m.

An elastic analyse has been performed for this work. Elasto-plastic modelling was not performed since the quality of the rock mass should be determined in more detail before doing that type of analyse. It is however assessed that the elastic results will give detailed information enough at this stage of the project.

In the model the ore body is treated as a void. Hence there is no material in the ore body which means that the effect of the mining becomes somewhat exaggerated. Three different mining depths have been modelled. The first one corresponds to a mining depth of 250 m which is the approximate depth of the abandoned mine. The next model depth is 350 m and the final depth 450 m. In addition to this calculations after excavation of the whole resource down to approximately 900 m depth.

Young's modulus in the model is set to 40 GPa as an average for the rock mass.

The model uses a co-ordinate system that is slightly rotated to true North. The rotation is approximately 20 degrees East. The co-ordinate axes are visible in most of the plots.

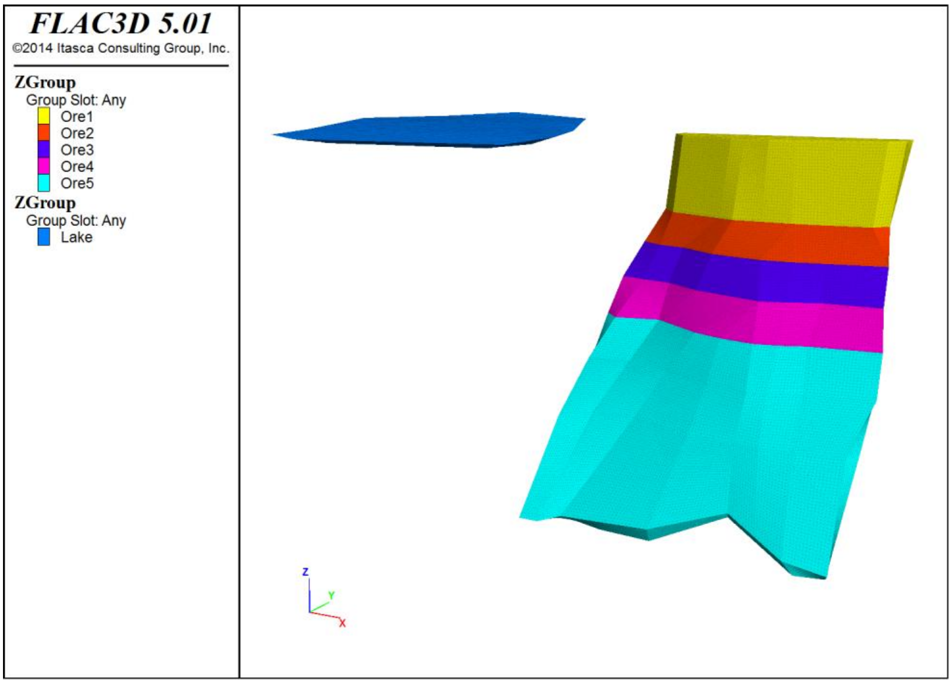

Figure 1. Overview of the numerical model with the lake and the ore body. Ore1 is the current mining depth to 250 m. Ore2 and Ore3 represents mining to 350 m and 450 m respectively.

6. Interpretation of numerical results

A number of cut planes have been produced for this report to make the results easier to understand. When viewing the results one must remember that in the plots compressive stresses are negative. This means that the major stress (most compressive stress) is denoted as minor stress in the plot legends.

To interpret the results the focus is put on the minor principal stress and displacements. The major principal stress is always compressive in the results and hence tightens fractures perpendicular to its orientation. The minor principal stress will though induce tensile stresses in the rock mass. These stresses might open up fractures or create new flow paths in the rock mass between the lake and the mine.

An overview of the minor principal stress changes is illustrated by a cut plane located between the lake and the ore body. The plane is placed in the area where the tensile stresses are largest. The plane is dipping 70 degrees to better follow the ore geometry. The geometry of the plane in relation to the lake and the ore body is presented in Figure 2.

The current stress situation in the area is presented in Figure 3 and the changes due to mining down to 350 m and 450 m are presented in Figure 4 and Figure 5 respectively. To easier follow the changes in the minor principal stress an iso-surface representing 0 MPa is plotted as a black line. This iso-surface represents where the minor principal stress situation in the rock mass changes from compressive to tensile. The figures indicates that the rock mass is subjected to tensile stresses closer and closer to the lake as the mining progresses. This is further illustrated in a horizontal cut plane located at 20 m below the ground surface. When the whole resource is mined tensile stresses will develop beneath the entire lake (Figure 6). The current situation is illustrated in Figure 7 and after mining to 450 m depth in Figure 8. The distance from the lake to the tensile regions is currently approximately 350 m and this distance decreases to approximately 200 m as the mine reaches 450 m depth.

To follow the change in the minor principal stress an iso-surface representing the border between compressive and tensile stresses are plotted as a black line. If comparing Figure 7 and Figure 8 representing 250 and 450 m mining depth it is found that the tensile region now is approximately 350 m from the lake and closes to approximately 200 m when the mine is deeper.

The major principal stress at the horizontal plane at 20 m depth is illustrated in Figure 9. High stresses develops close to the mined out area but the stresses close to the lake varies very little for the different mining stages.

Stress tensor orientations for the horizontal plane at 20 m depth are presented in Figure 10. The minor principal stresses (denoted maximum in the figure) are tensile inside the black line and oriented approximately NE-SW. This could tend to open fractures trending NW-SE (set 2 and 3). Fractures oriented in this direction are not likely to intersect both the lake and the mined out area. The major principal stress is oriented approximately N-S and should tighten fractures oriented W-E that could be possible direct flow paths between the lake and the mine.

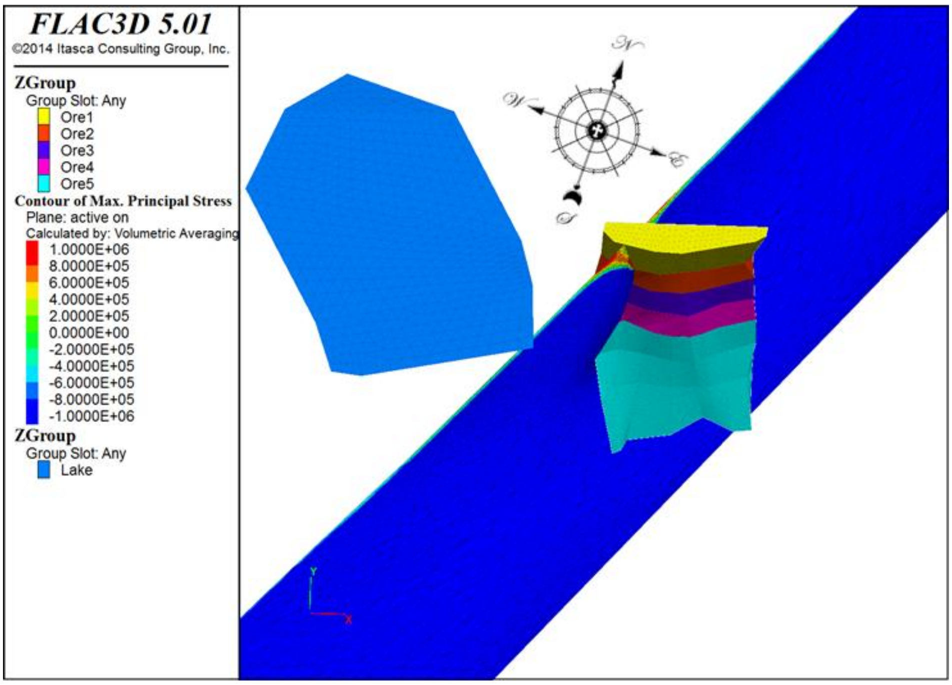

Figure 2. Orientation of plane dipping 70 degrees used for analysing the stresses between the mine and the lake. The figure also includes approximate geographical North.

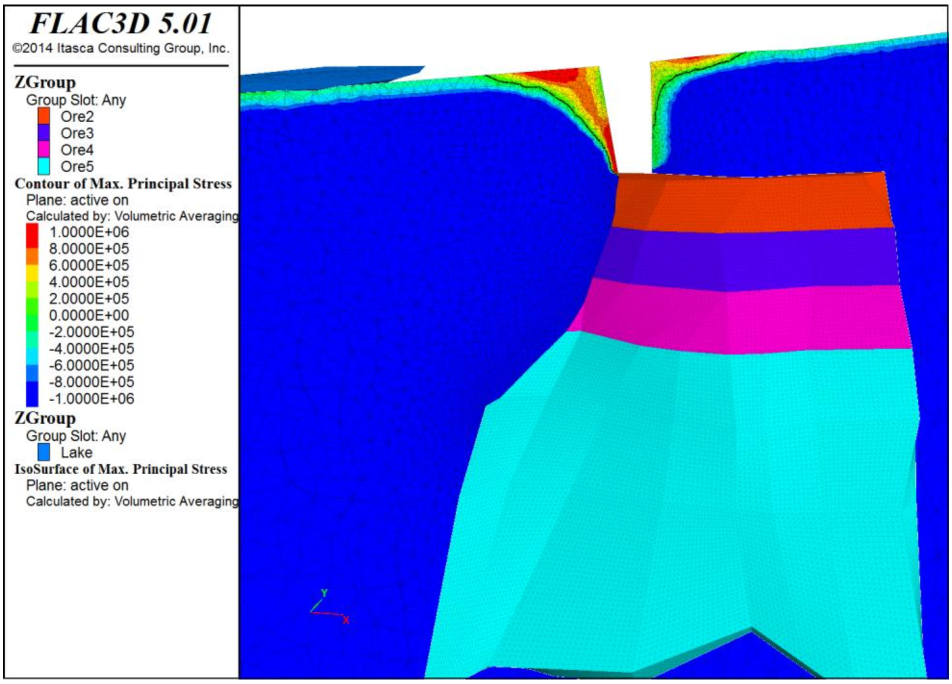

Figure 3. Minor principal stress for the current situation with a mining depth of 250 m. Black line represents 0 MPa iso-surface.

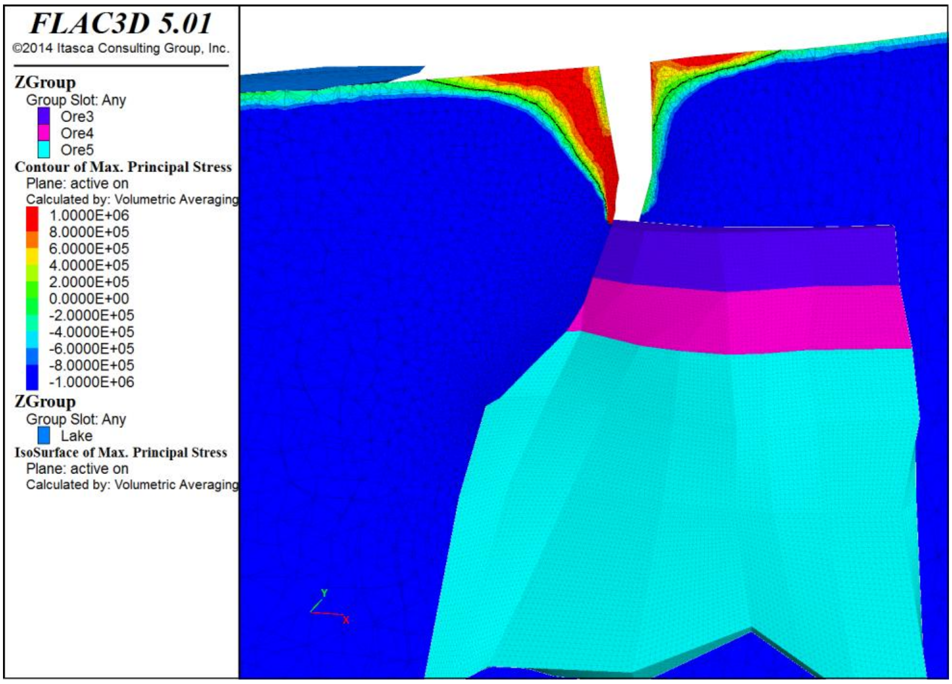

Figure 4. Minor principal stress for a mining depth of 350 m. Black line represents 0 MPa iso-surface.

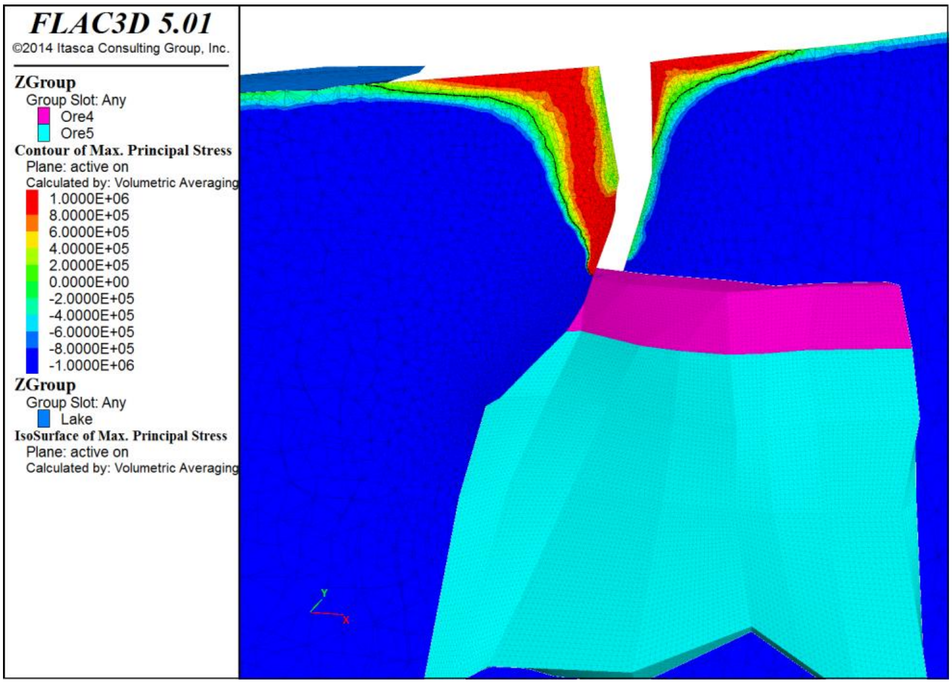

Figure 5. Minor principal stress for a mining depth of 450 m. Black line represents 0 MPa iso-surface.

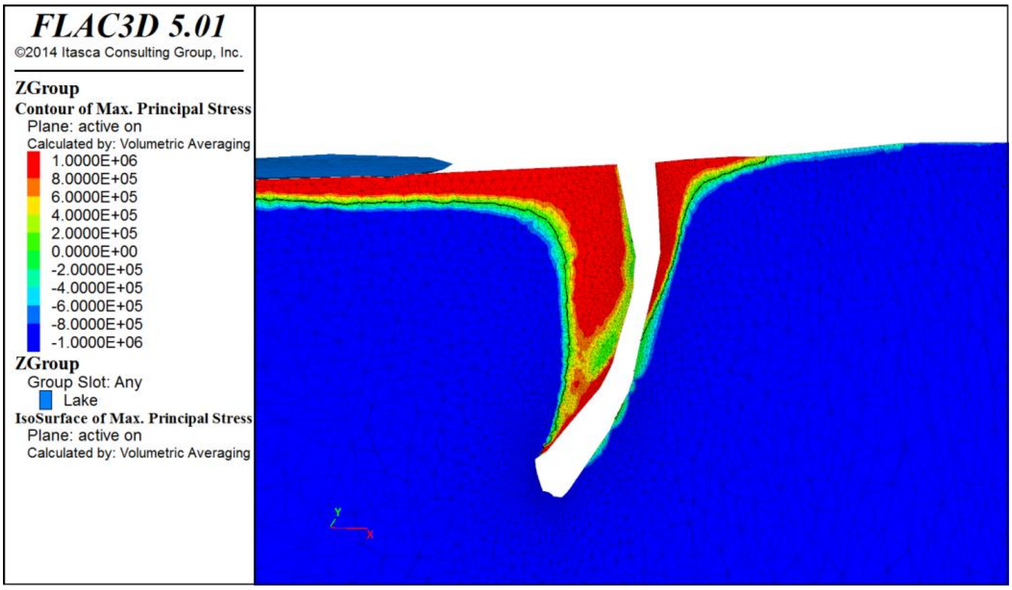

Figure 6. Minor principal stress after mining the known resourse. Black line represents 0 MPa iso-surface.

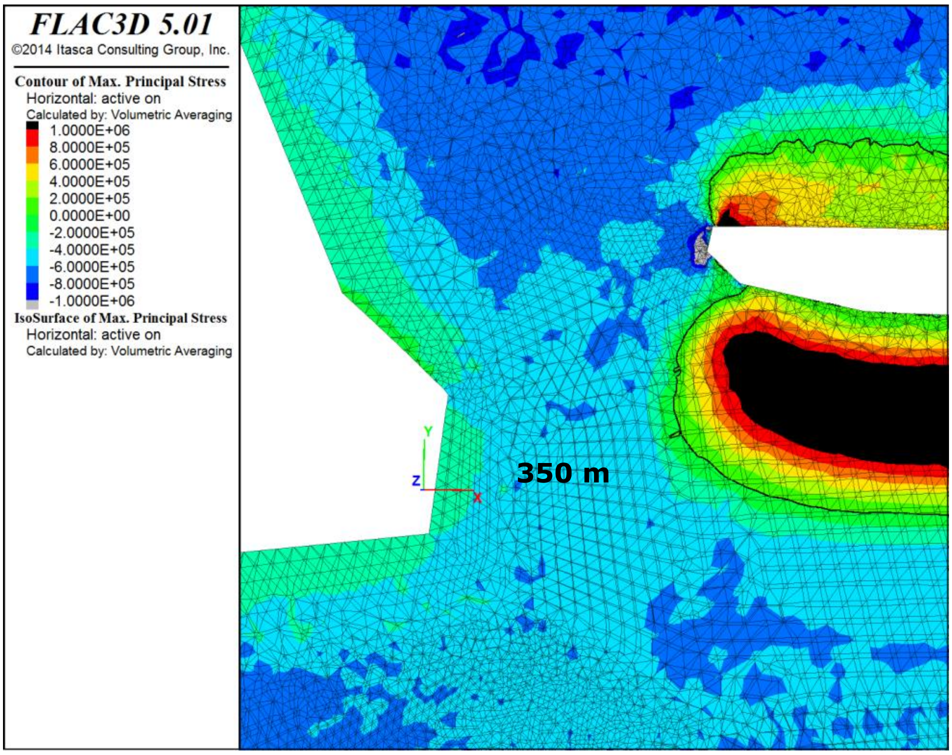

Figure 7. Minor principal stress for the current mining depth (250 m) at a plane located 20 m below ground surface. Black line represents a stress of 0 MPa.

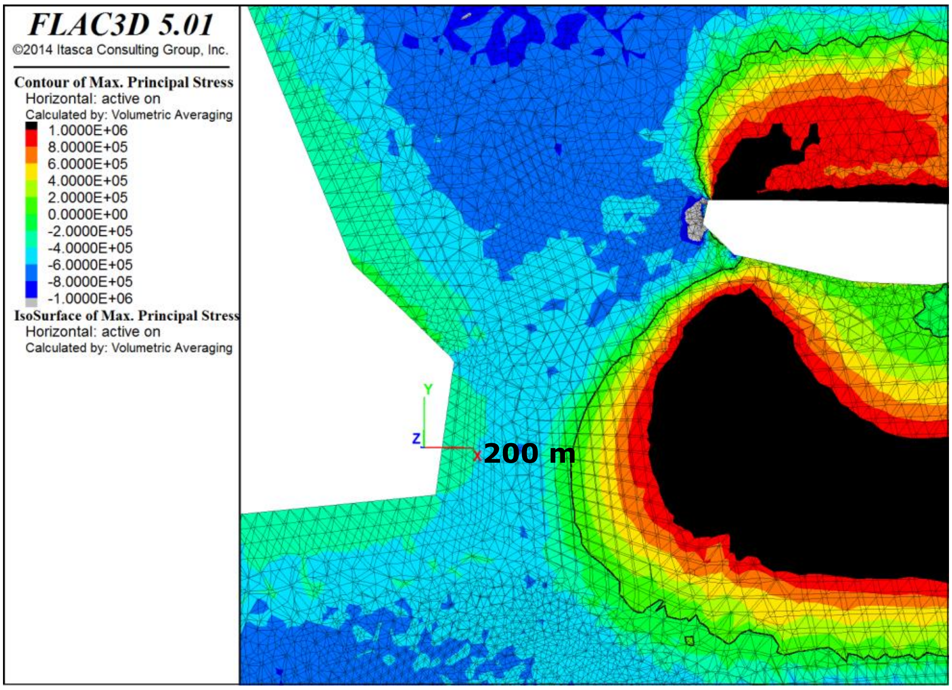

Figure 8. Minor principal stress at -20 m when the mining depth reaches 450 m. The black line represents a stress magnitude of 0 MPa.

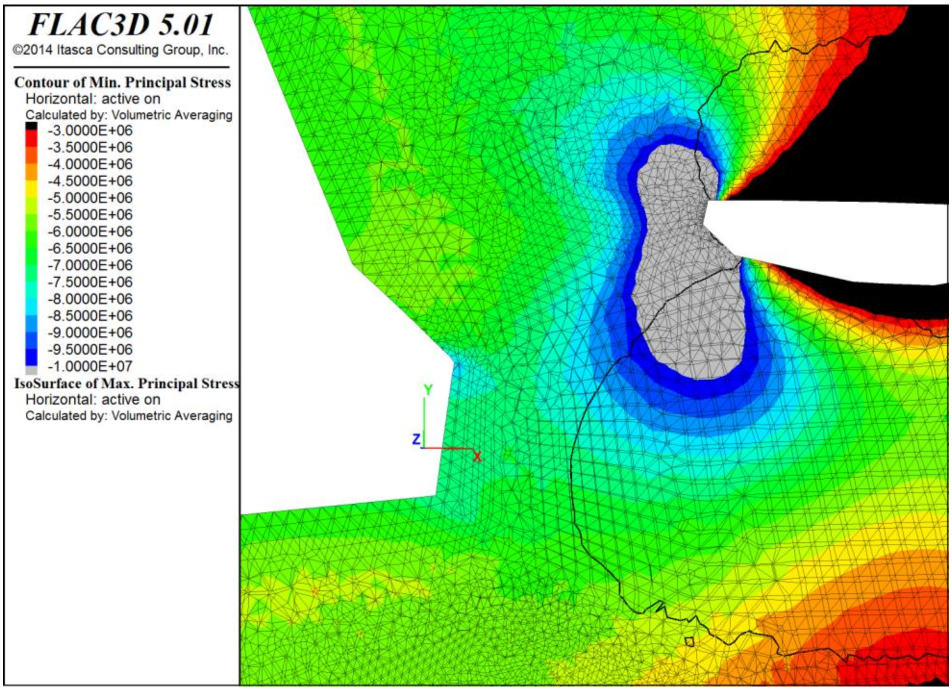

Figure 9. Major principal stress at -20 m when the mining depth reaches 450 m. The black line represents a stress magnitude of 0 MPa.

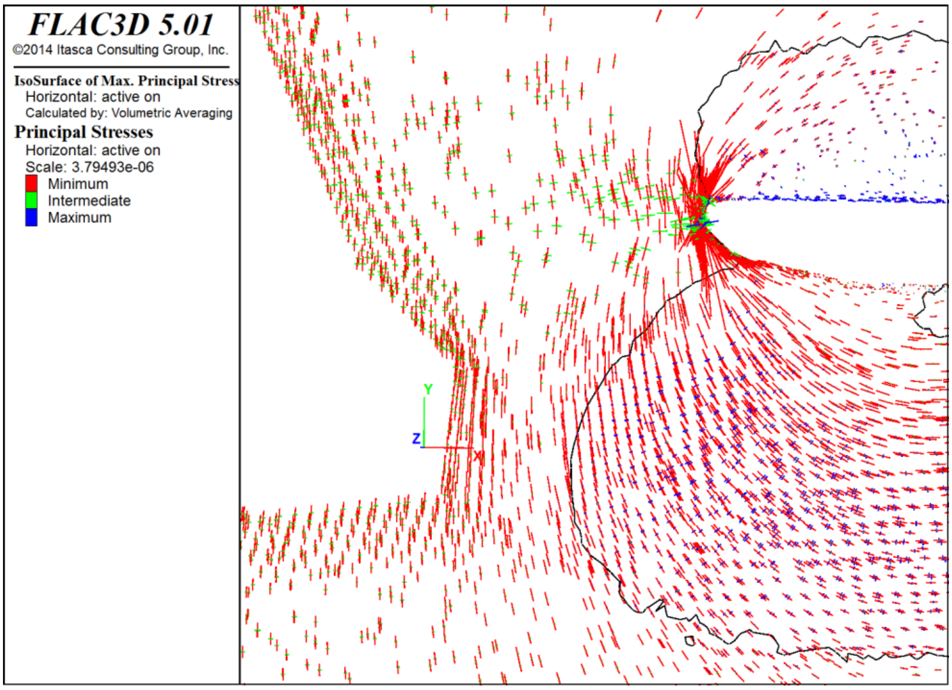

Figure 10. Stress tensors at -20 m when the mining depth reaches 450 m. The black line represents a minor principal stress magnitude of 0 MPa.

Displacement plots indicate the general deformations in the rock mass. The magnitudes are uncertain since they are directly dependent on the stiffness on the rock mass. For this analyse the stiffness is set to 40 GPa. This might be a bit low for the rock mass close to the ground surface and these deformations are therefore probably larger. When studying deformation results it is important to bear this in mind and mainly focus on directions and larger displacement magnitude differences.

The displacements presented are displacements occurring due to the future mining plans. The displacements that occurred when mining down to 250 m are not included.

As the ore body is mined out, the rock mass will displace towards the opening. For this elastic model the displacements are in the range of 10 cm close to the mined out ore body. Near the lake the displacements are approximately 10 % of that. Displacement magnitudes and vectors are illustrated in Figure 11 for the plane inclined 70 degrees. Displacement vectors on the horizontal plane at 20 m depth are illustrated in Figure 12. At ground surface the displacement reaches almost to the lake as illustrated in Figure 13. The rock mass behavior at the model boundaries are though uncertain due to the likelihood of increased fracturing at this level.

When the entire resource is mined displacements will likely occur beneath the lake as well. The total area influenced by displacements will be large, approximately 3 km South and 2 km North of the mine (Figure 14).

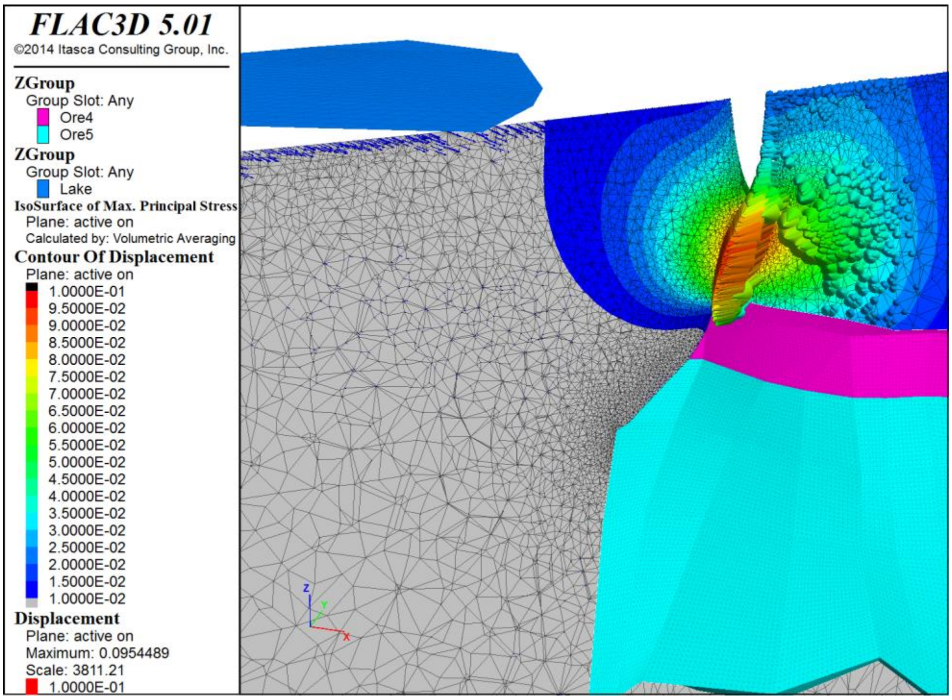

Figure 11. Illustration of total displacements and displacement vectors for the plane inclined 70 degrees. Mining depth 450 m.

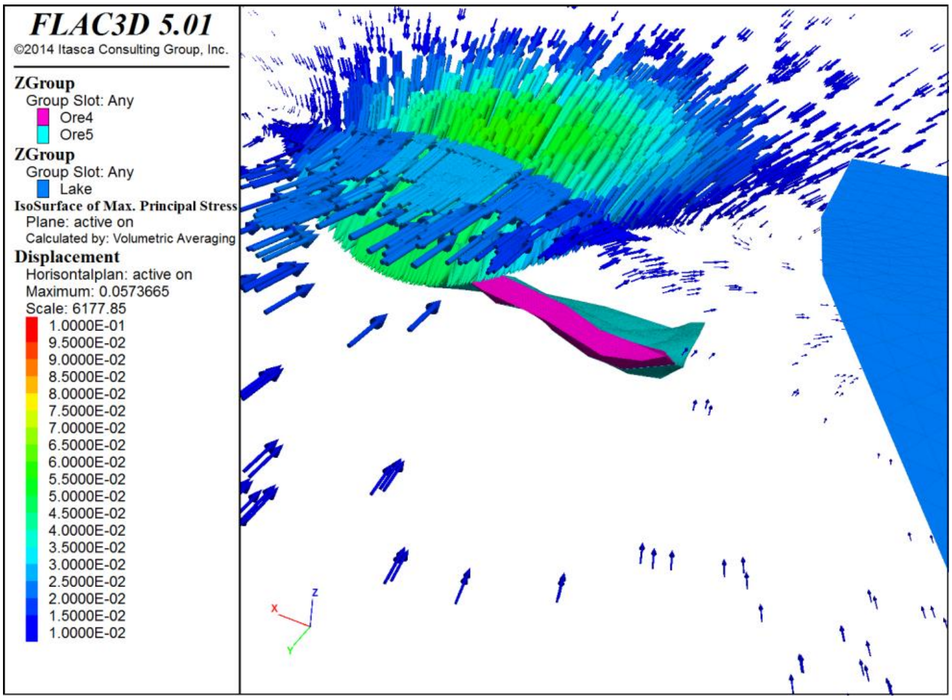

Figure 12. Illustration of displacement vectors for the horizontal plane at 20 m depth. Mining depth 450 m.

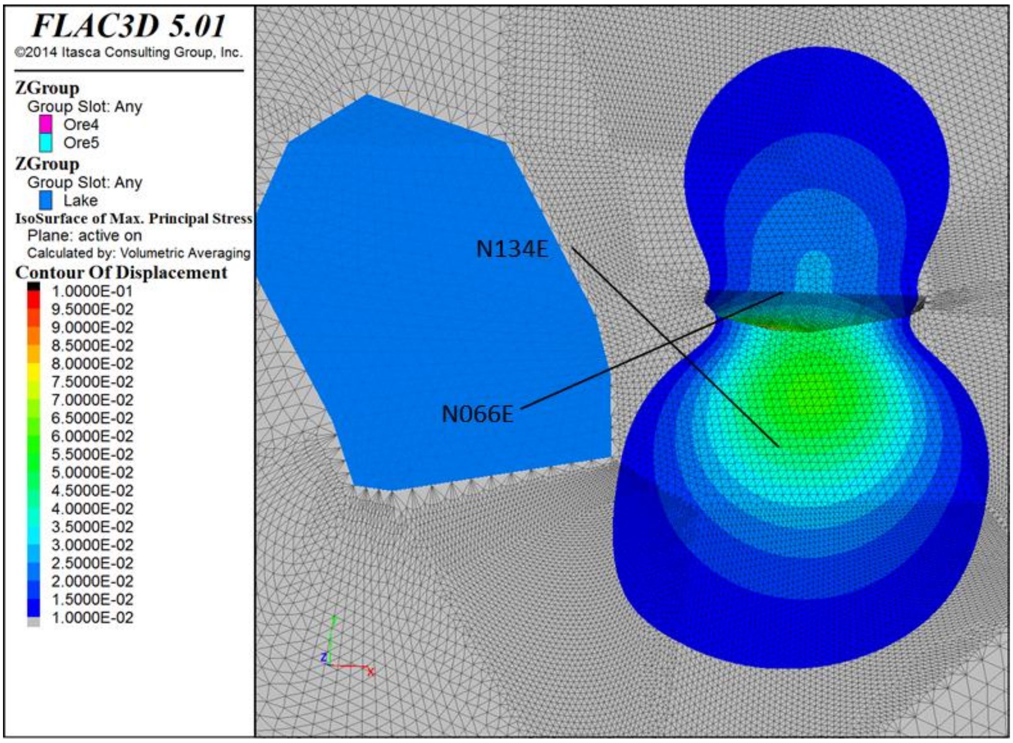

Figure 13. Illustration of total displacement at ground surface. The figure also includes the strike of the joint sets in Table 1. Mining depth 450 m.

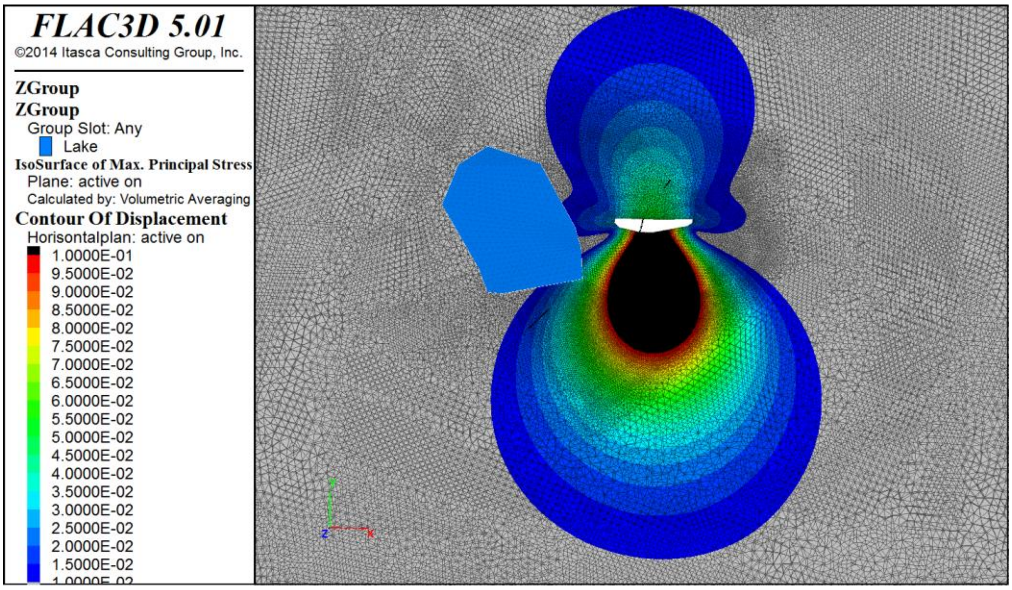

Figure 14. Illustration of total displacement at ground surface. Mining depth 900 m.

The joint sets found in the rock mass are illustrated in Figure 13. One of the joint sets (N066E) is trending in a direct line between the lake and the mine. The major principal stress is oriented almost perpendicular to this fracture set in the horizontal plane but the fracture dips 48 degrees making shearing possible close to the mine. The other joint sets striking, N134E, is common for two fracture sets with the dips 43 and 87 degrees respectively. This fracture orientation is almost perpendicular to the tensile stresses that develop between the mine and the lake. The fracture orientation is also such that shear stresses easily develop along the fracture planes because of the major principle stress.

7. Conclusions

The analyse indicates that stress and displacement changes will effect the rock mass relatively close to the lake when the mining depth reaches 450 m. When the whole resource is mined down to 900 m depth displacements will have occurred beneath the lake as well. The risk of the mining impacting the lake increases as the mining depth goes below 450 m.

The fractures in the rock mass have such orientations that they likely will be effected by the changes in stress and deformation and thereby may increase the conductivity of the rock mass between the lake and the mine.

8. Recommendations

The fracture orientations for the rock mass close to the ground surface should be verified.

People specialized in hydrogeology should look into the results from the numerical model and the fracture orientations in more detail to further assess the possible conductivity changes in the rock mass due to the mining activities.

The numerical results are dependent on the use of correct parameters. The in situ stress field is poorly known at the moment. Rock stress measurements should be performed relatively close to the mine. Though outside the influence area of the mine. The results should be used as input data in numerical models to increase the accuracy of the results. In the mean while, more scooping calculations should be performed to investigate the effect of higher and lower stresses on the rock mass response due to the mining.

The results from this model can be used for other purposes when studying the effect of the mining on the surrounding rock mass. The stress data can be imported in 2D numerical tools to quickly assess how the mining effect ramps and drifts. The 3D data can also be used for assessing where drifts and caverns should be located within the rock mass to avoid stability issues as the mining progresses downwards. The stress situation around the hanging wall is of interest when assessing probable failure modes in the side rock which can aid when determining the mining method. Yielding around the mine can be illustrated in elasto-plastic models by assessing the rock mass mechanical properties.

Appendix G

MATERIALS HANDLING OPTION STUDY

Nordic Iron Ore™

DMT

DMT Consulting Limited
International Mining Consultants

BLÖTBERGET DECLINE HAULAGE OPTION STUDY

PREPARED FOR
Nordic Iron Ore™

DOCUMENT REF.: C22-C-00

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Summary

DMT Consulting Limited (DMT) has been requested to provide engineering services to Nordic Iron Ore (NIO) to carry out a Comparison Study of Different Hauling and Hoisting Systems for the Blötberget Iron Ore Mine Project (the Project) in Sweden. The comparison study will consist of evaluating alternative hoisting systems (Task 1) and comparison of mine access tunnel alternatives (Task 2).

The Study will provide an economic comparison of the life costs (capital and operating from start of pre-production to end of mine life) of the alternative haulage systems and tunnel access options considered for use to access and transport the underground production to the surface on the Blötberget decline. The hauling alternatives include shaft hoisting, diesel powered trucks, electric powered trucks, trough belt conveyors and developing and working the mine with a movable crusher facility. The tunnel alternatives will consider single and twin tunnel access to the mine workings.

The study has reviewed various options for the method of transport of the materials but only those options considered practical have been included in the final cost comparison study.

The main emphasis of the study is the comparison between the use of trucks or conveyors as the means of transport and the potential to adopt the concept of a movable crusher and associated infrastructure equipment to develop and work the mine.

Task 1

Alt 1 Hoisting/Conveyor Option

This option considered the installation of a skip winding system at the 240 level in the existing Blötberget shaft, skip loading and discharge at the 570 and 270 levels respectively. The truck unloading and crushing facility would be the same as the conveyor and truck option and would load into the skips via the ore and waste silos at a rate of 600tph. The skip discharge arrangement is installed at the 270 level and ROM is transported to the surface via a feeder and conveyor arrangement which is rated at 600tph. Shaft systems are generally maintenance intensive and due to this requirement, 20 hours per day at 83% utilisation has been adopted to achieve the projected annual tonnage.

The transport of waste rock and ore will be controlled by the mine management and mined at pre arranged times which will allow the crushing and storage of the individual materials to be managed prior to being conveyed to the surface.

It is planned to reutilise the existing Blötberget shaft for the new winding apparatus when the mine has been dewatered. There is currently no indication of the condition of the shaft or what remedial work would be required to bring the shaft back to a sufficient standard to install and operate a winding system and therefore at this stage similar costs to sinking and relining of the shaft have been included to ensure sufficient allocation of finance to refurbish

the existing shaft; however only 300m of shaft have been considered in the financial estimation which represents the actual winding depth.

It is unclear what ventilation route will be adopted for the mine ventilation system but the introduction of a mid shaft winder will undoubtedly present an obstruction to the mine airflow if the shaft is planned to be used as a return airway.

The following table represents the financial estimates for the hoisting/ conveyor option:

		Pre-Production CAPEX €	Sustaining CAPEX €	OPEX €	Cost/t ore €
Winder 570 level to -270 & CV to Surface	**Winder/Conveyor**				
Winder& Conveyors		44,824,692	480,000	24,701,787	1.67

The operation of an underground winding system introduces additional maintenance issues for the winding apparatus. The changing of skips and winding ropes is normally carried out on the surface for winding installations and it will be essential that adequate excavations are provided at -240m level to facilitate these maintenance operations if this option is selected.

Alt 2 Conveyor Option

To convey the projected annual ore and waste from the mine workings at -570 level to the surface, a troughed belt conveyor arrangement has been reviewed.

The selected conveyor configuration for a conventional troughed belt installation will utilise three straight conveyor flight installations in the respective declines and a further conveyor to the loading point and forms the basis for the conveyor option compared to the truck haulage comparison.

Cost of Conveyor Options

The cost analysis has shown that the use of conveyors for the transport of ore and waste from the mine is cheaper both in capital cost and operational costs than the use of a fleet of trucks.

The following table represents the financial estimates for the conveyor option:

		Pre-Production CAPEX €	Sustaining CAPEX €	OPEX €	Cost/t ore €
Conveyor -570level to -39 Surface	**Conveyors / Feeders**				
Conveyors & Feeders	8	20,032,225	4,536,000	34,751,806	1.41

From a cost perspective the use of conveyors for the duty prescribed presents the most cost effective option.

Alt 3 Truck Option

The use of both electric and diesel driven trucks has been considered with the relative advantages and disadvantages of the two types of power source discussed; however the main advantages of electric trucks are faster constant speed and reduced ventilation requirements.

The profile of the proposed decline ($36m^2$) has not been agreed and would not permit two-way traffic; therefore trucks will have to pass each other in the roadway using pass-byes. An access roadway allowing two-lane traffic to operate to the -300 level has been provided for consideration.

To prioritise the loaded trucks travelling up the grade to maintain a constant speed and avoid unnecessary stopping and starting it would be beneficial to give them preference and make the trucks travelling down the grade wait at a pass-bye for the ascending loaded truck to pass.

In the interest of safety and efficiency a comprehensive traffic management system should be installed which could monitor the location of each truck and control the traffic flow in the decline.

The estimated time to complete one journey, surface to underground, load and unload has been calculated at approximately 51 minutes for the selected option. In the event of a truck having to make an emergency return to the surface, it is essential that areas are provided to allow the truck to turn around.

The following table identifies the number of trucks required for payloads of 50t and 40t. The calculations are based on 350 days/annum x 3 shifts at 85% availability (approximately 7 hours/shift) with the surface change over and the loading/unloading times as stated in the report and include both diesel and electric trucks.

The following table represents the estimates for the truck requirement if trucks were utilized to transport product from the 570 level to the surface:

Number of Trucks for 3 x Eight Hour Shifts		
	50t trucks	**35*/40t trucks**
(Diesel) 12km/hr.	10	12
(Electric)20km/hr.	6	9

The working hours during the shift will be crucial to the number of transport cycles achieved. At the end of the shift the truck should be on the surface to allow the drivers to changeover. Therefore if a truck is within 51min of the end of a shift (based on a 12km/hr. speed) it is unlikely the driver will start the next cycle because it cannot be completed within the shift time. This lost time will vary for each truck as they complete their individual cycles, but for an eight hour shift it could represent a significant loss of haulage time. An increase in waiting time during the shift cycle has been included to cover any potential operational issues which may arise.

Primarily for the purposes of the study a 50t truck has been selected as the operational case but the impact of using 40/35t trucks has been analysed for an alternative comparison.

The estimated capital and operating costs for the study and works have been collated from manufactures data, DMT experience and NIO previous study costs utilising standard industry databases. The operating costs have been estimated primarily based on the projected annual tonnage, total annual operating hours of each truck and estimated manpower costs.

The cost analysis of operating 50tonne trucks is summarised as follows;

	No Trucks	Pre-Production CAPEX €	Sustaining CAPEX €	OPEX €	Cost/t ore €
Selected Case (12km/hr)					
50t diesel truck	10	20,343,174	10,439,216	119,500,958	3.58
50t electric trucks	6	31,662,874	16,249,716	60,475,252	2.58

The chosen case of using 50t diesel trucks estimates a cost/t of ore at 3.58€. The cost/t of ore using a 50t electric truck is estimated at 2.58€.

The breakdown of costs identifies that there is significant financial difference between the use of an electric powered truck and a diesel driven truck. The electric trucks are more expensive to buy and install but the operating costs, particularly the energy, are very much reduced. If in the overall selection of the haulage option, the use of trucks is preferred to the installation of a conveyor system then the use of electric powered trucks should be given serious consideration.

Alt 4 Movable Crusher Facility

This option review incorporates a main conveyor installed in the decline down to the 365 level and considers the potential for installing a movable crusher facility. It is proposed to operate a mining system which will establish a crusher facility at 60m depth intervals and convey the ROM product by conveyor installations to the 365 level where the mineral will be conveyed to the surface on the main conveyor. The movable crusher design and associated equipment design has not currently been finalised.

The cost analysis of operating a Movable Crusher facility is summarised as follows;

	Pre-Production CAPEX €	Sustaining CAPEX €	OPEX €	Cost/t ore €
Movable Crusher & Associated Conveyors				
	15,138,048	10,520,486	35,754,794	1.46

The financial estimates used for the movable crusher option are estimated and will need to be reassessed when a final design has been agreed. The conveyor financial estimates are based on the costs used in the alternative options and should be representative of the equipment required.

Task 2

Alt 1 Two Tunnel Access

The study reviewed the possibility of utilising an access tunnel and conveyor tunnel to access the -365m level underground workings. The access tunnel would be utilized for all mobile and mining equipment to be transported underground and additionally have pass byes for two-way traffic. The conveyor tunnel would allow the installation of a 600tph conveyor and service vehicle access to the underground workings.

The capital drivage cost for an *access tunnel* is summarised as follows;

Blötberget CAPEX Drivage Costs (€2,400/m)				
			Unit Cost	
			Mkr	**€**
(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Access Ramp (36m²) -39 to -310			**Mkr**	**€**
Cost/Metre €		2400		
Ramp 1 -39 Surface to -108	1in8	554	12,381,089	1,329,600
Ramp 2 -108 to -141	1in8	280	6,257,590	672,000
Ramp 3 -141 to -209	1in8	514	11,487,148	1,233,600
Ramp 4 -209 to -310	1in7	817	18,258,754	1,960,800
X Slit 1 at -141 level		141	3,151,144	338,400
X Slit 2 at -238 level		324	7,240,926	777,600
Total		***2630***	***58,776,650***	***6,312,000***
Contingency (15%)			8,816,497	946,800
Total			**67,593,147**	**7,258,800**

The capital drivage cost for a *conveyor tunnel* is summarised as follows;

Blötberget CAPEX Drivage Costs (€2,400/m)				
			Unit Cost	
			Mkr	**€**
(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Conveyor Ramp (36m²) -39 to -365			**Mkr**	**€**
Cost/Metre €		2400		
Ramp 1 -39 Surface to -270	1in7	2268	50,686,480	5,443,200
Total		***2268***	***50,686,480***	***5,443,200***
Contingency (15%)			7,602,972	816,480
Total			**58,289,452**	**6,259,680**

A Capital estimate for a further alternative to allow two-lane traffic throughout its length was also reviewed and shown below.

The capital drivage cost for a *two-way truck tunnel* is summarised as follows;

Blötberget CAPEX Drivage Costs (€3,165/m)				
			Unit Cost	
			Mkr	**€**
(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Two Way Access Ramp (54m²) -39 to -365			**Mkr**	**€**
Cost/Metre €		3165		
Ramp 1 -39 Surface to -365	1in7	2268	66,842,795	7,178,220
Total		***2268***	***66,842,795***	***7,178,220***
Contingency (15%)			10,026,419	1,076,733
Total			**76,869,214**	**8,254,953**

Alt 2 Single Tunnel Access for Conveyor and Mine Equipment

The layout schematic for this option identifies a rectangular cross section roadway of 6.5m wide and the height will be specified by the ventilation requirement for the mine; however the cost of a 36m^2 cross section tunnel has been provided to allow the comparison of costs.

Blötberget CAPEX Drivage Costs (€2,400/m)				
			Unit Cost	
			Mkr	€
(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Conveyor Equipment Ramp (36m²) -39 to -365			**Mkr**	**€**
Cost/Metre €		2400		
Ramp 1 -39 Surface to -365	1in7	2286	50,999,359	5,476,800
Total		***2286***	50,999,359	***5,476,800***
Contingency (15%)			7,649,903	821,520
Total			**58,649,262**	**6,298,320**

Conclusions

All three haulage alternatives hoisting/conveyor, conveyor and truck systems are practical mining operations that can be utilised for the mining and transportation of ore and waste out of the mine at the proposed annual production rates. The practicality of operating a movable crusher facility has still to be confirmed; however the costs indicated should be representative of the planned mining system.

The selected option of 50 tonne capacity trucks which has been reviewed can be accommodated in the proposed roadway cross section ($36m^2$).

The financial cost analysis for the truck versus conveyor option identifies that the use of conveyors for the transport of ore and waste from the mine is the most cost effective option.

The capital drivage costs identified are representative for the tunnel alternatives reviewed and provide a realistic cost comparison of the alternative systems. The drivage costs should be reviewed when the mine design has been agreed and the required roadway specifications have been established.

1 INTRODUCTION

DMT Consulting Limited (DMT) has been requested to provide engineering services to Nordic Iron Ore (NIO) to carry out a Haulage Option Study for the Blötberget Mine Project (the Project) in Ludvika, Sweden.

The Blötberget Iron Ore Mine was closed in the early 1990's and is currently in a totally flooded status. The company has carried out investigation work for the re-opening of the Blötberget Mine and now wishes to move the project forward.

As part of the development process, NIO intend to review a number of options to transport the ore and waste product from underground to the surface to a location near the planned processing plant. The mine is planning to produce up to 3,000,000 tonnes per year of ore and 500,000 tonnes of waste product per year.

The optimum method of hauling the ore and waste out of the mine and tunnel development options are the subject of the review report.

1.1.1 Terms of Reference

The data produced from the Study will develop an economic comparison of the whole costs (capital and operating from start of production to end of mine life 14 years) of the alternative haulage systems reviewed for use at the Blötberget mine. The options to be reviewed will include diesel trucks, electric trucks, troughed belt conveyors, and a hoist/winder conveyor arrangement.

The level of engineering carried out will be conceptual but sufficient to allow a capital and operating cost estimate to be generated for each alternative for inclusion in a cost analysis of the various options considered.

DMT have worked closely with the NIO team to ensure that their proposed ideas on the potential haulage systems are taken into consideration in the development of the engineering and economic analysis.

1.1.2 Haulage options considered

The main emphasis of the study is the comparison between the use of trucks or conveyors as the means of transport. Variations within the options have been considered, including the use of electric powered trucks rather than diesel.

2 TRUCK OPTION

2.1 Truck Selection

The cross section of the decline has been is assumed to be approximately 36m^2 and this will assist to determine the maximum size of truck that can be used in the decline. There must be sufficient clearance for the trucks to pass other items of equipment required in the roadway. Based on manufacturer's specifications the maximum size of truck is considered to be in the range of 35 to 55 tonne capacity with a variety of body sizes dependant upon material bulk density and loading characteristics.

The use of both electric and diesel driven trucks has been considered. The relative advantages and disadvantages of the two types of power source are discussed later but the main difference affecting the analysis at this stage is the speed of the trucks and ventilation requirements. Electric trucks can attain a higher speed on a given gradient and consequently this will impact on the total number of trucks required.

The data relating to the choice of truck, electric or diesel, where applicable, are shown side by side throughout the report.

2.1.1 Body Capacity

The standard body capacity of trucks of this range is between 20m^3 and 27m^3. The waste material is noticeably heavier than the ore material and therefore a truck with a standard body can potentially carry more weight of waste than ore. Therefore if the body truck is sized to carry 50t of ore it may be possible to overload the machine by filling it with waste and as the majority of the product produced will be ore it will be necessary to manage the operation to avoid overfilling the truck.

The following table details the effect of using a standard body with a payload sized to carry 50t of ore. The analysis is based solely on the difference in specific gravity between the ore and waste and makes no allowance for any difference caused by bulk loading or moisture content.

Table 2-1 Truck loads for standard truck body sizes

				No Trucks/yr.
	Tonnes	SG	Cubic Metres	Standard Body
Ore	3,000,000	2.6	1,153,846	*60,000
Waste	500,000	3	166,667	10,000
Total Truck Cycles/yr.				70,000
Truck Loads per shift (350 days, 3 shifts)				22/shift

*50t Max Load

The number of cycles required is determined by the need to haul 3.5Mt of ore and waste per annum and shows 22cycles/shift.

2.2 Cycle Times

2.2.1 Ramp travel times and Pass-byes

The speeds of the trucks used in the calculation of cycle times are based on the speeds specified by the manufacturers for a fully laden truck travelling up the relative grades on the respective decline.

The proposed size of the decline ($36m^2$) will not permit two-way traffic; therefore the trucks will have to pass each other in the roadway using pass-byes. To enable the trucks travelling up the grade to maintain a constant speed and avoid unnecessary stopping and starting it would be beneficial to give them preference and make the trucks travelling down the grade wait at a pass-bye for the loaded truck to pass.

The decent of the trucks will therefore be determined by the speed of the trucks climbing the grade and the location of pass-byes.

A traffic control system should be adopted so that the trucks can travel in either direction on the decline without obstructing each other's path. The decline would have to be divided into predetermined sections defined by the pass-byes.

Three basic transport rules should be adopted:

1. Priority should be given to the loaded upward truck so that it does not have to stop (and restart) on the decline.
2. A truck must not enter a section if a truck travelling in the opposite direction is already in that section.
3. A downward truck must not enter a section if a loaded upward truck is in either of the next two sections.

2.2.2 Location of Pass-byes

The spacing of pass-byes should be designed to ensure the smooth flow of trucks throughout the decline. Downward trucks having greater speed potential should travel from section to section utilising the pass byes and wait for upward trucks to pass where necessary.

The spacing of the pass-byes will be determined by the speed of the upward trucks and the distance between them.

Two assumptions can be used to assess the location of the pass-byes:

1. The trucks are equally spaced throughout the ramp – in which case the location of the pass-byes will be determined by the number of trucks.
2. The spacing of the trucks will be determined by the loading time at the underground loading station – in which case the location of the pass-byes will be determine by the speed of the trucks i.e. the truck spacing is the distance travelled by a truck whilst the next truck is loaded.

In reality the second assumption will be prevalent at the start of the shift whilst all the trucks are loaded but during the shift the trend will be towards a more irregular spacing as trucks speeds and loading/unloading times vary and change.

To comply with the third basic transport rule for control of the trucks, the distance between pass-byes will be half the distance calculated by either method.

As an example, based on the application of 50 tonne trucks, and eight hour shift cycle , the follow spacing of pass-byes apply;

Table 2-2 Potential Pass-bye spacing based on speed

Based on loading time		Spacing (m)
Speed	8km/hr.	333
	12km/hr.	500
	15km/hr.	625
(Electric)	20km/hr.	833

Table 2-3 Potential Pass-bye spacing based on number of trucks

Based on No of Trucks	Spacing (m)
12	162
10	194
8	243
6	323

To prevent unnecessary delays it is essential that the pass-byes are as close as possible and for the purpose of the study a distance of **400m between pass-byes** has been assumed.

2.2.3 Turnarounds

The estimated cycle time of the trucks is based on a speed of 12km/hr. which is approximately 51mins. This means once the truck enters the roadway it could be nearly one hour before it can re-surface. There is a need therefore to provide means of turning the trucks around to allow for an emergency return to the surface if required.

'Turnarounds' should therefore be located throughout the ramp to facilitate this need. The ramp is approximately 3,885 metres long and potentially at least four such turnarounds should be provided. A turn-around would normally comprise of widening of the roadway, probably in the form of a short roadway stub, which would allow a truck to drive in to and reverse out of, thereby turning the vehicle around.

During the development of the decline, short side stubs 20m deep are normally driven every three hundred metres approximately for the parking and storage of the development equipment and for the turning of vehicles. Several of these cross cuts could be maintained during the production period to act as turnarounds.

For the purposes of the report four turnarounds have been included in the capital cost of the trucks.

2.2.4 Average Speed for each cycle

The speed of the upward truck should, as far as possible, be maintained constant throughout the climb, avoiding the need to stop and start with consequential increased wear and tear on the vehicle. The downward truck can vary its speed more easily and will have to stop and start frequently during it's decent to allow the upward vehicle to pass. However, its average speed for the decent must not be less than that for the ascent otherwise the production rate cannot be met. If the decent is faster the truck may have to wait to be loaded and any advantage gained by a fast decent will be lost.

Based on the above comment, the need for pass-byes and the control of traffic giving preference to the upward truck, it has been assumed that the time for the downward truck to complete its decent will be the same as the upward truck, i.e. the average speeds will be the same in each direction.

2.2.5 Transport of Ore and Waste

The trucks will transport either ore or waste which will be loaded and transported constantly throughout the day. The ore will be loaded underground from an ore pass via chutes and control gates. The waste will be loaded and transported separately when required from a similar nearby waste pass/chute. Both the ore and waste will be loaded without any secondary sizing other than through a grizzly.

The ore and the waste will be delivered to adjacent sites on the surface. The ore will be tipped directly into a conveyor hopper feeding the secondary crusher prior to entering the processing plant.

As mentioned earlier, the actual payload of the truck will depend upon the density of the material hauled and management procedures should be introduced to ensure that the truck cannot be overloaded with waste.

2.2.6 Loading and Unloading times

To calculate potential overall cycle times it has been necessary to assume times for the loading and unloading of the trucks. For the ore and waste it has been assumed that five minutes will be required to load each truck underground and two minutes to discharge on the surface.

Potential variations in these times has been included in the study and increased waiting times has been incorporated to facilitate any operational issues.

2.3 Number of Trucks

2.3.1 Basis of calculations

The base case for determining the number of trucks required for the Blötberget mining operation is based on the following data and assumptions:

Table 2-4 Basic Parameters

DECLINE	
Grade 14.3% (-39 to 363)	2,286m
Grade 14.3% (-363 to 512)	1043m
Grade 14.3% (-512 to -570)	406m
Grade 1% -570	150m
Distance to u/g loading (level)	350m
Distance surface to stockpile(not ramp)	150m
Total Distance	**4,385m**
WORKING TIME	
Days/yr.	350
No Shifts/day	3
Hours/shift	8
TRUCK DATA	
Availability	85%
PRODUCTION	
Tonnage Ore	3,000,000t
Tonnage Waste	500,000t
Total Tonnes	**3,500,000t**

LOADING TIMES	
U/G Loading time	5min
Surface Dump time	2min

2.3.2 Number of Cycles

The number of truck cycles that can be achieved during the day is relative to the average speed of the truck and the working time during a shift.

The average speed is determined by the selection of the truck and the percentage of full load at which it operates. For a 50t diesel driven truck the average speed has been estimated as 12km/hr and for a 40t truck 10.5km/hr. The electric trucks are faster and an average speed of 20km/hr.has been used in the study.

The following table shows the cycle times for a 50t diesel truck over a range of average speeds.

Table 2-5 Truck Cycle Times

Average Speed		Minutes
Speed	8km/hr.	73
	12km/hr.	51
	15km/hr.	43
(Electric)	20km/hr.	34

The working hours during the shift is crucial to the number of cycles achieved. At the end of the shift the truck should be on the surface to allow the drivers to changeover. If a truck is within 51min of the end of a shift (12km/hr. speed), it is unlikely the driver will start the next journey because it cannot be completed within the shift time. This lost time will vary for each truck as they complete their individual cycles, but for an eight hour shift it can represent a significant loss of haulage time. For the purpose of the study the selected case has assumed an eight hour working shift.

2.3.3 Selected Case

Calculations have been made of the number of trucks required for payloads of 50t and 40t. These calculations assume an 8 hour shift with surface changeover and the loading/unloading times as stated previously.

Table 2-6 No of trucks for various speeds

Number of Trucks			
		50t trucks	35*/40t trucks
Diesel	12km/hr.	10	12
(Electric)	20km/hr.	6	9

From the above table it can be seen that utilising the smaller 40t truck would have a significant impact on the number of trucks required on the 8hr shift working pattern.

Analysis of truck sizes indicates that the size of a 50t truck will fit into the decline without any operational problems. Typical profiles of the trucks under consideration are shown below in Figure 2-1 and Figure 2-2 50t Electric Truck Profile.

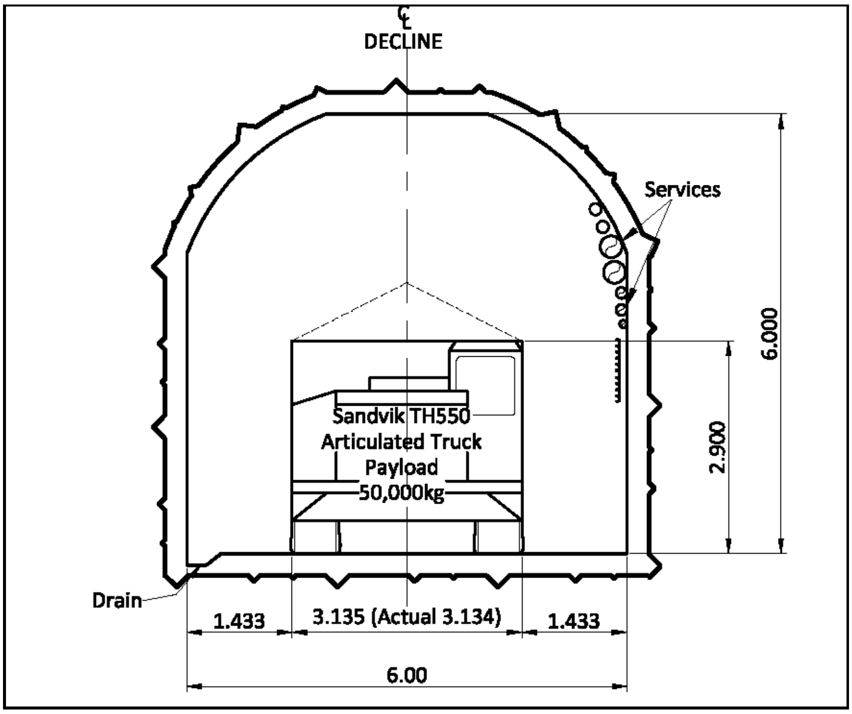


Figure 2-1 50t Diesel Truck Profile

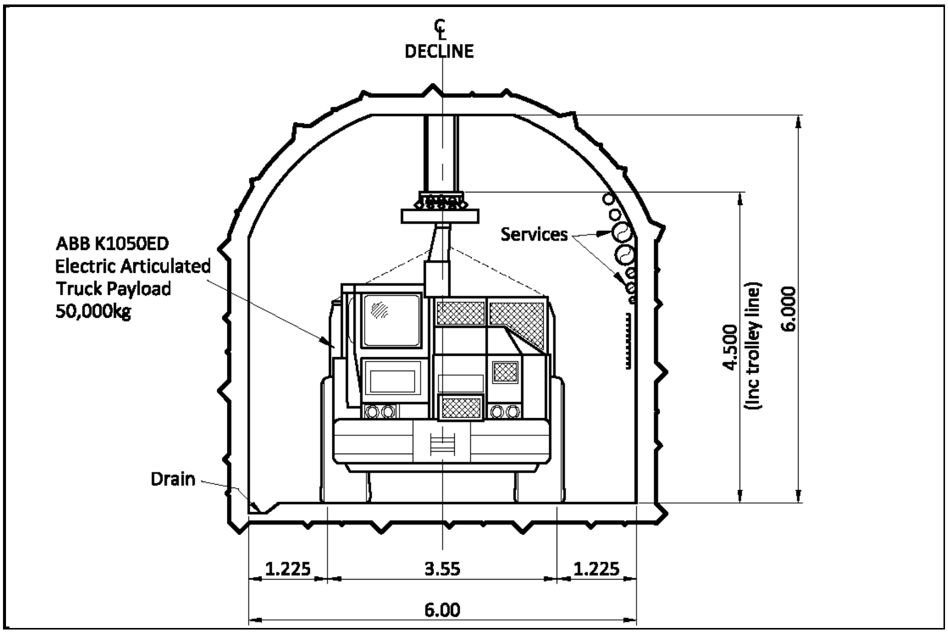


Figure 2-2 50t Electric Truck Profile

Manufactures technical data for the trucks available show an average speed of 12km/hr. is considered achievable travelling up the proposed declines. The selected case for the truck option has therefore been selected as:

- **50 tonne truck hauling ore, waste attaining an average of 12km/hr.**

On this basis the following numbers of trucks will be required.

Table 2-7 No of 50t trucks required

	Number of Trucks
	8 Hour Shift
50tdiesel truck, 12km/hr.	10
50t electric truck, 20km/hr.	6

2.3.4 Loading/Unloading Times

The effect of varying the loading and unloading times can have a significant operational impact and are shown below. These calculations are based on a 50t truck at speed of 12km/hr.

Table 2-8 Effect of loading/unloading times

	Number of Trucks
	8 Hour Shift
Half Loading/Unloading times	9

Selected Case Loading/Unloading times	10
Double Loading/Unloading times	12

The relative increase in the time assumed for loading and unloading does not significantly affect the number of trucks required for the haulage duty.

2.4 Cost Analysis of Truck Option

The estimated capital and operating costs of equipment and works have been collated from manufactures, DMT, NIO data and standard industry databases.

2.4.1 Capital Costs

The capital costs used in the truck option cost analysis are summarised in the following tables.

Table 2-9 Capital Costs

Item	Cost €	Replacement
Diesel Truck – 50t each	978,500	6/7 years
Electric Truck – 50t each	2,575,00	10 years
Other vehicles	154,000	7/8 years
Truck Unloading/Crushing	8,240,000	
Traffic Management	772,000	Annual maintenance spares only
Workshop/service bay	4926045	
Mining works	284,280	
Road surfacing	356,940	10% per year

Costs have been included for the provision of an underground workshop fully equipped to maintain and overhaul the haul trucks. The cost of a service bay for routine servicing of the trucks during the shift underground has also been included.

Provision has also been made for a comprehensive traffic management system which would be required to ensure the safe and smooth flow of trucks through the decline.

Provision for a substantial road surface has been included to reduce the wear and tear of the truck tyres and an additional 10% annual cost for repairs.

2.4.2 Operating Costs

The operating costs are mainly based on the estimated annual operating hours of each truck. From the calculations of cycle times the average operating hours for each 50t truck are as follows.

Table 2-10 Average Operating Hours

		Shift Time 8hr
Diesel	12km/hr.	6,429
Electric	20km/hr.	6,689

The individual operating costs for the respective trucks have been estimated utilising the following unit costs which have been acquired from internet databases, manufacturers and industry standards.

Table 2-11 Unit Costs

Fuel	1.24	€/l
Maintenance Parts & Oils	10.3	€/hr.
Maintenance Labour	786,000	€/year
Tyres	10,300	€ each

The operating costs estimated for the selected case of 50t trucks at 12km/hr are shown below. Estimates are shown for both diesel and electric trucks.

Table 2-12 Operating costs -50t diesel truck

50t Diesel Truck - 8 hour shift -12km/hr - Backfill					
No trucks	10				
Operating hrs/yr/truck	6,429				
OPERATING COSTS	Unit Cost		Rate		Annual cost
	€				€
Driver	35,000	/yr	36		1,260,000
Fuel	1.24	/l	4,179,075	l/yr	5,165,337
Maintenance Parts & Lub	10.30	/hr	64,293	hr	662,223
Maintenance Labour					786,000
Tyres	10,300	each	64.3	/yr	662,223
Total					**8,535,783**

Table 2-13 Operating costs – 50t electric truck

50t Electric Truck - 8 hour shift -20km/hr - Backfill					
No trucks	6				
Operating hrs/yr/truck	6,689				
OPERATING COSTS	Unit Cost		Rate		Annual cost
	€				€
Driver	35,000	/yr	22		770,000
Energy	9.5	/hr/truck			381,277
Maintenance Parts & Lub					1,969,000
Maintenance Labour					786,000
Tyres	10,300	each	40.1	/yr	413,384
Total					**4,319,661**

3 CONVEYOR OPTION

3.1 Conveyor Capacity

The design capacity of the conveyor systems assumes an annual throughput from the underground mine of 3,500,000 tonnes per year (3,000,000tonnes of Ore and 500,000tonnes of waste).

3.1.1 Average throughput for ore and waste

Operating Hours per day = 3 shift at 7 hours per shift = 21 hours/day

For conveyor operation assume 350 days per year.

Therefore annual hours = 350 x 21 = 7350hrs/year

Total Tonnage transported = 3,000,000 + 500,000 tonnes / year

Therefore average capacity = 3,500,000 / 7350

= 476tph

Assume conveyor availability = 85%

Therefore design capacity = average throughput / availability

= 476 / 0.85 = **560tph**

3.1.2 Four Main Trunk Conveyors

The basic parameters for a four conveyor option would be:

Table 3-1 Conveyor parameters

Parameter		Leg 1	Leg 2	Leg3	Leg4
Length	m	2286	1043	406	350
Lift	m	324	149	58	Level
Belt width	mm	1050	1050	1050	1050
Belt Spec		ST2500	ST2500	ST2500	ST2500
Belt Speed	m/s	3	3	3	3
Installed Power	kW	1000	500	200	100

3.1.3 Ore & Waste Belt Conveyor Loading

It is anticipated that a primary crusher installation underground would receive ore and waste rock from the ore and waste passes by truck. Figure 3-1 shows an arrangement of a crusher chamber incorporating an arrangement of a MMD apron feeder and sizer loaded from one side. The final arrangement will depend upon the type of crusher, mine layout and location of the ore and waste passes. If required the crusher could be fed from both sides. This capital estimate for a truck unloading & crushing application has been included in all three haulage study options for cost comparison excluding the 60m decline option.

It is recommended to crush to at least <250mm to minimise the risk of plugging in the surge bunkers and reduce impact loads on the conveyor belting. Note: The 1050mm wide belt adopted on the decline conveyors could accommodate lump sizes up to 300mm.

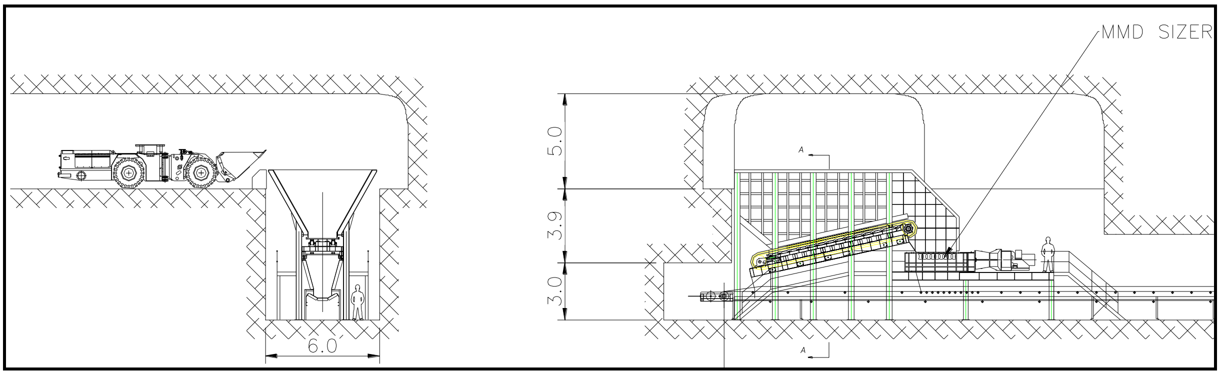


Figure 3-1 – Arrangement of a Crusher Chamber

The feed from apron feeders below each bunker will be regulated to prevent conveyor overloads.

3.1.4 Ore & Waste Belt Discharge

The ultimate leg of the conveyor system will exit the decline portal and transfer the material onto either the ore stockpile conveyor or the waste pile conveyor.

Two 150m long conveyors have been included in the costs to transport the ore to the ore stockpile and waste pile feed conveyors. However the final layout of these conveyors will depend upon the surface works, waste handling system and processing plant.

3.2 Cost Analysis of Conveyor Option

3.2.1 Capital Costs

Equipment

The capital costs of the equipment required are summarised as follows.

Table 3-2 Conveyor Option Capital Equipment Costs

Item	Total Cost €
Truck unloading & crushing	8,240,000
Apron Feeders	457,725
Conveyors	9,542,100
Foundations etc.	129,650

Electrical	990,675
Service Vehicles etc.	129,650
Total	**19,519,285**

Mining

Costs have been estimated for a solution considered practical and feasible. Further detailed design work must be undertaken to ascertain the optimum layout of roadways and excavations to accommodate the arrangement of equipment required; however the estimated cost for mining excavation work for some activities is shown below.

The Mining Costs are summarised as follows based on a ramp length of 3,885m:

Table 3-3 Conveyor Option Mining Costs

Item	Total Cost €
Pass-byes	92,700
Turn-a-rounds	49,440
Excavations	370,800
Total	**512,940**

3.2.2 Operating Costs

The operating costs of the equipment required are summarised as follows.

Table 3-4 Conveyor Option Operating Costs

Item	Annual Cost €
Power	661,353
Materials	1,134,992
Labour	678,000
Other (vehicles etc.)	7,927
Total	**2,482,272**

4 HOIST / CONVEYOR OPTION

4.1 Hoisting / Conveyor Capacity

The hoist/conveyor option consists of a winding system installed at the -240 level in the existing Blötberget shaft with a discharge facility at the -270 level and a loading facility at the -570 level giving a winding depth of 300m. The truck discharge and crushing facility would be the same arrangement as for the truck and conveyor option; situated at the -510 level discharging to the skip loading plant at -570 level. The design capacity requirement of the hoist/conveyor system is required to transport a combined annual throughput from the underground mine of 3,500,000 tonnes per year (3,000,000 tonnes of Ore + 500,000tonnes of waste). An automatic winding facility is generally maintenance intensive and for the basis of this option, winding for 22 hours per day and 2 hours per day for maintenance activities has been considered and an availability of 83%.

4.1.1 Average throughput for ore and waste

Hoisting hours per day = 22 hours/day

For conveyor operation assume 350 days per year.

Therefore annual hours = 350 x 22 = 7700 hrs/year

Total Tonnage transported = 3,000,000 + 500,000 tonnes / year

Therefore average capacity = 3,500,000 / 7700

= 454tph

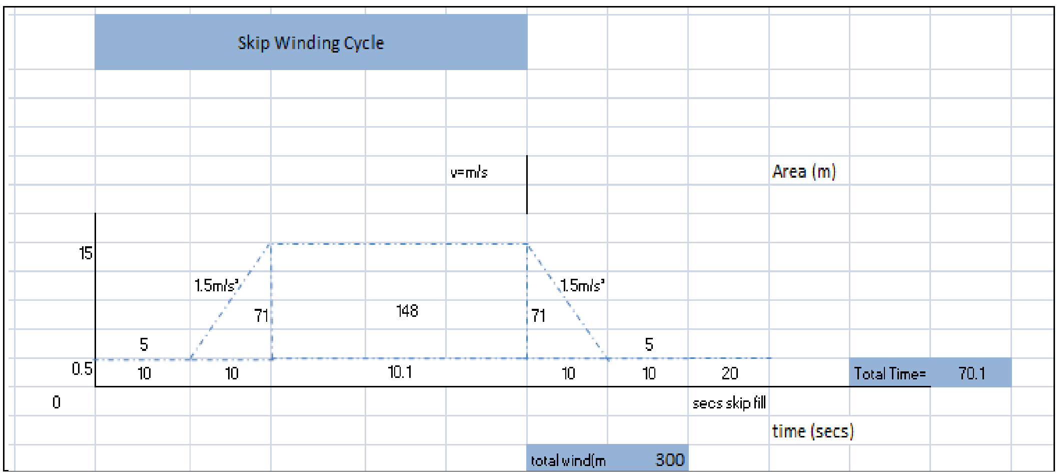


Figure 2 Estimated hoist time per cycle

4.1.2 Ore & Waste hoist and conveyor design capacity

The following are the assumptions adopted for this analysis:

Assume the Hoist and Conveyor combined availability = 83%

Therefore design capacity = average throughput / availability

= 455 / 0.83 = 547tph

The hoisting cycle has been estimated at approximately 71 seconds, therefore

Therefore skip design capacity = seconds per hour / cycle time

= 3600 / 71 = 50 skips/hour

Assume a skip capacity of 12 tonne at 50 skips per hour = 600 tonnes per hour.

The table below identifies varying number of operating days per year at 83% utilization and the tonnage per hour required to achieve the planned 3.5 million tonnes per year.

Table 4-1 Hoisting Capacity

Operating Days/Annum	350	340	330	320	310	300	290	280	270
Winding Tonnes/Hour at 22 hours/day and 83% Utilization									
4000000	626	644	664	685	707	730	755	782	811
3500000	**548**	**564**	**581**	**599**	**618**	**639**	**661**	**685**	**710**
3000000	469	483	498	513	530	548	567	587	608
2500000	391	403	415	428	442	456	472	489	507
2000000	313	322	332	342	353	365	378	391	406

The above table identifies that the proposed production (3,500,000tonnes) using a 12 tonne skipping arrangement can be achieved with a minimum 320 working days per year.

4.2 Cost Analysis of Hoist/Conveyor Option

4.2.1 Capital Costs

Equipment

The estimated capital costs of the equipment required are summarised as follows.

Table 4-2 Hoist/Conveyor Option Equipment Costs

Item	Total Cost €

800Kw Shaft & Hoisting Equipment	25,907,287
Civils Shaft & Hoisting Equipment	5,853,387
Conveyors (2x150m on Surface and 1 x Incline CV 1,617m from -270 to -39 Surface)	4,824,017
Truck Unloading & Crushing	8,240,000
Total	**44,824,692**

4.2.2 Operating Costs

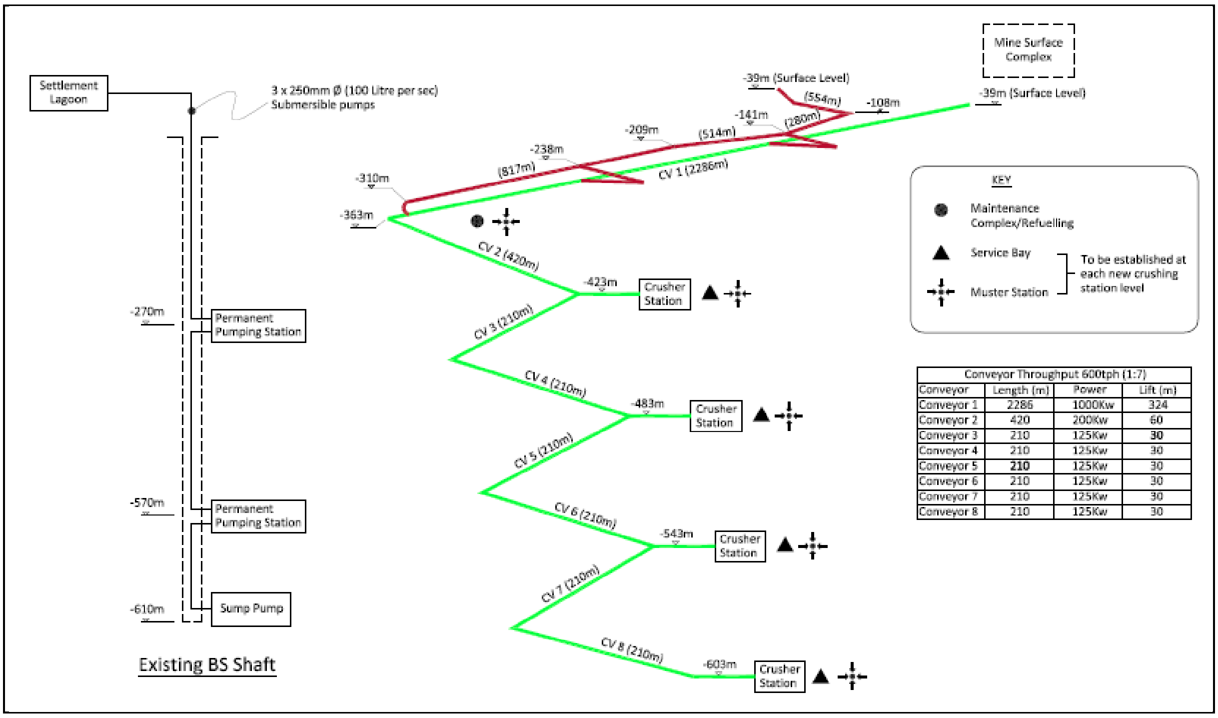
The operating costs of the equipment required are summarised as follows.

Table 4-3 Hoist/Conveyor Option Operating Costs

Item	Annual Cost €
Power	374,436
Materials	570,051
Other (vehicles etc.)	7,927
Total	**1,764,413**

5 MOVABLE CRUSHER/CONVEYOR OPTION

This option review incorporates a main conveyor installed in the decline down to the 365 level and considers the potential for installing a movable crusher facility. It is proposed to operate a mining system which will establish a crusher facility at 60m (depth) intervals and convey the ROM product by conveyor installations to the 365 level where the mineral will be transferred and conveyed to the surface on the main conveyor. The movable crusher design and associated logistics have not currently been finalised; however a schematic layout below is representative of the planned mining activities which will utilize the system.



Conveyor Throughput 600tph (1:7)			
Conveyor	Length (m)	Power	Lift (m)
Conveyor 1	2286	1000Kw	324
Conveyor 2	420	200Kw	60
Conveyor 3	210	125Kw	30
Conveyor 4	210	125Kw	30
Conveyor 5	210	125Kw	30
Conveyor 6	210	125Kw	30
Conveyor 7	210	125Kw	30
Conveyor 8	210	125Kw	30

Figure 3 Estimated hoist time per cycle

5.1 Conveyor Capacity

The design capacity of the conveyor systems which would need to be installed as the mining operations descend (60m) and assumes an annual throughput from the underground mine of 3,500,000 tonnes per year (3,000,000tonnes of Ore and 500,000tonnes of waste).

5.1.1 Average throughput for ore and waste

Operating Hours per day = 3 shift at 7 hours per shift = 21 hours/day

For conveyor operation assume 350 days per year.

Therefore annual hours = 350 x 21 = 7350hrs/year

Total Tonnage transported = 3,000,000 + 500,000 tonnes / year

Therefore average capacity = 3,500,000 / 7350 **= 476tph**

Assume a combined conveyor and crusher availability = 85%

Therefore design capacity = average throughput / availability

= 476 / 0.85 = **560tph**

5.1.2 Main Conveyors

The basic parameters for a crusher/conveyor option would be:

Table 5-1 Conveyor parameters

Parameter		Leg 1	Leg 2	Leg3	Leg4	Leg 5	Leg 6	Leg7	Leg8
Length	m	2286	420	210	210	210	210	210	210
Lift	m	324	60	30	30	30	30	30	30
Belt width	mm	1050	1050	1050	1050	1050	1050	1050	1050
Belt Spec		ST2500	Type8	Type8	Type8	Type8	Type8	Type8	Type8
Belt Speed	m/s	3	3	3	3	3	3	3	3
Power	kW	1000	200	125	125	125	125	125	125

5.1.3 Ore & Waste Belt Conveyor Loading

It is anticipated that a movable crusher installation would receive ore and waste rock from the ore and waste passes direct or by truck. The final arrangement for this will be specified on completion of the mine plan and will identify the type of crusher, mine layout and location of any ore and waste passes required. This truck unloading & crushing application will be specified as part of the mine plan.

It is recommended to crush to at least <250mm to reduce impact loads on the conveyor belting. Note: The 1050mm wide belt adopted on the decline conveyors could accommodate lump sizes up to 300mm. The general arrangement drawing below illustrates the type of crusher facility which will be required for the proposed type of mining operation.

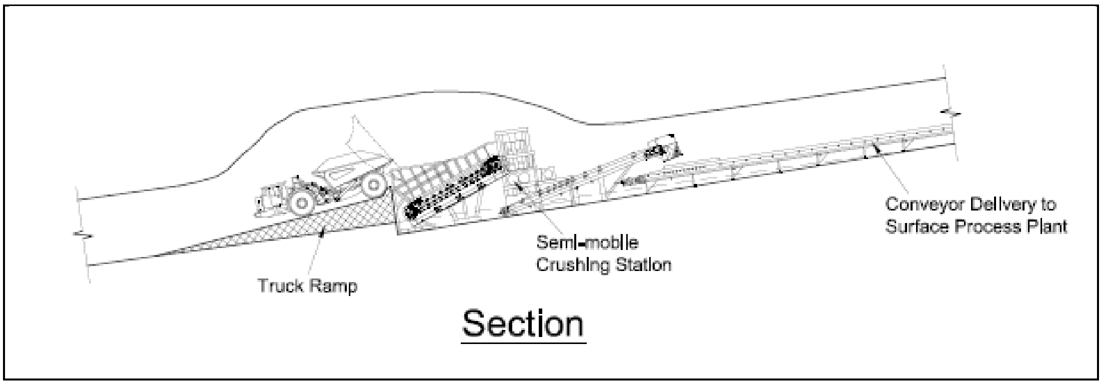


Figure 4 Arrangement layout of a crusher installation

5.2 Cost Analysis of Movable Crusher/Conveyor Option

5.2.1 Capital Costs

5.2.1.1 Equipment

The capital costs of the equipment required are summarised as follows.

Table 5-2 Conveyor Option Capital Equipment Costs

Item	Total Cost €
Truck unloading & crushing	1,969,875
Apron Feeders	346,650
Incline Conveyors	11,862,600
Foundations etc.	65,900
Electrical	670,650
Service Vehicles etc.	129,650
Total	**15,032,988**

5.2.1.2 Mining

Costs have been estimated for a solution considered practical and feasible. Further detailed design work must be undertaken to ascertain the optimum layout of roadways and excavations to accommodate the arrangement of equipment required.

The Mining Costs are summarised as follows based on a ramp length of 3,885m:

Table 5-3 Conveyor Option Capital Mining Costs

Item	Total Cost €
Pass-byes	55,620
Turn-a-rounds	49,440
Total	**105,060**

5.2.2 Operating Costs

The operating costs of the equipment required are summarised as follows.

Table 5-4 Conveyor Option Operating Costs

Item	Annual Cost €

Power	669,690
Materials	1,177,297
Labour	699,000
Other (vehicles etc.)	7,927
Total	**2,553,914**

6 COST ANALYSIS

The alternative haulage and operating options identified for Task 1 of the haulage review included a hoist/conveyor, conveyors, a number of truck configurations and a movable crusher facility which would utilize conveyors to transport the ROM to the surface.

A cost-effective review has been carried out for each of the estimated financial alternative haulage options with a summary of the financial data for the economic analysis presented later in the back of this document. The cost evaluation involved determining the Opex per tonne and the total costs (Opex and Capex) per tonne for each of the haulage options based on an annual production of 3,000,000 tonnes of ore and 500,000 tonnes of waste throughout the life of mine (14 years). These two basic cost parameters should help to assist and identify the most cost effective option of hauling the ore out of the mine.

6.1 Alternative Capex Costs

The Capex costs for the various haulage options can have a significant impact on the overall cost to a mining operation. This is particularly the case concerning pre-production Capex costs which can negatively impact upon the final Net Present Value (NPV) of a potential mining operation to the point where it may become unprofitable. Furthermore sustaining CAPEX costs associated with maintaining the mining operation throughout the LOM may necessitate buying new equipment or extensive maintenance to existing and aging machinery and can also negatively impact upon the annual life of mine cash flows.

The Table 6-1 Capex Costs for the Various Haulage Options below identifies the total estimated pre-production Capex and sustaining Capex figures used to provide the cost comparison for the LOM.

Table 6-1 Capex Costs for the Various Haulage Options

Haulage Options	Capex(€)		
	Pre-production	Sustaining	Total
Diesel – 50t trucks (10)	20,343,174	10,439,216	30,782,390
Electric – 50 t trucks (6)	31,662,874	16,249,716	47,912,590
Diesel – 40t trucks (12)	20,446,174	20,584,716	41,030,890
Electric – 35t trucks (9)	37,533,874	22,120,716	59,654,590
Hoist/Conveyor	44,824,692	480,000	45,304,692
Conveyor	20,032,225	4,536,000	24,568,225
Movable Crusher/Conveyor	15,138,048	10,520,486	25,658,533

6.2 Alternative Opex Costs

The estimated Opex costs listed below are based on some operational assumptions throughout the LOM and may change if the mining operational plan varies.

Table 6-2 Opex Costs for the Various Haulage Options

Haulage Options	Total Opex(€)
Diesel – 50 t trucks	119,500,958
Electric – 50 t trucks	60,475,252
Diesel – 40t trucks	140,773,683
Electric – 35t trucks	76,611,855
Conveyor	34,751,806
Hoist/Conveyor	24,701,787
Movable Crusher/Conveyor	35,754,794

Table 6-2 Opex Costs for the Various Haulage Options

Haulage Options	Total Opex(€)
Diesel – 50 t trucks	119,500,958
Electric – 50 t trucks	60,475,252
Diesel – 40t trucks	140,773,683
Electric – 35t trucks	76,611,855
Conveyor	34,751,806
Hoist/Conveyor	24,701,787
Movable Crusher/Conveyor	35,754,794

From the Opex costs per tonne of the alternative Haulage Options, figures presented in above, the most cost effective Opex estimate is the hoist/conveyor option. The truck haulage configuration review of the Opex costs per tonne of ore hauled compares the various truck configurations and the 50 tonne electric trucks is identified as the most cost effective option. The Opex estimate comprises of a significant proportion of the costs associated with each of the haulage options including the variations and subsequently the profitability of a mining operation. These costs are always present throughout the life of a mine and can vary significantly for the various options. As they will be present throughout the 14 years of the mine's life there is also the potential for variations in costs due to fluctuations in prices for labour, parts, fuel etc. Labour costs at this stage of the study have been excluded.

6.3 Total Capex & Opex Costs

The Total Opex and Capex presented in Table 6-2 indicate that the conveyor is the option with the least costs. No contingency has been included at this stage for the estimated figures supplied.

Table 6-3 Total Capex and Opex for Various Haulage Options

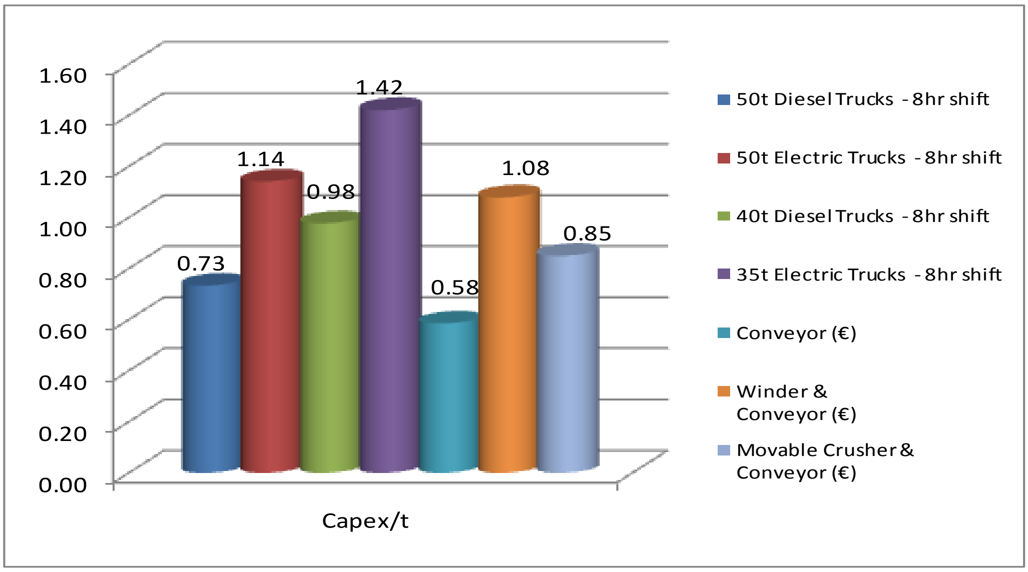
Haulage Options	Total Opex(€)	Total Opex &Capex(€)
Diesel – 50 t trucks	119,500,958	150,283,348
Electric – 50 t trucks	60,475,252	108,387,841
Diesel – 40t trucks	140,773,683	181,804,572
Electric – 35t trucks	76,611,855	136,266,444
Conveyor	34,751,806	59,320,031
Hoist/Conveyor	24,701,787	70,006,479
Movable Crusher/Conveyor	35,754,794	61,413,327

6.4 Capex, Opex & Total Costs/tonne

The cost/tonne estimates for the Capex, Opex and Total are presented in graphical form below. The data used to generate the cost results can be found in Table 6-7 Combined Financial Data for Various Haulage Options**Error! Reference source not found.**. No contingency has been included at this stage for the estimated figures supplied.

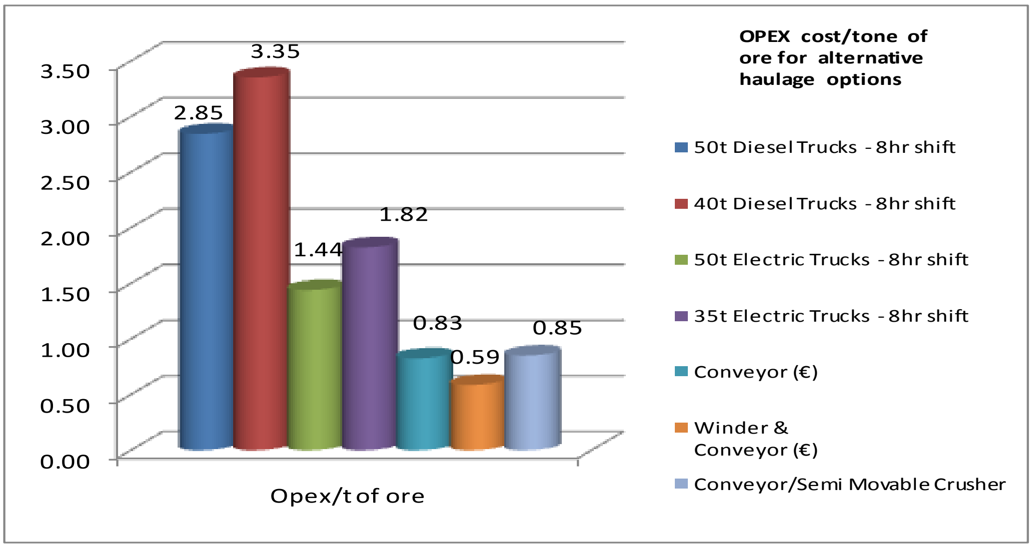
6.4.1 Capex Cost/tonne

Table 6-4 Total Capex/tonne Graph for Various Haulage Options



6.4.2 Opex Cost/tonne

Table 6-5 Total Opex/tonne Graph for Various Haulage Options



6.4.3 Total Cost/tonne

Table 6-6 Total Cost/tonne Graph for Various Haulage Options

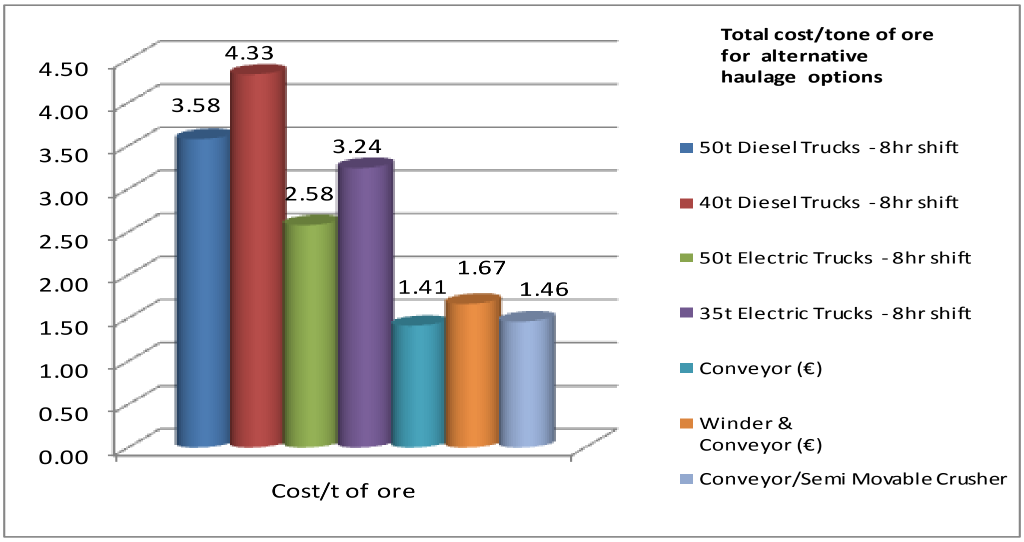


Table 6-7 Combined Financial Data for Various Haulage Options

Haulage Study Alternatives	50t Diesel Trucks - 8hr shift	50t Electric Trucks - 8hr shift	40t Diesel Trucks - 8hr shift	35t Electric Trucks - 8hr shift	Conveyor (€)	Winder & Conveyor (€)	Movable Crusher & Conveyor (€)
No Trucks	10	6	12	9	N/A	N/A	N/A
Pre-production Capital Cost	20,343,174	31,662,874	20,446,174	37,533,874	20,032,225	44,824,692	15,138,048
Production Capital Cost	10,439,216	16,249,716	20,584,716	22,120,716	4,536,000	480,000	10,520,486
Operating Cost	119,500,958	60,475,252	140,773,683	76,611,855	34,751,806	24,701,787	35,754,794
Total Capital & Operating Cost	150,283,348	108,387,841	181,804,572	136,266,444	59,320,031	70,006,479	61,413,322
Capex/t of ore	0.73	1.14	0.98	1.42	0.58	1.08	0.85
Opex/t of ore	2.85	1.44	3.35	1.82	0.83	0.59	0.85
Cost/t of ore	3.58	2.58	4.33	3.24	1.41	1.67	1.46

7 COMPARISON OF FINANCIAL OPTIONS

7.1 Trucks or Conveyor

The cost analysis has shown that the conveyor option is considerable cheaper than the truck option. The following table compares the base cases for both options.

Table 7-1 Combined Financial Cost/tonne for Various Haulage Options

Option	Cost/ t €
Trucks – 50t diesel driven	3.58
Trucks – 50t electric driven	2.58
Trucks – 40t diesel driven	4.33
Trucks – 35 electric driven	3.24
Conveyor	1.41
Hoist/Conveyor	1.67
Movable Crusher/Conveyor	1.46

From a cost perspective the use of various conveyor configurations for the duty prescribed is clearly the most cost effective option. Other alternatives for trucks involving changes to shift patterns and source of power for the trucks could be more cost effective than the selected case for the trucks; however this cost is still greater than the conveyor option.

7.2 Diesel or Electric Trucks

The analysis of costs has shown that there is significant difference between the use of an electric powered truck and a diesel driven truck. The electric trucks are more expensive to buy initially and install but the operating costs, particularly the energy, are very much reduced. If in the overall selection of the haulage option, the use of trucks is preferred to the installation of a conveyor system and the use of electric powered trucks should be given serious consideration.

7.2.1 Benefits of Electric Trucks compared with Diesel Trucks

The following list highlights the benefits of using an electric truck:-

- Minimal diesel fumes and considerable reduction on ventilation systems
- Faster speeds, hence,
  - shorter cycle times
  - fewer trucks for same haulage
  - fewer drivers
- New technology such as regenerative braking down ramp adds power back to the network
- Long asset life approx. 60,000 hours

- Environmental, health and safety friendly
- Low noise level
- Low heat generation

Lower operating cost on a fleet by fleet basis (about half)

By using electric trucks rather than diesel savings could be made in the cost of providing the ventilation system of the mine.

7.3 Further Design Considerations

7.3.1 Development of the Decline

If the conveyor option is selected it is imperative that the decline is driven to the prescribed alignment and grade to enable the conveyor to be installed correctly. Any deviations must be controlled within acceptable limits.

This requirement is not as critical for the truck option as variations in alignment and grade caused by problems during the development of the decline can be more easily accommodated.

7.3.2 Design of the decline

The study has been based on a schematic layout of the decline discussed with DMT and NIO.

8 TASK 2

In addition to the alternative haulage options, DMT reviewed the potential declines configurations for the haulage options and arrangements to access the mining operation.

The following options were considered as part of the mine access development;

8.1 Two Tunnels (two lane traffic all the way & one lane with passing bays)

The schematic layout below indicates the proposed arrangement to access the underground workings. The deviation in the access tunnel direction may allow NIO access to some ore bodies whilst developing the mine.

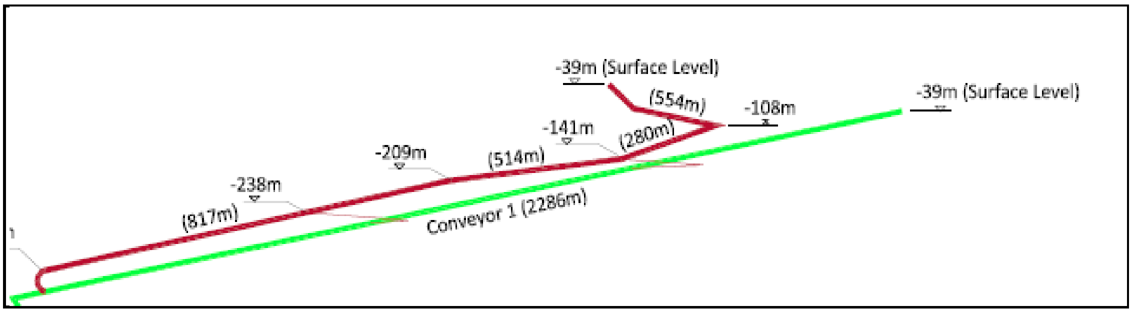


Figure 5 Initial arrangement layout access tunnels

8.1.1 Access Tunnel

The access tunnel would be driven at a cross sectional area of 36m^2 to facilitate the use of 50tonne trucks which could be utilised in the mine development and operations. Additionally all operational and service traffic inclusive of twin boom machines, shot-crete sprayers will need to access the underground workings.

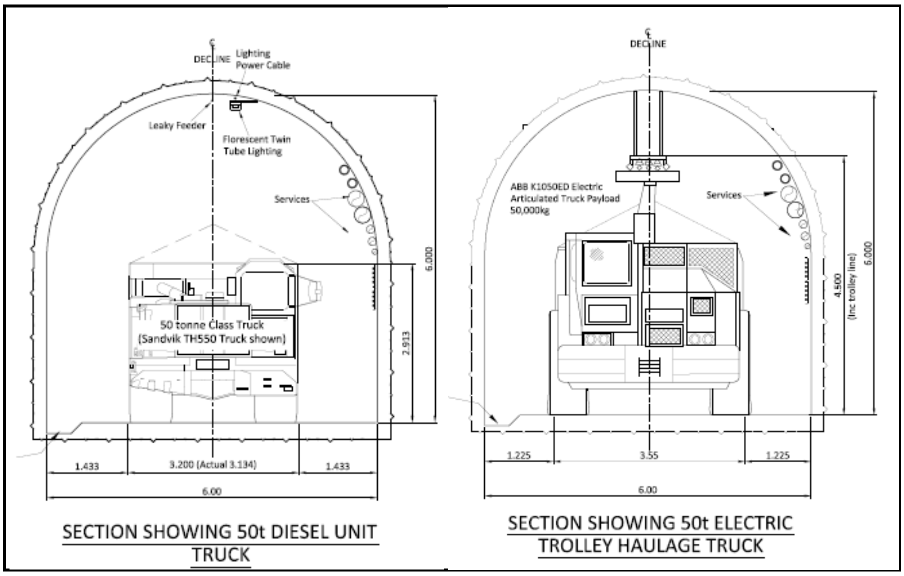


Figure 6 Schematic arrangement layouts of access tunnel and 50 tonne trucks

The general arrangement drawings above show a typical tunnel cross section (36m^2) utilising diesel or electric 50 tonne trucks. If the smaller trucks (40tonne) were considered, the design of the tunnels could be reduced; however no confirmation of the ventilation demands for the mine have been supplied.

8.1.1.1 Drivage Costs

The following table represents the estimated development costs for the road way cross section specified.

Table 8-1 Drivage Cost for Access tunnel option

Blotberget CAPEX Drivage Costs (€2,400/m)				
			Unit Cost	
			Mkr	**€**
(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Access Ramp (36m²) -39 to -310			**Mkr**	**€**
Cost/Metre €		2400		
Ramp 1 -39 Surface to -108	1in8	554	12,381,089	1,329,600
Ramp 2 -108 to -141	1in8	280	6,257,590	672,000
Ramp 3 -141 to -209	1in8	514	11,487,148	1,233,600
Ramp 4 -209 to -310	1in7	817	18,258,754	1,960,800
X Slit 1 at -141 level		141	3,151,144	338,400
X Slit 2 at -238 level		324	7,240,926	777,600
Total		***2630***	***58,776,650***	***6,312,000***
Contingency (15%)			8,816,497	946,800
Total			**67,593,147**	**7,258,800**

A contingency of 15% has been added to cover possible variations in contractor prices.

8.1.2 Conveyor Tunnel

The conveyor tunnel would be driven at a cross sectional area of 36m^2 to allow the installation of the planned conveyor and also allow service vehicles to operate in the tunnel during the mine development and operations. The cross section specified is to allow development equipment to operate during the construction of the tunnel.

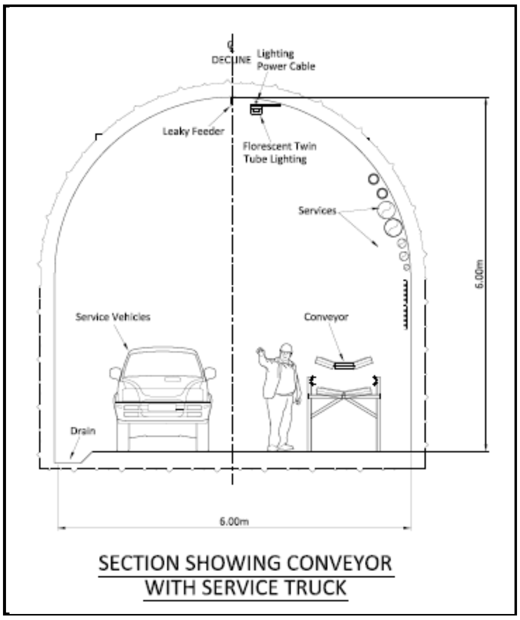


Figure 7 Schematic arrangement layouts of conveyor tunnel and service vehicles

8.1.2.1 Drivage Costs

The following table represents the estimated development costs for the road way cross section specified.

Table 8-2 Drivage Cost for Conveyor tunnel option

Blötberget CAPEX Drivage Costs (€2,400/m)				
			Unit Cost	
			Mkr	**€**
(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Conveyor Tunnel (36m²) -39 to -365			**Mkr**	**€**
Cost/Metre €		2400		
Ramp 1 -39 Surface to -365	1in7	2268	50,686,479	5,443,200
Total		***2268***	***50,686,479***	***5,443,200***
Contingency (15%)			7,602,971	816,480
Total			**58,289,450**	**6,259,680**

8.1.3 Single Tunnel with two lane access all the way surface to -365

The schematic tunnel for a two way operation is shown driven at a cross sectional area of 54m^2 to allow the operation of two way traffic in the tunnel. It will also allow service vehicles to operate in the tunnel during the mine development and operations.

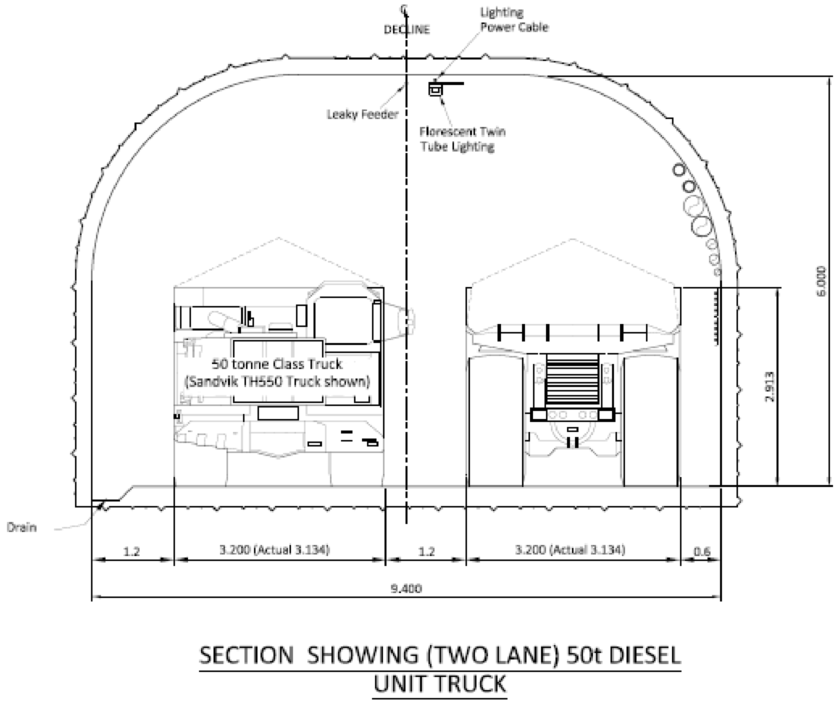


Figure 8 Schematic arrangement layouts of two way vehicles

The above layout for the two way operation of vehicles is representative of diesel or electric trucks. No pass byes would be required for the trucks or service vehicles. The increased cross section may be necessary to supply sufficient ventilation based on a single tunnel operation.

8.1.3.1 Drivage Costs

The following table represents the estimated development costs for the road way cross section specified.

Table 8-3 Drivage Cost for Two Lane tunnel option

Blötberget CAPEX Drivage Costs (€2,400/m)				
			Unit Cost	
			Mkr	**€**
(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Two Way Access Ramp (54m²) -39 to -365			**Mkr**	**€**
Cost/Metre €		3165		
Ramp 1 -39 Surface to -365	1in7	2268	66,842,795	7,178,220
Total		***2268***	***66,842,795***	***7,178,220***
Contingency (15%)			10,026,419	1,076,733
Total			**76,869,214**	**8,254,953**

8.2 Single Tunnel (accommodating a conveyor, one lane for trucks with passing bays)

A single conveyor tunnel driven at a cross sectional area of 36m^2 has been assumed as a minimum for the ventilation requirements of the mine. Based upon the design of the tunnel, it could allow the installation of the planned conveyor and also allow truck vehicles with pass byes to operate in the tunnel during the mine development and operations. The cross section specified is to allow development equipment to operate during the construction of the tunnel and may change with the final mine plan.

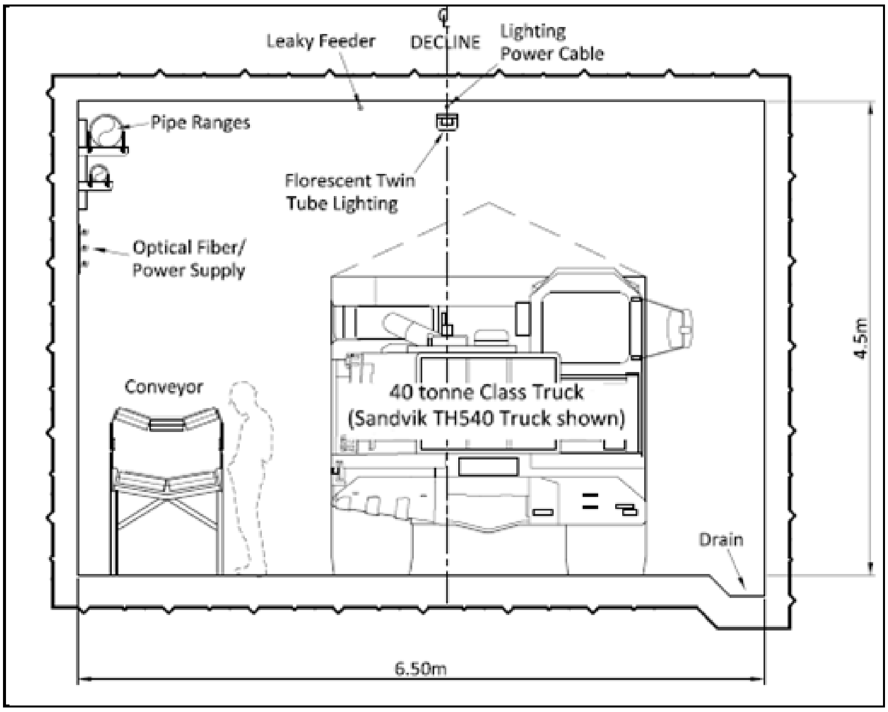


Figure 9 Schematic arrangement layouts of conveyor tunnel and truck vehicles

The above layout demonstrates that a conveyor and truck can operate in the same cross sectional area (36m^2) as the previous alternatives with a rectangular tunnel rather than the conventional arch section.

8.2.1.1 Drivage Costs

The following table represents the estimated development costs for the road way cross section specified.

Table 8-4 Drivage Cost for Conveyor/Truck tunnel option

Blötberget CAPEX Drivage Costs (€2,400/m)				
			Unit Cost	
			Mkr	**€**

(1€ = 9.31189kr)	***Gradient***	***Quantity (m)***	9.31189	
Conveyor Tunnel (36m²) -39 to -365			**Mkr**	**€**
Cost/Metre €		2400		
Ramp 1 -39 Surface to -365	1in7	2268	50,686,479	5,443,200
Total		***2268***	***50,686,479***	***5,443,200***
Contingency (15%)			7,602,971	816,480
Total			**58,289,450**	**6,259,680**

The drivage costs are calculated at the same cross section as the conveyor/service vehicle option although the tunnel profile has changed.

9 CONCLUSIONS AND RECOMMENDATIONS

The haulage study report has assumed that both hoist/conveyor, conveyors, trucks and a movable crusher facility system are feasible and can be used for the mining and transportation of ore and waste out of the mine.

The variable conveyor systems require an arrangement of conveyors and loading/discharge chutes at the transfer points and increased roadway size to accommodate the transfer arrangements.

The use of trucks would be more flexible and the production would be less susceptible to minor breakdowns. Trucks of 50 tonne capacity have been identified as the selected case and can be accommodated in the proposed roadway cross section. However, in the interests of safety, the trucks need exclusive use of the decline whereas the conveyor option would allow the decline to be used as a service road and consideration could be given to operating trucks and conveyors in the same roadway.

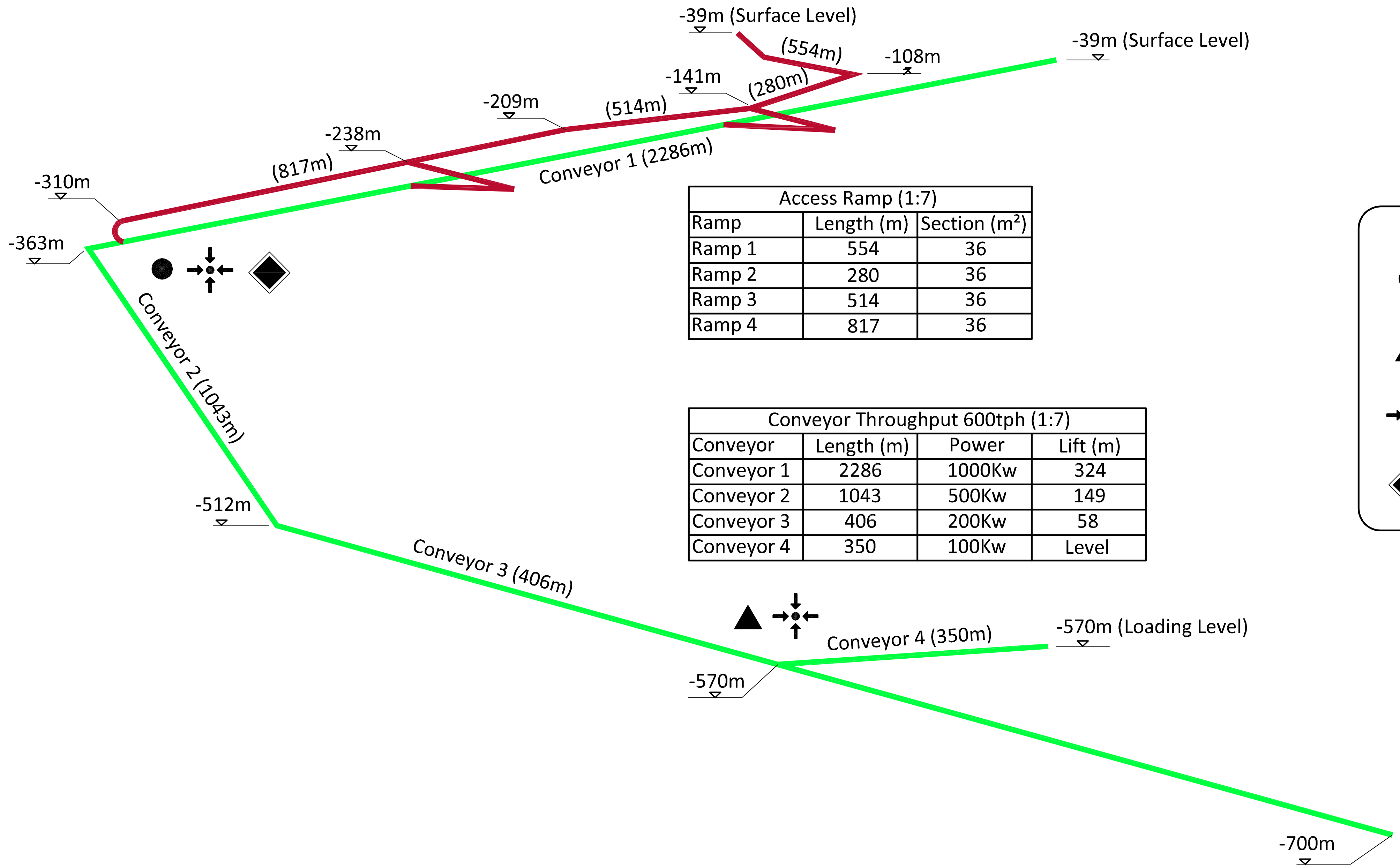
The initial cost analysis has shown that the use of conveyors for the transport of ore and waste from the mine is more cost effective capital cost and operational costs than the use of a fleet of trucks or utilising a combined hoist and conveyor system.

With the truck option, analysis of both diesel driven trucks and electric powered trucks has been considered. The electric trucks are cheaper to operate and also more beneficial to the working environment. If the use of trucks is the preferred option, the application of electric trucks may be considered at the procurement stage.

The application of a movable crusher facility is a viable option and has been considered; however some of the financial costs are based on estimates of other crusher applications and may change when the final specification has been agreed. The movable crusher option still requires the operational aspects of delivering the required tonnages per hour to the location and will be covered as part of the overall mine plan.

APPENDIX A

Schematic Layout Hoisting Options



Access Ramp (1:7)		
Ramp	Length (m)	Section (m²)
Ramp 1	554	36
Ramp 2	280	36
Ramp 3	514	36
Ramp 4	817	36

Conveyor Throughput 600tph (1:7)			
Conveyor	Length (m)	Power	Lift (m)
Conveyor 1	2286	1000Kw	324
Conveyor 2	1043	500Kw	149
Conveyor 3	406	200Kw	58
Conveyor 4	350	100Kw	Level

KEY

- Maintenance Complex/Refuelling
- Service Bay
- Muster Station
- Cap Magazine Store

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Earth. Insight. Values

Project: Nordic Iron Ore™
Blötberget Iron Ore Project
Definitive Feasibility Study

Drawing Title:
Schematic Layout of Hoist/ Conveyor & Truck Option

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Nov 14				

Project No.	Drawing No.	Revision
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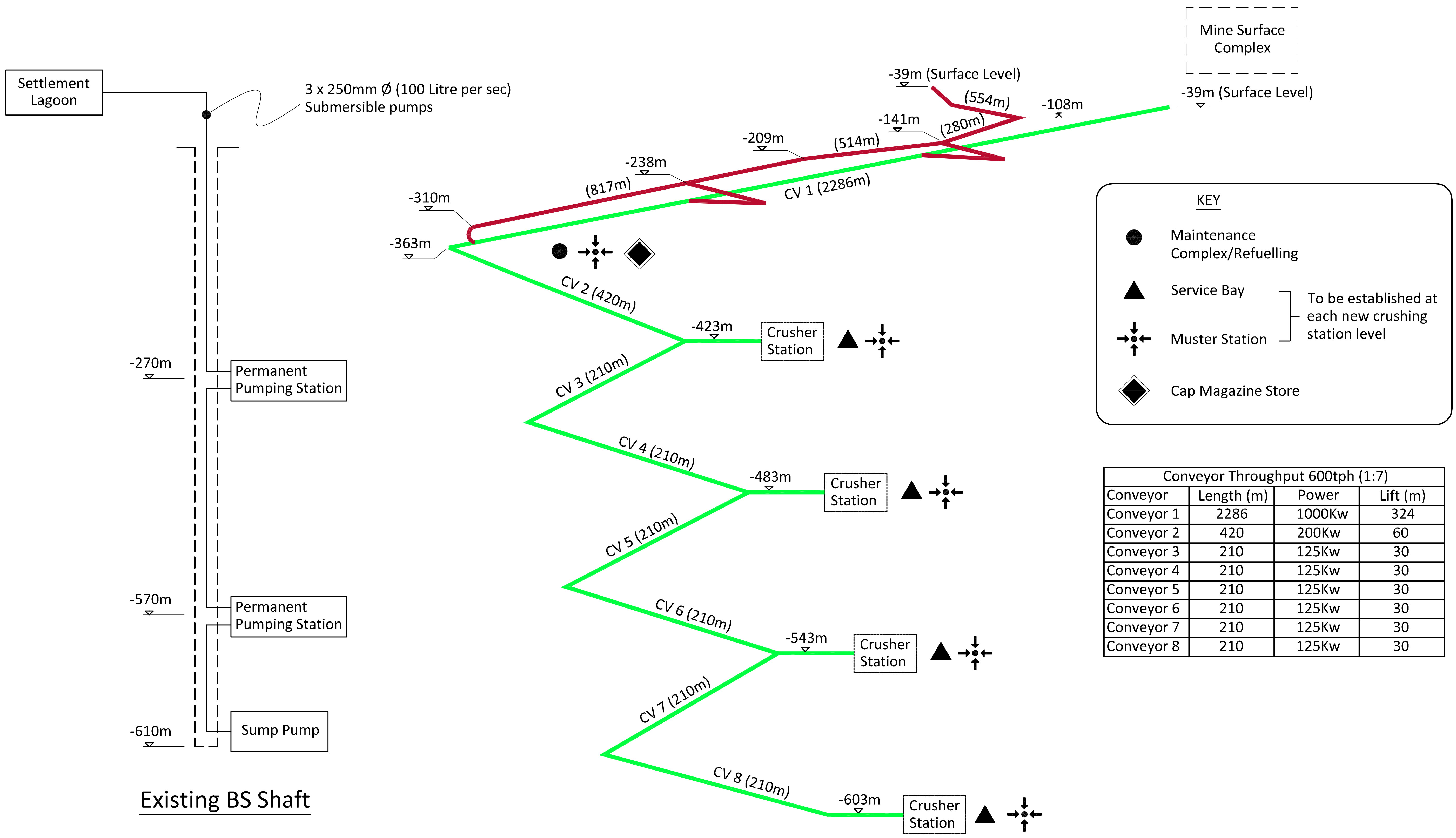

Conveyor Throughput 600tph (1:7)			
Conveyor	Length (m)	Power	Lift (m)
Conveyor 1	2286	1000Kw	324
Conveyor 2	420	200Kw	60
Conveyor 3	210	125Kw	30
Conveyor 4	210	125Kw	30
Conveyor 5	210	125Kw	30
Conveyor 6	210	125Kw	30
Conveyor 7	210	125Kw	30
Conveyor 8	210	125Kw	30

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Project: Nordic Iron Ore™
Blötberget Iron Ore Project
Definitive Feasibility Study

Drawing Title:
Schematic Layout of Movable Crushing Station

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Nov 14				

Project No.	Drawing No.	Revision
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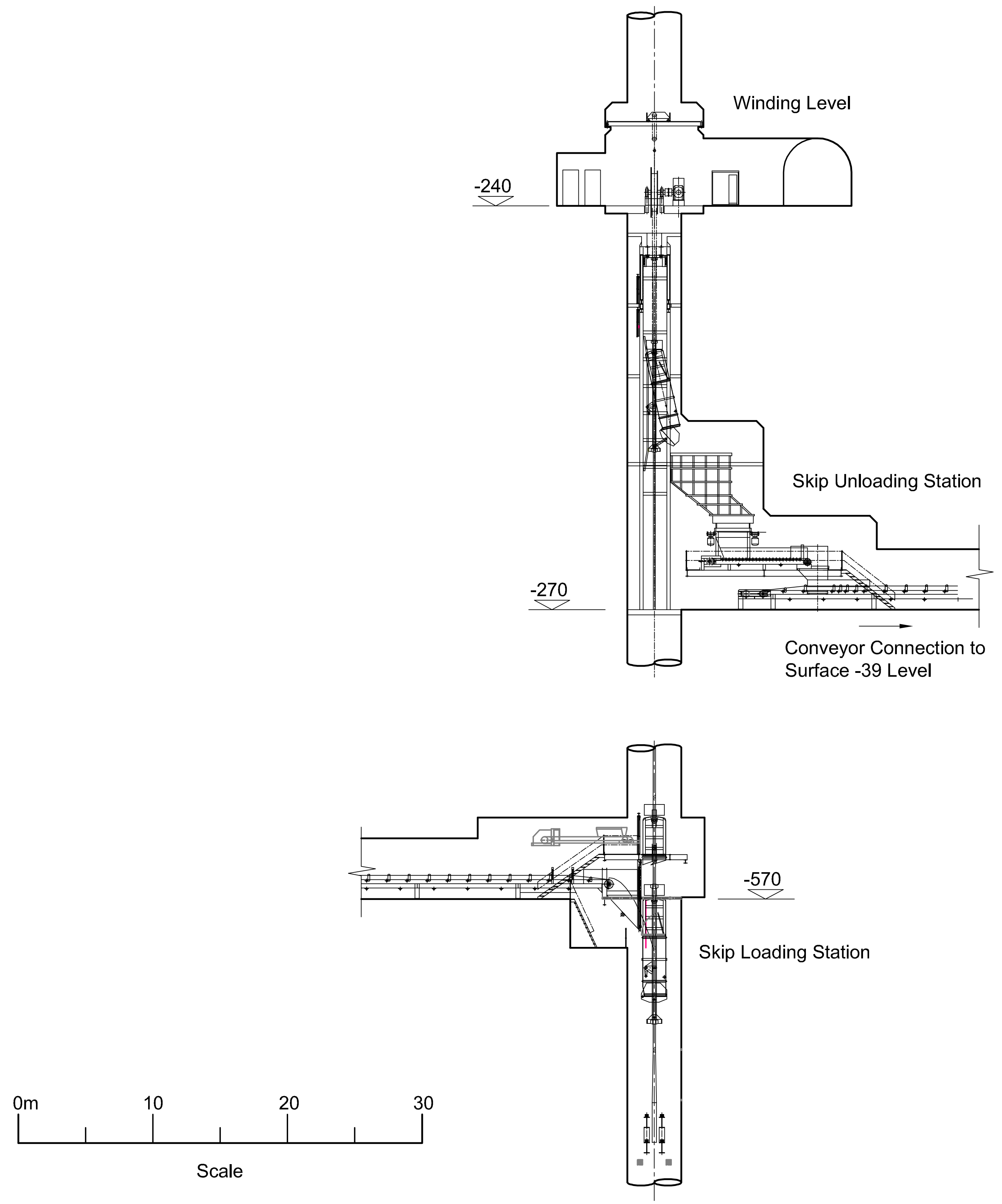

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Project: Nordic Iron Ore™

Blötberget Iron Ore Project
Definitive Feasibility Study

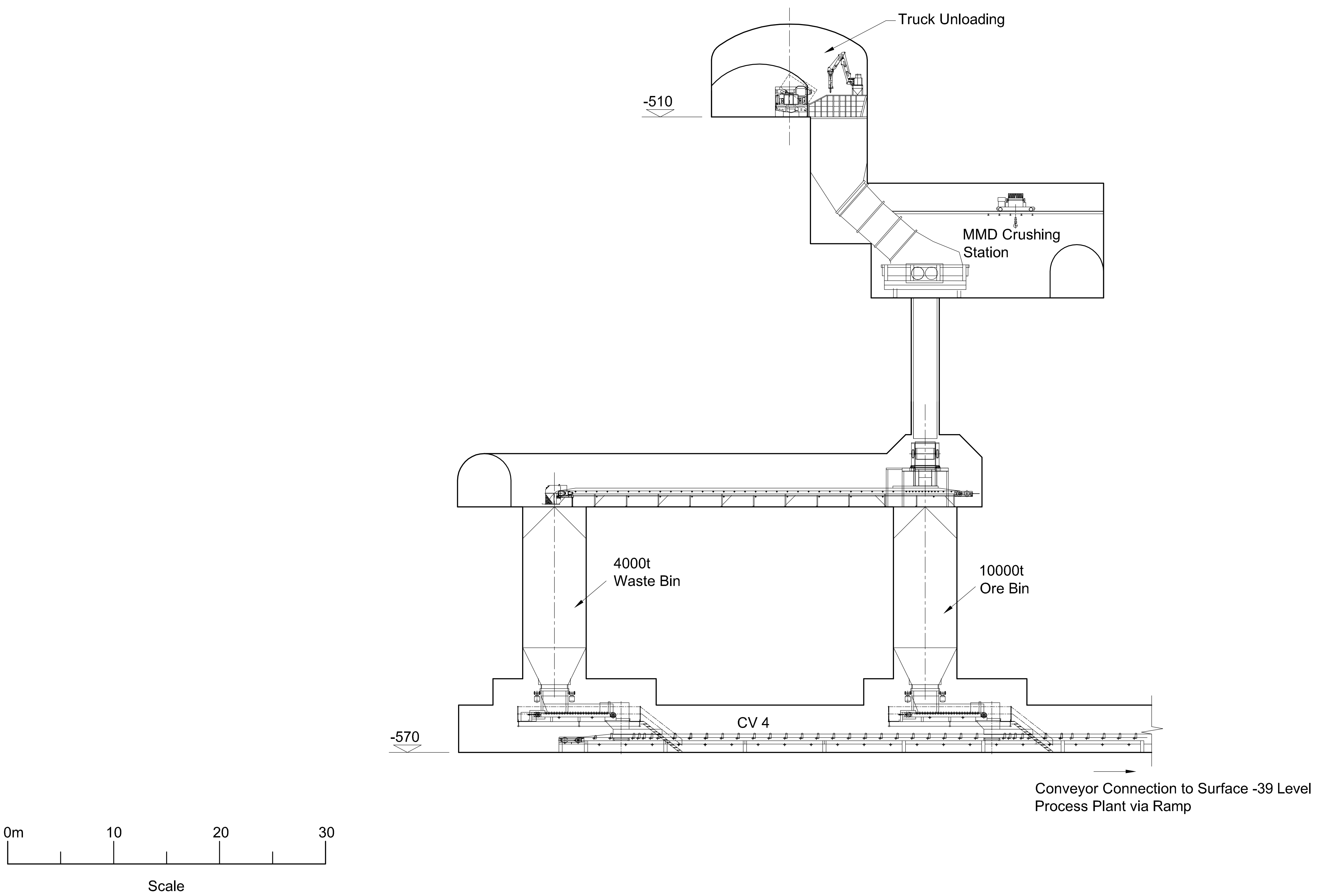
Drawing Title:

Alternative 1: Skip Hoisting in Existing BS Shaft with underground Inclined Conveyor -270 to Surface -39

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Nov 14				

Project No.	Drawing No.	Revision
C22	001	



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Project: Nordic Iron Ore™

Blötberget Iron Ore Project
Definitive Feasibility Study

Drawing Title:

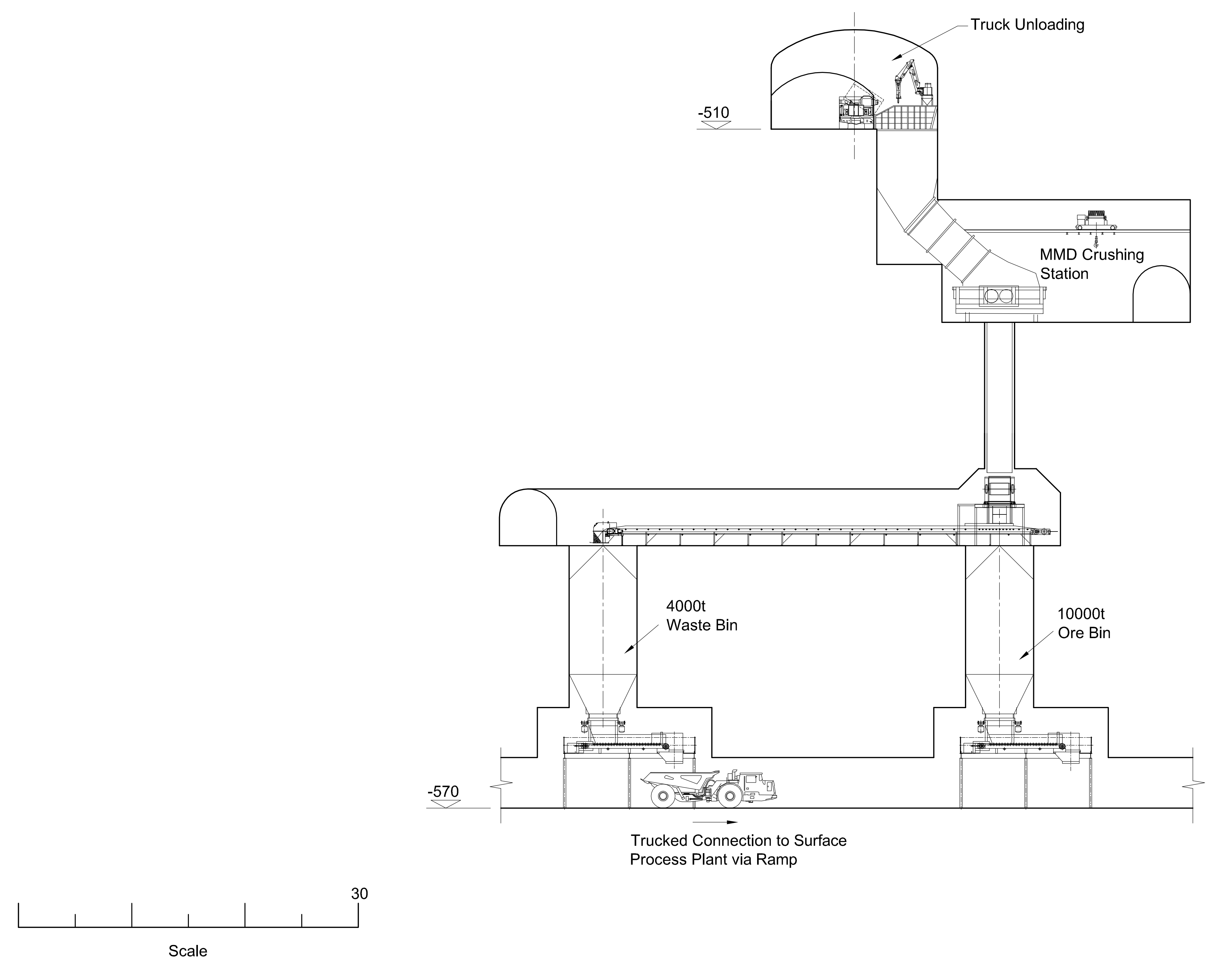
Alternative 2: Conveyor Hoisting from Existing Crusher Station at -510m level

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By / Date	Checked By / Date	Approved By / Date
N.T.S.	RS Nov 14		

Project No.	Drawing No.	Revision
C22	002	



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Project: Nordic Iron Ore™

Blötberget Iron Ore Project
Definitive Feasibility Study

Drawing Title:

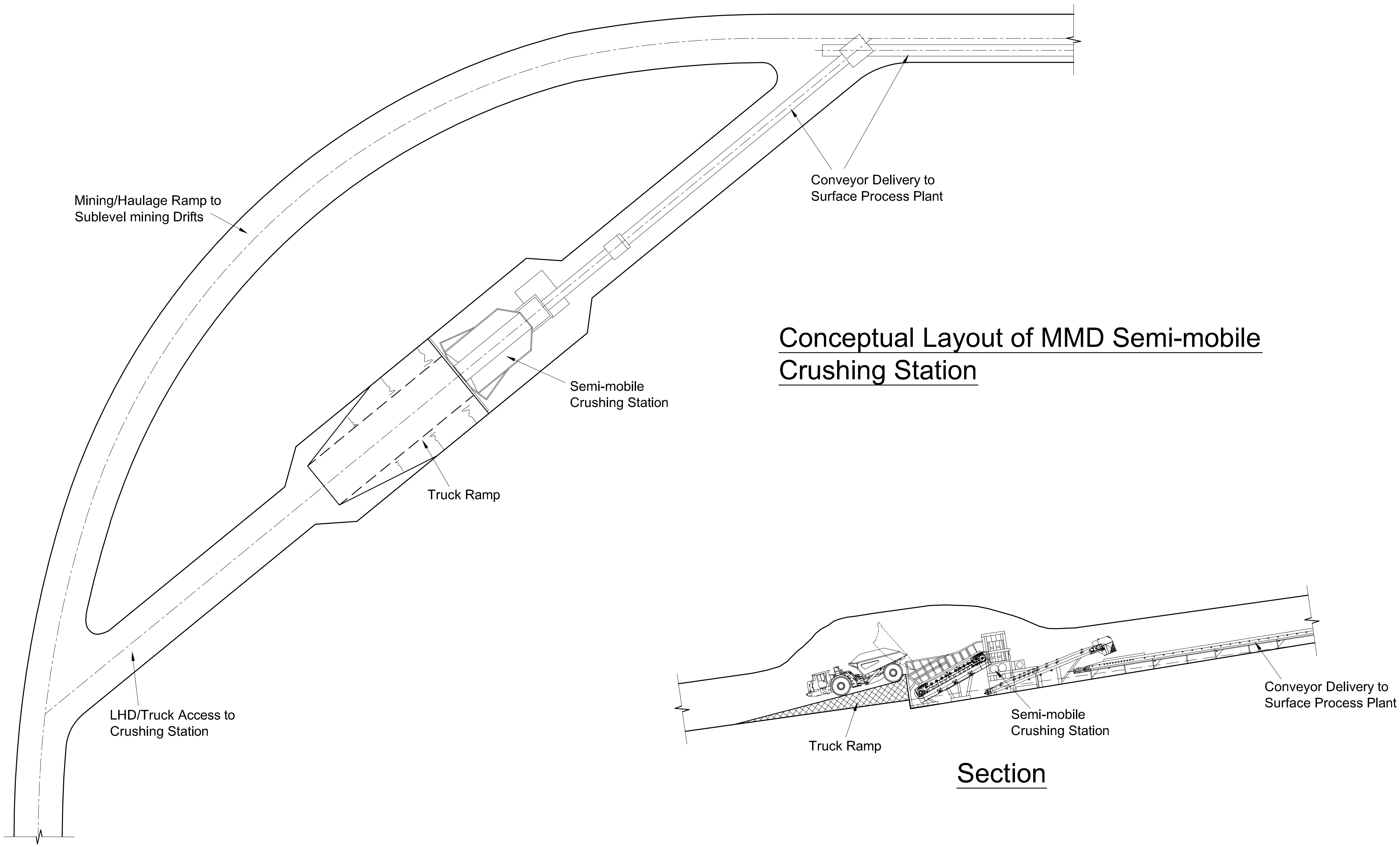
Alternative 3: Truck Hoisting from Existing Crusher Station on -570m Level

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By / Date	Checked By / Date	Approved By / Date
N.T.S.	RS Nov 14		

Project No.	Drawing No.	Revision
C22	003	



Conceptual Layout of MMD Semi-mobile Crushing Station

Section

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Project: Nordic Iron Ore™ Blötberget Iron Ore Project Definitive Feasibility Study

Drawing Title: Alternative 4: Semi-Mobile Crushing Station

Rev	Description	By	Chk	App	Date
	Saved location path				

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S.	RS	Nov 14				

Project No.	Drawing No.	Revision
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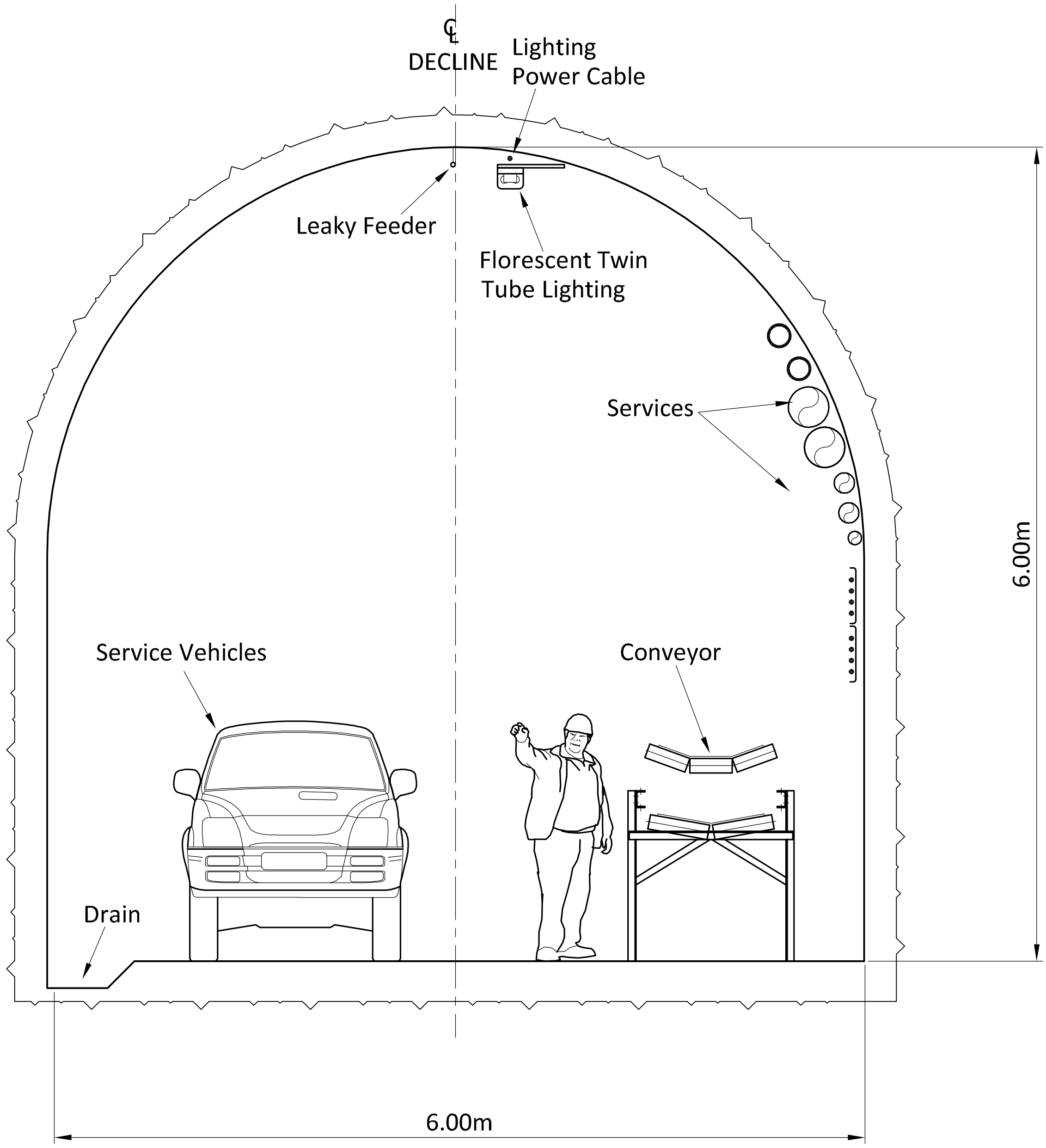

SECTION SHOWING CONVEYOR WITH SERVICE TRUCK

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	Saved location path					
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	Project No. C22		Drawing No. 006		Revision	

DMT

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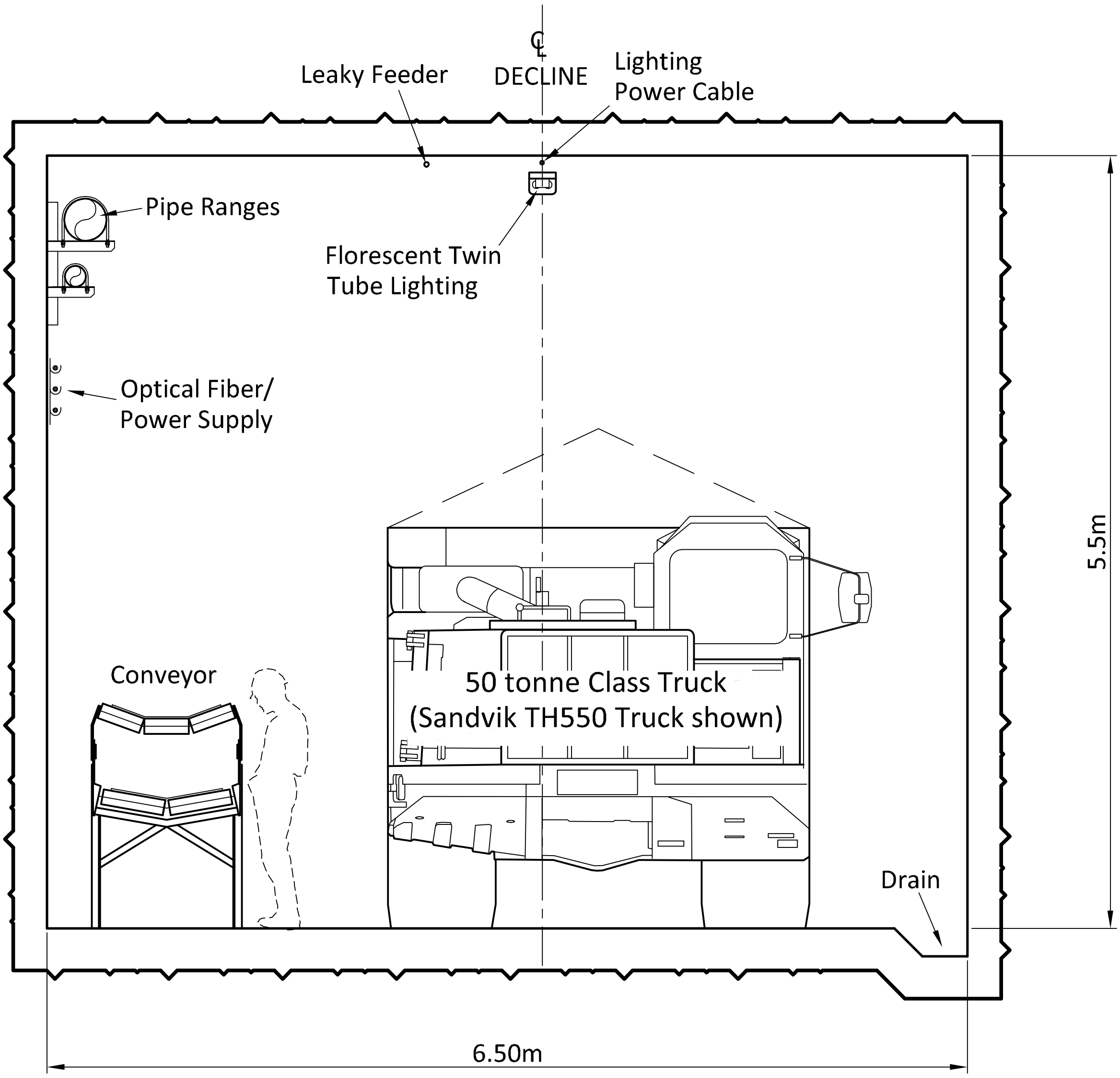

SECTION SHOWING CONVEYOR WITH SERVICE TRUCK

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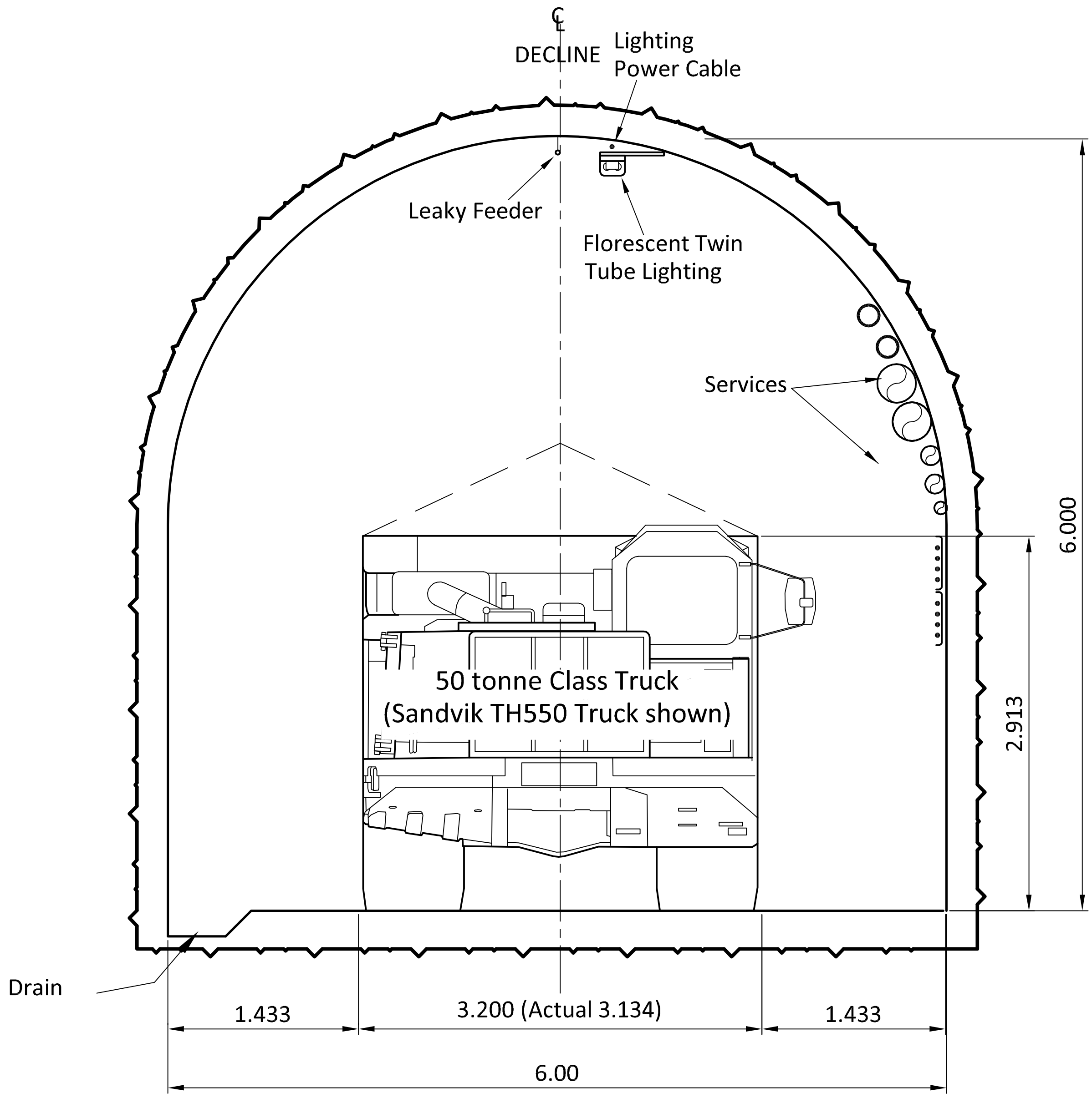
Project:
Nordic Iron Ore,
Blotterget Mine,

Drawing Title:
Typical Decline Section
6.5m x 5.5m

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S						

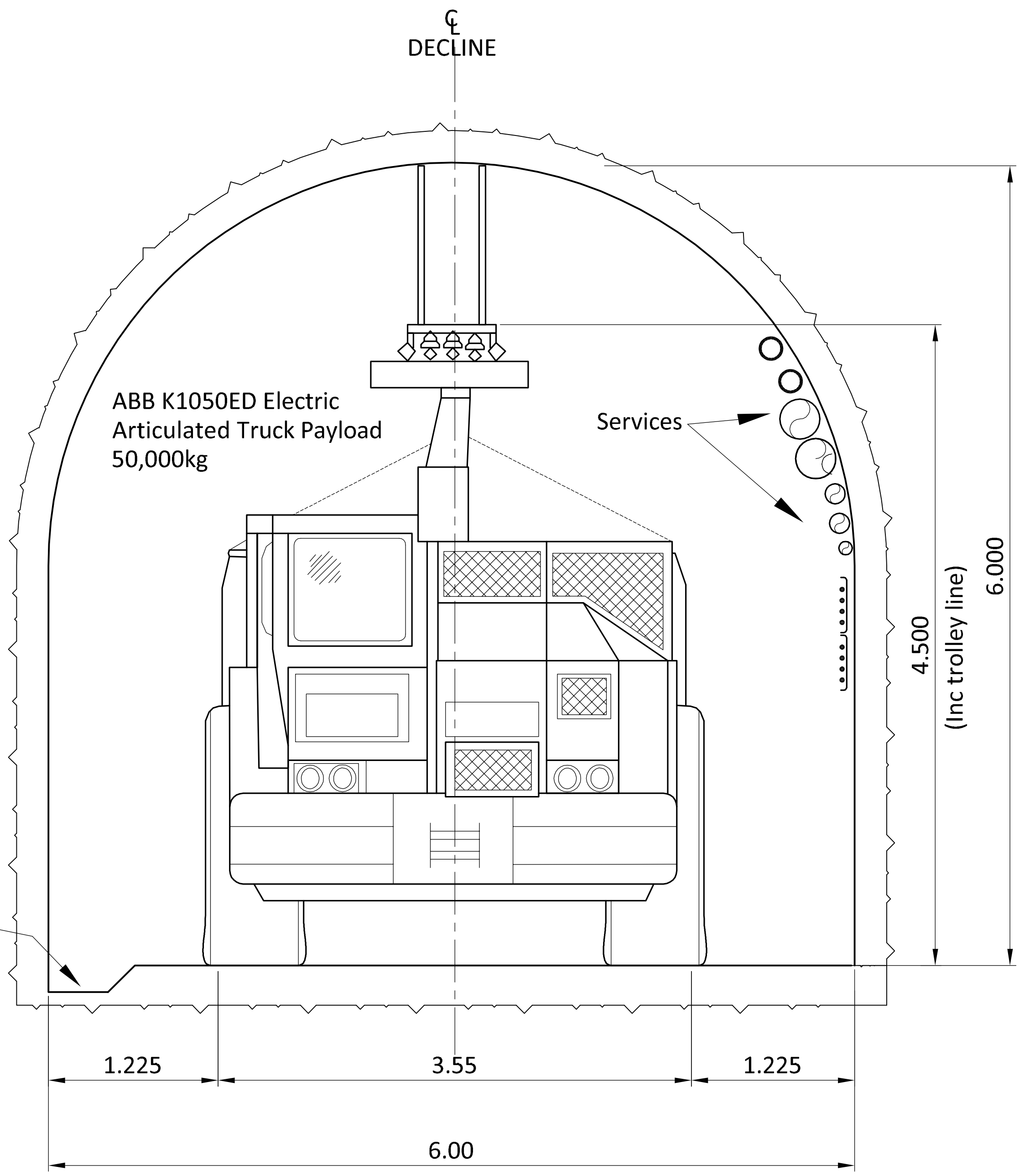
Project No.	Drawing No.	Revision
C22	011	



SECTION SHOWING 50t DIESEL UNIT TRUCK

DMT Earth. Insight. Values TUV NORD GROUP	Project: Nordic Iron Ore, Blotterget Mine,	Rev	Description	By	Chk	App	Date

Drawing Title:	Typical Decline Section 6.0m x 6.0m
Saved location path	
Scale at A3	N.T.S
Drawn By / Date	
Checked By / Date	
Approved By / Date	
Project No.	C22
Drawing No.	008
Revision	



SECTION SHOWING 50t ELECTRIC TROLLEY HAULAGE TRUCK

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Project:
Nordic Iron Ore,
Blotterget Mine,

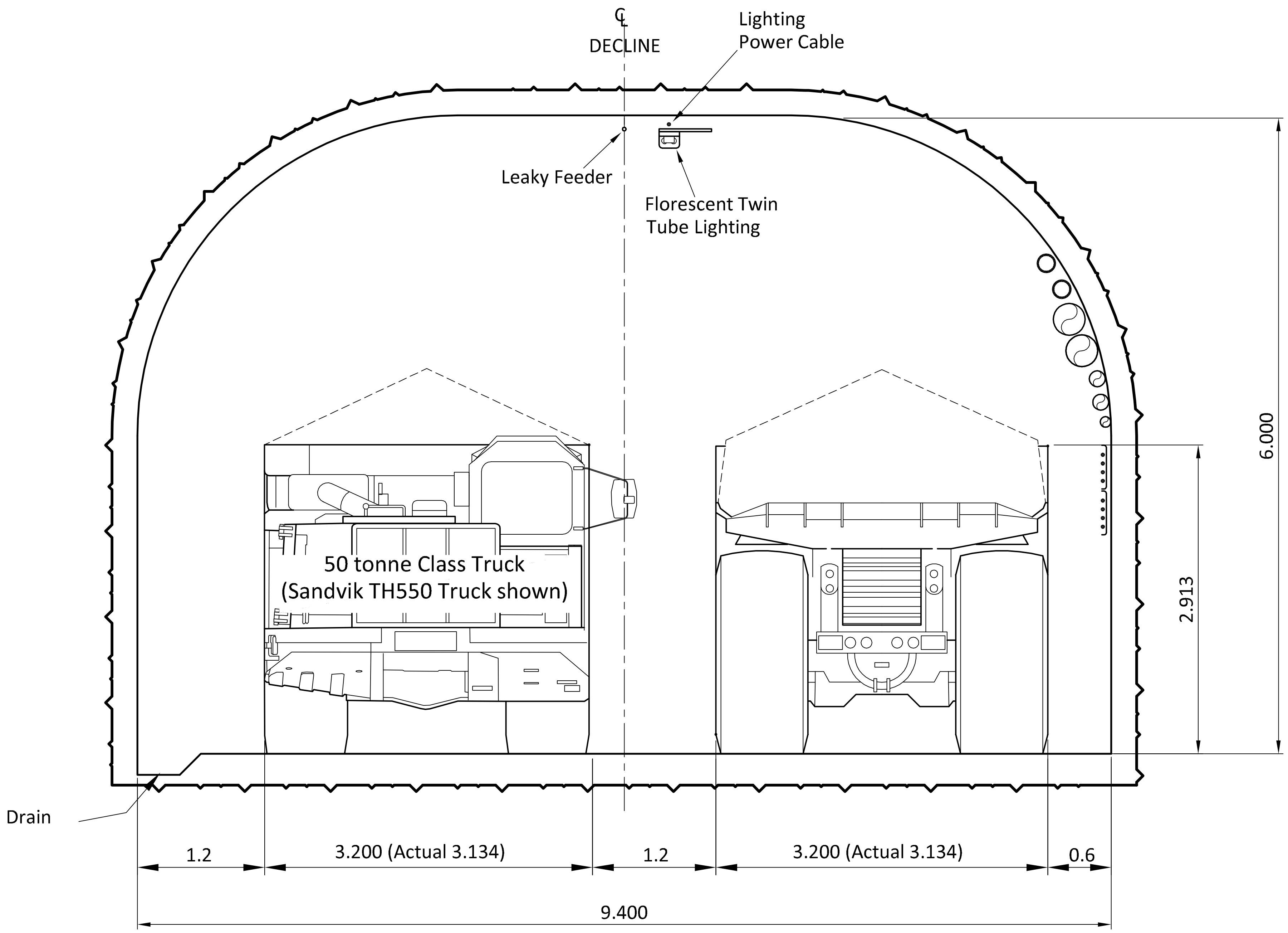
Drawing Title:
Typical Decline Section
6.0m x 6.0m

Rev	Description	By	Chk	App	Date

Saved location path

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date
N.T.S						

Project No.	Drawing No.	Revision
C22	007	



SECTION SHOWING (TWO LANE) 50t DIESEL UNIT TRUCK

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Project:
Nordic Iron Ore, Blotberget Mine,

Drawing Title:
Typical Section Two Lane Traffic

Rev	Description	By	Chk	App	Date
Saved location path					

Scale at A3	Drawn By	Date	Checked By	Date	Approved By	Date

Project No.	Drawing No.	Revision
C22	009	

Appendix H

TAILINGS STORAGE FACILITIES REPORT

Nordic Iron Ore AB

Technical report - Dams tailings and water

Version 1

Luleå, 2015-03-27

Technical report - Dams tailings and water

Date	
Project No.	1320011785
Version/Status	1

Lisa Fernius	Hans Häggström, Håkan Åkerlund	Klas Strömberg
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Executive summary

Nordic Iron Ore AB (NIO) has commissioned Ramböll AB (Ramböll) to update the conceptual design and cost estimate of the clarification pond and tailings management facility (TMF) dams previously developed part of the Preliminary Economic Assessment (PEA Blötberget and Håksberg) 2012 to a more detailed conceptual design and economical assessment part of the Technical Economical Study (pre-Feasibility level study) for the Blötberget iron ore mine near Ludvika in Sweden. An important prerequisite of the study set by NIO is to reduce the initial Capex costs.

The scope of work for this study comprises a more detailed conceptual design of the clarification pond and the TMF dams with associated structures (spillway, drainage ditches etc.) based on updated and more comprehensive basic design data and from this producing a more accurate economical assessment also based on more accurate cost unit data. The design of the dams is based on the more detailed conceptual design developed part of the environmental permit application process 2013.

The report and scope includes;

- design basic data (site geotechnical conditions and tailings characteristic properties)
- site water balance,
- deposition strategy, tailings and water management
- design and construction of the dams, and
- Bill of quantities and cost estimate for construction (CAPEX) and operation (OPEX).

Tailings slurry, containing a mix of water and tailing from the enrichment of the iron ore from the Blötberget underground mine in the process plant, is pumped to the tailings management facility (TMF). Sufficient volume is needed to store the tailing produced during life of mine.

The deposition strategy previously selected is based on conventional hydraulic deposition with segregation of the tailing. The coarse particles are settled near the spigots of a starter dam forming a beach and the finer particles are settled further away. The beach is used as part of raising the dam structures upstream reducing costs as much as possible for the available dam and pond foot prints. Thickened tailings deposition with higher beach slopes has also been considered as deposition strategy however this method is not practically usable since the tailing contain to low amount of fines giving it poor thickening properties.

To store all the tailing produced two different facilities are needed, here called northern TMF and southern TMF. The northern TMF has a capacity of around 5,5 (M)m^3 and the south TMF 12,5 (M)m^3. This give a total capacity of 18 (M)m^3 which

is more than the volume of tailing produced throughout life of mine (17 (M)m^3 as design amount).

Tailing drainage water, which is excess water drained when the tailing has been deposited, is pumped or diverted using a spillway to the nearby clarification ponds. Also site run-off and mine ground water are pumped to the clarification pond. The ponds need to have sufficient volume to store the water needed for production especially during winter and to attenuation higher flows during spring flood but, also large enough surface area to treat and remove sediments from the water diverted to the ponds. The clarification ponds consist of a smaller northern pond (volume around 150 000 m^3) and larger southern pond (volume around 1 000 000 m^3). The clean water from the ponds is pumped to the process plant and excess water discharged to Gonäsån.

Site geotechnical investigations have been done part of the permitting process. No additional investigations were made for this study. The area the dams will be located consist of peat and loose sediments in the wetland area next to the canal but otherwise moraine.

Tailing from the pilot plant operation have undergone test work to study rheological and physical properties. The tailings sample where slightly on the fine side in the pilot plant operation but can still be considered conservative. A sample was shipped to MRM lab n Luleå where lab tests were done. A segregation test was executed to produce coarse and fine tailing. Test on the total tailing showed target solids content of around 45 %-w/w to get good segregation of the tailing and still with as little water in the tailing as possible. The tailing showed poor thickening properties due to low amount of fines giving which means it does not start to thicken until around 65-70 %-w/w and with steep increase hereafter. Test of shear strength and permeability showed values close to those assumed in previous studies. Results of stability and seepage calculations done part of previous studies are thus still valid with only minor differences and give same conclusions.

The dam layout has been optimised (within what assumed allowed by the permit) some based on more detailed terrain map to use the natural elevation as confinement, but additional improvements need to be studied and evaluated in future studies.

The dam designs are the same as the ones developed part of the permitting process. The clarification ponds consist of a zoned moraine core dam with rock fill support and filters.

The TMF dams is constructed and raised in three stages for the northern TMF. The southern TMF is raised in two stages.

First a starter dam is constructed at the northern TMF up to +186 masl where needed (i.e. natural elevation below +186 masl), consisting of a supporting structure of rock fill (blasted rock) on the downstream side with coarse and fine filter layer on the upstream side both on the rock fill and as horizontal filter blanket under the tailing. Spigotting start from the crest of the dam.

During the second stage (first stage for southern TMF) the starter dam and supporting structure with filter layers are raised to full initial height at +190 masl. In the sections where the natural elevation is above +186 masl a starter dam also constructed up to +190 masl at this stage.

At the third stage the dam is gradually raised upstream to the final level of +210 masl using the spigotted coarser tailing as foundation. The outside of the embankment is protected with an erosion protection layer, and up to level +200 masl and filter a layer placed in-between the tailing and the erosion layer. The slope of the embankment from this stage on 1H:6V.

The dam embankment in-between is basically constructed in the same way as the clarification pond dam with a zoned sloping moraine core (up to level +189,15 masl as the clarification pond dam and with at crest at +191,0 masl) and a rock fill supporting structure. The difference is that the horizontal drainage filter blanket has been replaced with an upstream sloping filter (fine and coarse filter layers) on the rock fill and a horizontal filter blanket (fine and coarse filter layers) is used under the tailing to lower the phreatic surface as for the TMF embankment

There are two alternatives when starting up the production and taking the TMFs into operation, either starting with the northern and when it is full continuing with the southern or vice versa. Since the length of both the dam for the northern clarification pond and TMF are shorter than the southern alternative starting with northern dams result in the lowest Capex. Starting with the northern TMF mean both clarification ponds are needed during life of mine; the northern pond to start with to reduce initial Capex and the southern pond when the larger southern TMF is taken into operation since the surface area of the northern pond is to small so treat the water from site at this stage (even though the volume is large enough to manage water all year round). The southern facilities will be constructed around 5 years after production start-up taken into operation year 7.

Consequence classification of the dams is according to previous undertaken assessment 1B for southern TMF and clarification pond and 2 for the northern TMF.

Conclusions and recommendations - Costs

The majority of the Capex for the northern dams consist of cost of constructing the dam for the clarification pond including the dam in between the TMF. The scenario is basically the same for the southern dams.

By assessing the costs more carefully in future studies, it will be possible to increase the accuracy of the costs of some activities. For instance by looking when these activities will be done, it will be possible to consider different advantages with seasonal conditions. For example excavations in the wet areas are preferably done wintertime.

On the other hand costs related to drainage and dewatering activities of the wet areas during construction of the dams and construction of temporary coffer dams to be able to excavate the loose sediment, and place moraine and dam material under dry conditions, can be more expensive than estimated. Here more planning and investigations is needed

Another example is filter production may decrease. However filter cost can also increase if difficult to produce using the rock material from site.

So far all assumptions on Capital costs, especially for the Northern dams when the production haven't started, are made assuming it is possible to excavate and extract all materials (moraine and rock) from either within the inundation areas of the dams or the industrial area. When production has started waste rock might also be used to produce the rock material needed for the dam constructions, however this has not been assumed. If waste rock can be used the annual Opex will be reduced with around 2 Msek. If not possible the materials needs to be sourced from a local quarry. According to information from Maserfrakt it be possible, however not suitable, to get the material from local quarry in Håksberg. It would add an extra transportation cost of about 70kr/m^3 which would mean an increase of the Capex for the northern dams with around 12 MSek.

Another aspect that needs to be studied further is the dam layout for the northern TMF. The current dam layout in the northern part is located behind the old tailings stack. Besides from operational issues with the tailings and water management, if the dam layout is moved up on the old tailings stack the sustained Capex for stage 2 and especially Opex will be reduced (around 2 Msek a year during 2-3 years) until the +200 level is reached.

Conclusions and recommendations - Other

At the next stage as a basis for detailed design but also construction strategies and more accurate costing estimate, detailed geotechnical site investigations are needed. These will focus mainly on the wetland area but also on sections now categorized as solid grounds (moraine) and the previously deposited tailing and e.g. include ground penetrating radar, sounding (e.g. CPT and other), drilling and borrow pits. Laboratory test work will include e.g. PSDs, permeability tests, direct shear test, and proctor test.

An issue with the layout in the northern section of the TMF is the location of the dam behind the old tailings stack. This means that you basically construct a dam behind an already existing "natural" confinement. One thing that has to be investigated further is if suitable to construct a starter dam on the old tailing, e.g. considering the PSD, fine layers close to surface etc. An additional issue with the current dam layout is management of the tailing and water where the old tailings stack is too close to embankment to get sufficient segregation of the coarse and fine tailings. If raised the embankment would be founded partly on fine sediments which can't be allowed.

Tailings management poses many challenges during operation and needs to be studied and planned more in detail in future. Since spigotting will take place from opposite sides of the perimeter dams at both TMFs is means that potentially fine tailing from one side can flow over and settle close to the other embankment. However fine tailings cannot be allowed as a layer under the coarse tailing this close to the embankment as it can potentially can act as a slip surface and increase the risk of dam failure. Also the decant water (containing fine sediment and increasing the phreatic level) needs to be controlled and prevented from gathering to close to the other embankment. Both these issues are managed by carefully controlling the height of the lift on either side, i.e. moving the spigots well in time before the height one embankment get so high. Berms inside the TMF can also be constructed to stop the flow of fine tailings and water.

Management of the water during the different stages have many challenges both to divert the water from the TMFs to the clarifications ponds considering topographical differences inside the TMF and with a threshold of the spillway located a few meter above the ground at start-up of each of the TMFs (meaning pumping needed to the ponds) and to prevent too much sediments from entering the ponds. In future studies the management of water need to be studied more in detail during the different stages of operation including how the impact on dam minor changes dam locations.

An aspect that needs to be further studies is if the dam section in-between the TMF and clarification pond can be constructed as a non impervious embankment (i.e. a rock fill embankment with filter layers) since water at least initially will be stored on either side of the dam. Also when the dams are raised this concepts might be possible to use since water need to seep to the clarification pond anyway. This concept would make the excavation and of filling in the wetland much easier.

One alternative that need to be investigated further, if more cost and time effective, is whether the supporting rock structure of the dams can be used as a temporary coffer dams/dykes on one side. This might be possible if the loose sediments can be excavated wintertime without dykes on both side and the back-filled moraine replaced with blasted rock under the rock fill.

In future studies the consequence classification needs to be studied more in detail with dam breach simulations where needed.

Table of Contents

Technical report - Dams tailings and water

1. Introduction

1.1 Background and Scope of Work

Nordic Iron Ore AB (NIO) has commissioned Ramböll AB (Ramböll) to update the conceptual design and cost estimate of the clarification ponds and tailings management facilities (TMF) dams developed part of the Preliminary Economic Assessment (PEA) 2012 to a more detailed conceptual design and economical assessment part of the Technical Economical Study (pre-Feasibility level study) for the Blötberget iron ore mine near Ludvika in Sweden.

Part of the PEA 2012 a conceptual design was developed for the dams. In the study iron ore from both Blötberget and Håksberg (61 Mton) was part of the mine production. The deposition strategy of the tailing was based on conventional hydraulic deposition with upstream raise of the dam on the coarse tailing (beach) due to the restrictions in site area and amounts of tailings produced. Thickened tailings deposition with higher beach slopes was also discussed as a potential deposition strategy. The clarification pond consisted of a zoned moraine core dam. Deposition of the tailings was split in a northern and slightly larger southern TMF and the clarification ponds likewise in a smaller northern and larger southern pond.

The conceptual design was further detailed (dam foot prints the same) part of the environmental permit application process 2013. The design basis was primarily updated in respect of results from limited site geotechnical study and better terrain map. The design of the dams (structure and filters) was further developed and detailed including stepwise raise of the TMF dams. Stability and seepage assessment and preliminary consequence classification was also carried out.

The iron ore from the Blötberget underground mine is transported to the industrial area where an iron ore concentrate is produced along with tailing. The tailing is pumped as slurry to the TMF where it is deposited using convention deposition with coarse tailing closest to the embankment used part of the raise of the dam structures to reduce costs. Tailing drainage water is pumped or diverted using a spillway to the nearby clarification ponds. Also site run-off and mine ground water are pumped to the clarification pond. Water from the ponds is pumped to the process plant and excess water discharged to the canal flowing into Gonäsån. The clarification ponds need to be of sufficient size for treatment (sedimentation), store water and to attenuate flows.

The scope of work for this study comprises a more detailed conceptual design of the clarification pond and the TMF dams with associated structures (spillway, drainage ditches etc.) based on updated and more comprehensive basic design data and review of the deposition strategy and layout of the facilities. Based on this updated and more comprehensive and accurate economical assessment also based on more accurate cost unit data.

The report and scope includes;

- design basic data; site geotechnical conditions and tailings characteristic properties
- site water balance,
- deposition strategy, tailings and water management
- design and construction of the dams, and
- Bill of quantities and cost estimate for construction (CAPEX) and operation (OPEX).

Additional site geotechnical investigation (drilling/sounding and ground penetrating radar) in the wet areas was first intended to be included in the scope, but had to be excluded because of the project time schedule and also mild winter conditions not ideal to construct winter roads. The knowledge still of the geotechnical conditions in the relatively short distance of the dams located in the wet lands is fairly good.

1.2 Site location

The area for the TMFs and clarification ponds dams are located approximately 5 km west of the town of Ludvika and 1,5 km west of the industrial area with the process plant.

The dam area is situated just west of road nr 50 and south west of the small mountain Norsberget (Figure 1 and 2). To the west the dam area borders to a smaller local road. The clarification ponds are located on either side of the small constructed canal which flows into a tunnel at the north eastern side of the dams, with the TMFs located directly north and south of the ponds. IN the north section of the northern TMF the old tailings stack is located.

Areas with low elevations closest to the canal (figure 2) in the middle of the dam areas are covered with peat or have open water, while vegetation otherwise mainly consists of trees and bushes in the remaining areas.

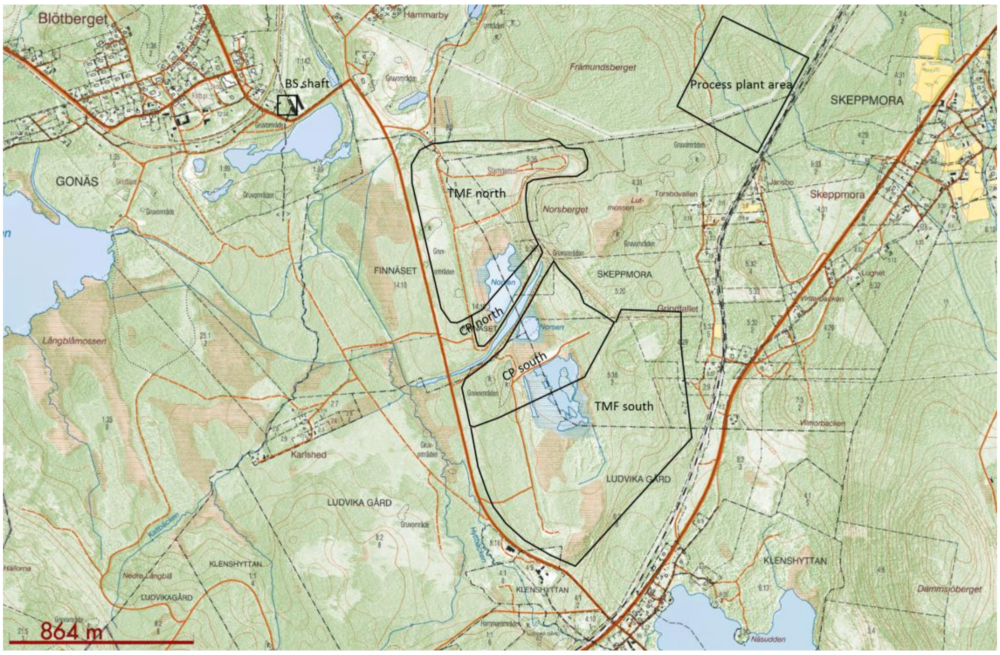

Figure 1. Map of the dam and mining area.

Figure 2. Ortho photo of the dam and mining area.

1.3 Dam regulations

1.3.1 RIDAS and GruvRIDAS

The Swedish hydropower industry provides dam safety guidelines, RIDAS, covering guidelines on construction, operation, maintenance and surveillance, and providing a consequence classification system, for water retention dams. The mining industry dam safety guidelines, GruvRIDAS, are based on RIDAS, with a number of adjustments and additions to cover the conditions that are specifically applicable to tailings storage facilities. GruvRIDAS outlines the major principles for the design, operation, surveillance and closure of tailings dams in Sweden. All member companies of SveMin have agreed to follow the requirements of GruvRIDAS.

The overall objectives of the dam safety guidelines are:

- Define requirements and establish guidelines for adequate and uniform dam safety;
- Constitute a basis for a uniform evaluation of dam safety and identify measures needed to improve dam safety; and
- Support authorities in their supervision of dam safety.

The guidelines given in RIDAS, apply for the design of the clarification pond dams and the guidelines given in GruvRIDAS for the design of the TMFs. The main part of RIDAS and GruvRidas used for the design of the dams includes detailed design guidelines on the foundation of an embankment, requirements on the moraine used as impervious dam core, rules for filter layers and erosion protection, as well as guidelines on dam crest, freeboard, instrumentation and monitoring, stability and seepage analysis.

According to the Swedish Environmental Code anyone owning a dam is responsible for maintaining it so as to safeguard against damage to public or private property or interests following an alteration of water conditions. A dam owner in Sweden has paramount responsibility for the safety of the facility.

1.3.2 Swedish Design Flood Guidelines

The Swedish design flood guidelines were originally published by The Swedish Committee for Design Flood Determination (Flödeskommittén) in 1990. The guidelines were an important upgrade to Swedish dam safety criteria regarding extreme floods. The current guidelines are presented in an edition from 2007, which replaced those from 1990 (Svensk Energi, 2007).

The guidelines are primarily directed at dam owners and consultants who carry out design flood calculations. Design flood determination is based on a classification into two categories depending on the potential consequences of a dam failure during flood conditions. Flood Design Category I should be applied to dams for which failure could cause loss of life or personal injury, considerable damage to infrastructure, property or the environment, or other large economic damage. Flood Design Category II should be applied to dams for which failure could only cause damage to infrastructure, property or the environment.

Design flood determination in Flood Design Category I should be based on hydrological modeling techniques that describe the effects of extreme precipitation under particularly unfavorable hydrological conditions. The different flood generating factors, each within limits of what has been observed, are combined to give the most critical total effect on the river system. A flood determined by this method has return periods that are estimated to exceeding 10,000 years. Design Category II should be able to pass a flood with a return period of at minimum 100 years at full supply level. Frequency analysis is applied for this determination.

2. Geology

2.1 Bedrock Geology

The regional geology is mainly dominated by primeval granite with streaks of gneiss and where biotite is the dominating darker mineral. Tectonic processes have enhanced the occurrence of schist which is typical for the older bedrock in the area.

Mining activities has previously been performed in the area and bedrock is very shallow in some parts especially the western area where the TMF and clarification pond are located.

2.2 Quaternary Geology

Most of the area where the TMF and clarification pond are located consists of ground with shallow moraine.

In the middle of the area a wetland with peat and woodland lakes are located.

During the previous mining period a tailings facility was constructed in the northern part of the area.

3. Topography

In general the topography consists of relatively flat terrain inside the dam areas and to the west with elevation around +180 masl. To the east and north east the terrain is hillier with elevation ranging from +220 to +230 masl and in the north part of the northern TMF around +190 masl. The old tailings stack has an average height of around +202 masl (varying from +198 to +207 masl). The process plant is located at +220 masl. A topographical map of the dam area also including the process plant location are presented in Figure 3 and 4.

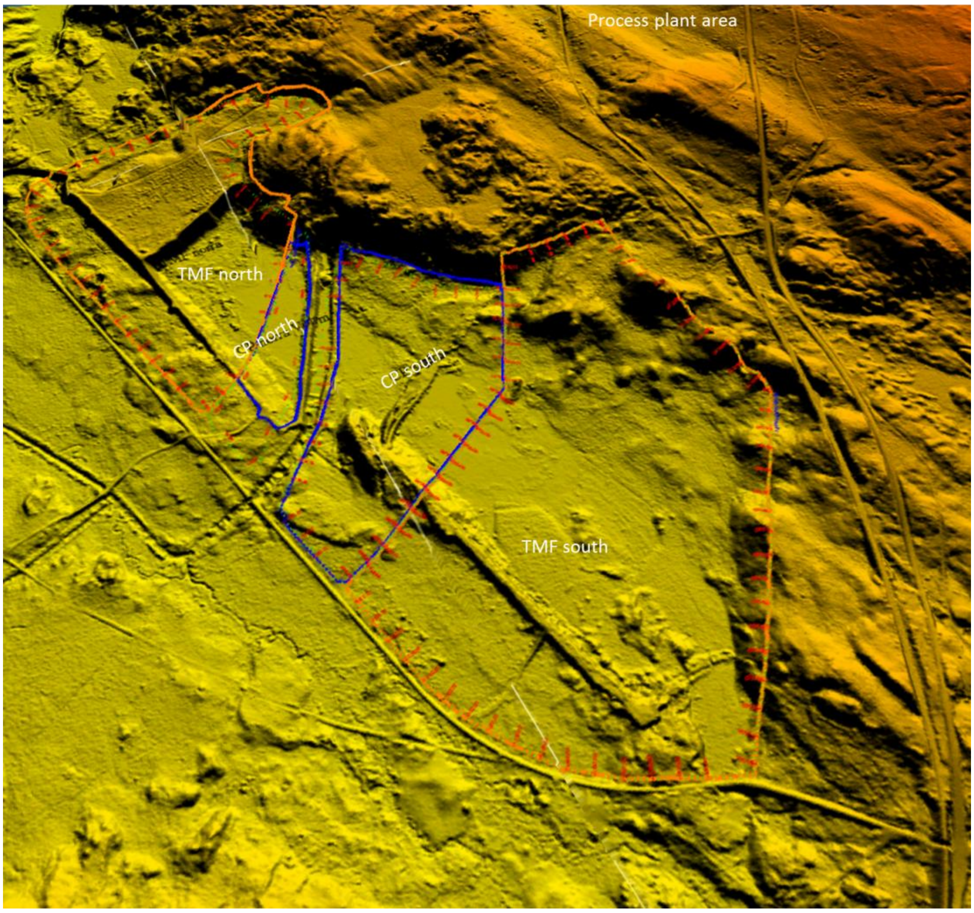

Figure 3. Topographical presentation of the dam areas. View from south towards process plant location.

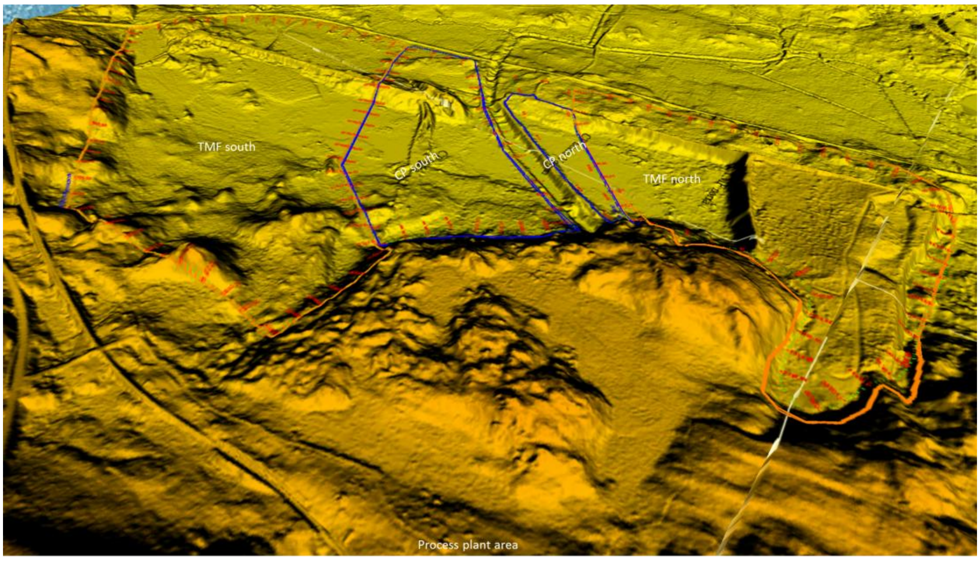

Figure 4. Topographical presentation of the dam areas. View from north and process plant location.

4. Site geotechnical assessment

4.1 Introduction

Initial geotechnical investigations were executed in Mars and April 2012. These targeted the area (Figur5) of the dam for the clarification pond dam parallel to the canal and mainly the wet area.

The field investigations included:

- Pyramid penetration test (Tr) in 17 locations.
- Percussion sounding with registration in 9 locations.
- Stick sounding in 5 locations.
- Disturbed soil sampling in 14 locations.

Laboratory investigations of soil samples included:

- Water ratio.
- Type of soil.
- Classification of material type according to Swedish regulation TK geo.

No investigations were executed outside the wet are. The remaining length of the dam layout was ocular inspected and has been classified as ground with moraine.

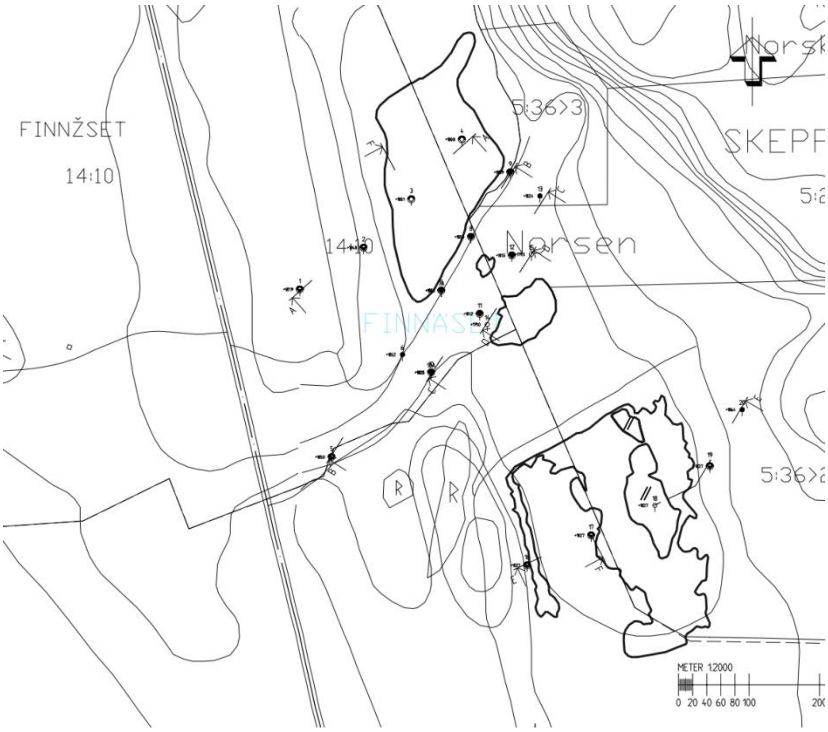

Figure 5. Locations of penetration tests/sounding part of site geotechnical investigations.

4.2 Dam between northern TMF and northern clarification pond

The water depth in small lake has been measured at a few locations and vary between 1,0 to 1,7 m. Water level is situated at approximately +181,8 masl.

The soil and sediments at the shores and bottom of the lake consists of approximately 5 m of soft material (peat and loose sediments). The loose sediment could be a relic of finer failings from the upstream old tailings facility.

The length of the dam over the wetland and lake area is approximately 200 m.

Other parts of the northern tailings dam are located outside the wetland and will be founded on till and to some extent older tailing.

4.3 Northern and southern clarification pond dams parallel to canal

When constructed the material in canal was excavated and the masses placed directly beside it. The material consists of a mixture of sand, gravel and silt which were dumped and pressed down near to solid ground (moraine). Natural soil can be found at level +176 to +179 masl, which is approximately 5 m below ground surface.

Beside the material next to the canal it is likely the subsurface soils consists of soft soils such as peat and sediment soils.

4.4 Dam between southern TMF and southern clarification pond

The investigated section is situated partly on solid grounds and partly in the wet land.

In the wetland the soil and sediments at the shores and bottom of the lake consists of soft material (peat and loose sediments). At the deepest parts the thickness of peat and loose sediment is 6 to 7 m.

The length of the dam over the wetland and lake area is approximately 200 m.

Other parts of the northern tailings dam are located outside the wetland and will be founded on till and to some extent older tailing.

4.5 Recommendations on further studies

At the next stage as a basis for detailed design but also construction strategies and more accurate costing estimate, detailed geotechnical site investigations are needed.

These will focus mainly on the wetland area but also on sections now categorized as solid grounds (moraine) and the previously deposited tailing. The investigations will include for example.

- Ground penetrating radar to map the wider extent and depth of the peat and loose sediments
- Sounding (e.g. CPT and other) to investigate layering and depth to bedrock
- Drilling for soil classification and sampling
- Borrow pits for soil classification and sampling

Other techniques can also be used if needed.

Laboratory test work will include e.g. PSDs, permeability tests, direct shear test, and proctor test.

5. Tailings characterisation

5.1 Introduction

The tailings characteristic properties are important in many ways for the tailings management for example pumping, deposition strategy, dam design and design capacity. Physical properties are important e.g. for the dam design and final consolidated volumes of tailings (design capacity), rheological properties e.g. for pumping and deposition strategy and geochemical for deposition strategy.

In the PEA study very basic assumptions were made of the tailings properties mainly related to density for volume calculations.

In the work with the updated design part of the environmental permitting process further assumptions were made of the physical properties (e.g. friction angle, cohesion and permeability) based experience values from other mines to be able to detail the design in respect of e.g. dam slopes, filter materials and stability and seepage calculations. No tailings material was available for test work at that time.

Part of the scope of this study the rheological and physical properties of the tailing have been further studied and evaluated with test conducted on the tailing produced from the pilot plant run at GTK 2014. Segregation test in a small bench scale test was performed to produce a coarse and fine tailing part of the test work. The physical tests were conducted by MRM lab in Luleå and the rheological tests by Ramböll. Evaluation of results has been done by Ramböll.

Geochemical properties have been studied previously e.g. part of the environmental permitting process where it has been concluded that is potentially net buffering due to the very low sulphur content and thus not requiring any special deposition strategy or quality closure cover design. No additional studied have been made part of the scope of this study.

5.2 Pilot plant operation and tailing

At the GTK pilot plant operation 17-20 November 2014 21 ton of ore was processed producing around 12 ton of tailing. During the first two days set up and testing of the process was done with the main pilot plan run on the 19-20th. The tailings produced during the whole pilot plan run were collected in a large thickener. No tailings was removed or pumped from the thickener for sampling during the runs which means that in the end there was a mixture of tailings from all days in it. Consequently the thickener and total tailing mixture in it contain potentially less representative tailing from the first two days.

The primary grinding of the pulp as established through bench-scale test work had a D80 target of around 500 µm. During the main pilot plan run 19-20th the D80 however was lower due to process equipment restrictions and varied between 230 to 280 µm. This consequently means that the tailing produced is on the fine side with slightly more fines especially above 100 µm.

From the thickener two 200-L barrels were filled with in total around 200 litres of tailing (around 300 kg) and shipped to MRM lab in Luleå for tests. Two samples were taken named tailing 1 and 2.

During the main pilot plan run 19-20th samples on the coarse and fine tailings pumped to the thickener was sampled for PSD (particle size distribution) analysis, and since the tonnages of these flows are known a total PSD for the tailings on these two days can be calculated and compared to the PSD on the tailings sample.

5.3 Bench segregation test of tailing

At deposition the tailing will segregate with the coarse particle settling closest to the spigot and embankment and the fine particles further out in the TMF. The property especially of the coarse tailing is an important design factor for the dam design and stability. No standard practice exist on how to segregate the tailing in lab scale and the experience in mine literature is very limited. Segregation can be done both vertical e.g. in a cylinder or horizontal e.g. in a flume or similar, the later better simulating the actual conditions in-field.

5.3.1 Preliminary segregation test

Preliminary test work of how the tailing slurry settles and segregates once agitation is stopped was first studied by Ramböll using vertical settling in a smaller cylinder (54 mm inner diameter) of a mixture of tailing 1 and 2. The tests also give information at about which solids content the tailing start to form a thickened non-segregating tailing.

Different solids contents were tested ranging from 40 to 70 %-w/w. Photos were taken and observations done of the tailing during the first 10 minutes when primary segregation and settling takes place.

Photos of the cylinders with the tailings samples were taken at 0 min, 3 min (only 40, 45 and 50 %-w/w), 5 min and 10 min and are presented in Figures 6, 7, 8 and 9. It should be noted that the amount of tailing in the samples increases with higher solids content which is why the height of the coarse material is larger in these cylinders.

Figure 6. Vertical segregation and settling test at different solids contents tailing 1+2 at 0 min.

Figure 7. Vertical segregation and settling test at different solids contents tailing 1+2 at 0 min.

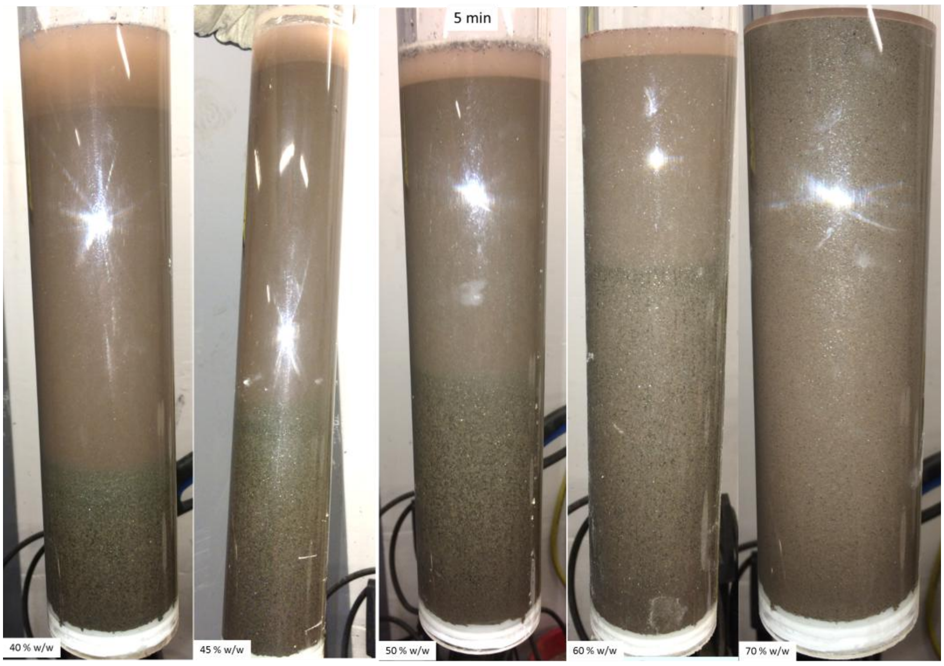

Figure 8. Vertical segregation and settling test at different solids contents tailing 1+2 at 0 min.

Figure 9. Vertical segregation and settling test at different solids contents tailing 1+2 at 0 min.

In the samples with 40, 45 and 50 %-w/w solids the segregation and settling of the coarse particles was very fast and already after a few seconds coarse material could be seen at the bottom of the cylinders (Figure 5). After around 3-5 minutes basically all of the coarse particles in these three samples had settled with little difference hereafter, e.g. comparing 5 and 10 minutes.

The segregation was slightly faster and better at 40 %-w/w as compared to 50 %-w/w were minor interaction from the fines could started to be seen. Based on the test 45 %-w/w solids content was selected as the solids for the horizontal segregation test and also the planned process. In-field it is very likely, based on the observations made and previous experience that the segregation will function as good at 50 %-w/w as at 45 %-w/w. The lower limit was set to 40 %-w/w solids content. Lower solids content will still result in segregation but at the same time the amount of water in the tailing and thus the flow that need to be pumped to the TMF will increase.

From the test it could also be observed that the tailing contain a high portion of coarse particles compared to the amount of fines which is beneficial when using conventional deposition with beach.

It could also be observed that at around 60 %-w/w solids content the segregation was not fully with some fine particle in the coarse and thus also more hindered settling. However the solids content needed to be as high as 70 %-w/w to start forming a non-segregating tailing.

5.3.2 Horizontal segregation test

The horizontal segregation test was done by MRM using a gently sloping (around 3 %) steel flume structure about 1 m wide 2 m long (Figure 10). For the test a mixture of tailing 1+2 was used at a solids content of 45 %-w/w solids. The tailing slurry was mixed and transferred to a large 30-L bucket (Figure 10) where an agitator was used to keep the particles in suspension and from settling before the test.

The tailing was transferred by gravity to the flume using a 25 mm pipe at an appropriate flow. Water accumulating at the other end flume was removed by a hole as can be seen in Figure 10.

Result of the test can be seen in Figure 10 and 13.

Figure 10. Flume used for horizontal segregation test. Coarse tailing at the bottom of the picture and fines at the top.

Figure 11. 30-L bucket from which the tailing slurry was transferred to the flume.

Figure 12. Result of horizontal segregation test. Coarse tailing to the left and fines to the right.

The test and segregation worked very well. As can be seen in figure 12 the length of the coarse beach was quite large in respect to the fines material which, as mentioned above, is clear indication of the amount of coarse particles in the tailing. Also keeping in mind that the pilot plant tailing was slightly on the fine side as compared to the target average tailing.

After the test all the excess water was drained and the coarse and fine material samples for further lab test work.

5.4 Rheological properties

5.4.1 Test method

Rheological test was executed to determine the static yield stress of the tailing slurry, i.e. the force at required to set the tailing slurry in motion or similarly to stop it from continue flowing. The test was done using the so called slump test (test results by experience very similar using Vane equipment). The test is performed by using a cylinder with approximately the same diameter as height, in this case 84 mm, which is placed on a horizontal board or similar (Figure 13). The tailing mixture is agitated to fully shear the sample and then directly poured into the cylinder to the rim after which the cylinder is lifted up quickly. The tailing flow out and the height of the slump is measured in the middle.

The yield stress is calculated (in Pa) from the height of the slump and measured slurry density (measured using the 54 mm cylinders) using specific formulas.

Figure 13. Slump test equipment.

5.4.2 Results and conclusions

The test was done for tailings with solids content of 50, 60, 70, 72,5 and 75 %-w/w.

Photos of the slumps at the different solids contents are presented in figure 14 and 15.

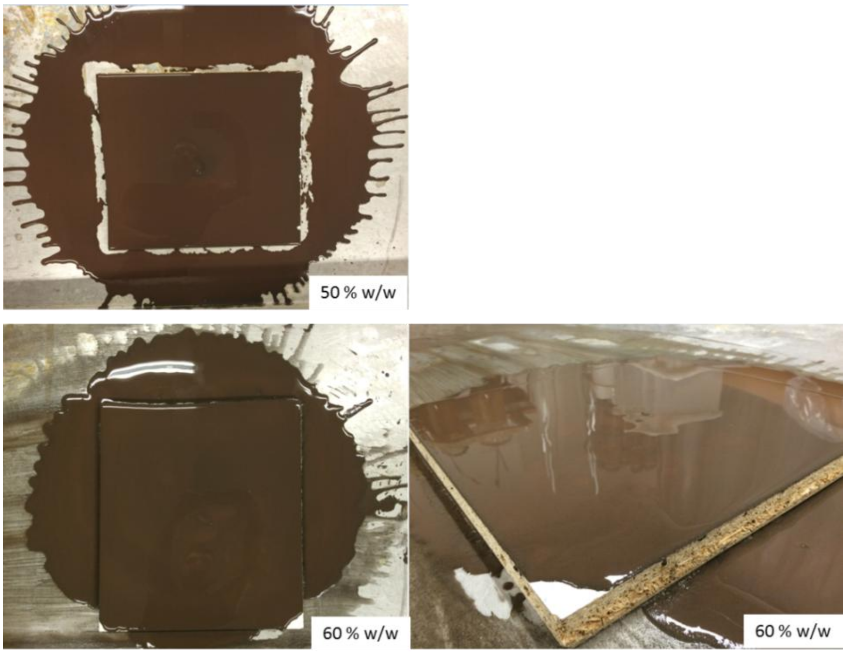

Figure 14. Slumps at 50 and 60 %-w/w solids content.

At 50 %-w/w the tailing slurry segregate so fast that the test was difficult to perform. At the 60 %-w/w the tailing was still very easy flowing, slump height around 1 mm and with a very low yield stress around 1 Pa.

It is not until 70 %-w/w that the tailing can be observed to start having non-segregating properties and behave as a thickened tailing. The yield stress is still relatively low around 10 Pa.

With higher solids content going to 72,5 and on to 75 %-w/w the slump height and yield stress increases quite fast with a yield stress of around 100 Pa (basically paste properties) at 75 %-w/w.

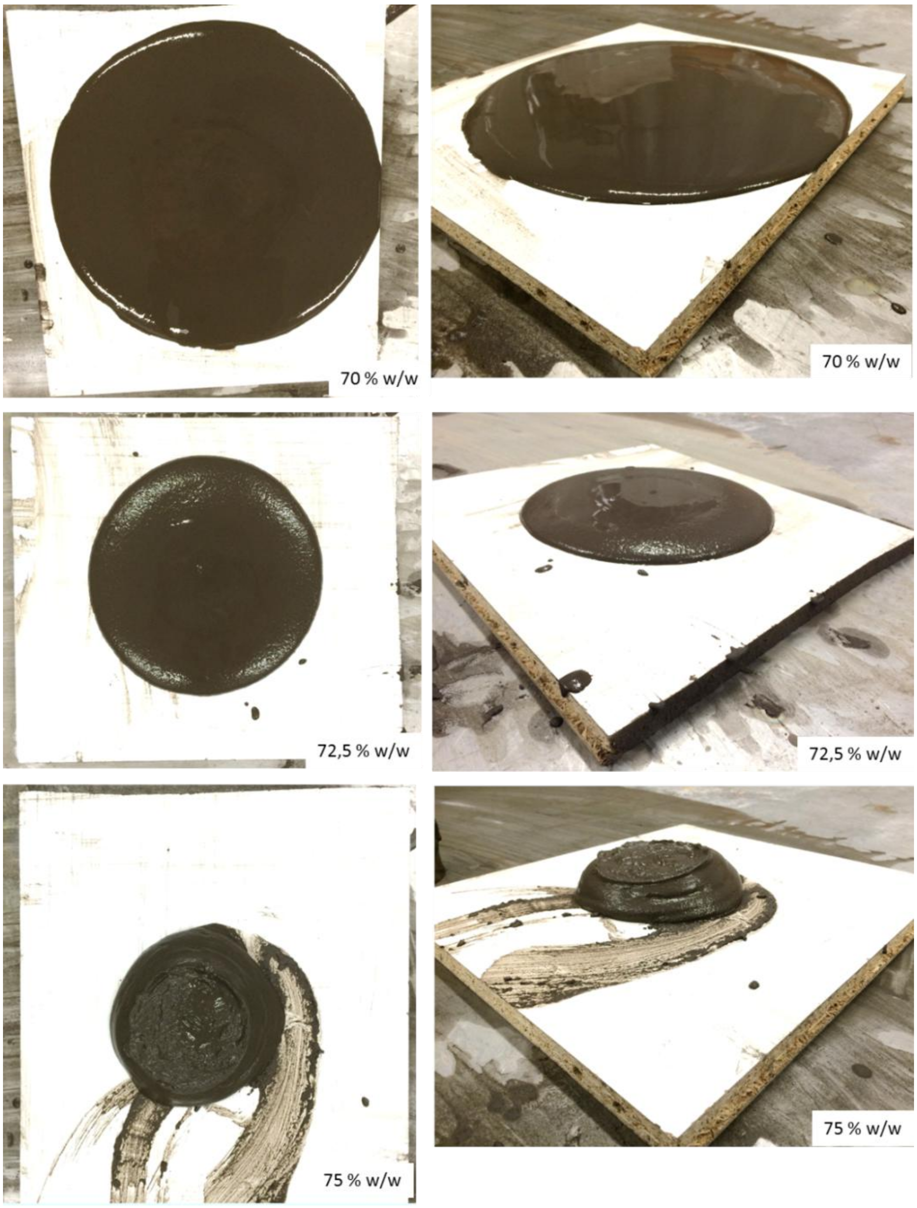

Figure 15. Slumps at 70, 72,5 and 75 %-w/w solids content.

In figure 16 the yield stress is plotted against the solids content of the tailings samples. As a comparison values for the Northland Resource tailings is included as comparison. From then segregation test and the rheological test of the yield stress it can be concluded that very high solids content around over 65 %-w/w is needed before a non-segregating tailing starts to form. The reason for this as presented further below in the section with result of PSD data is the low quantity of fines (<20 µm) in the tailings. A certain amount of fines (normally at least 20% <20 µm) are needed to create an increased interaction between the particles so the tailing start having non-segregating properties (the fines keep the coarse in suspension) and thickened properties with a yield stress >5-10 Pa.

A consequence of the low quantity of fines is that the interaction between the particles increases very rapidly within a short interval of solids content (70-75 %-w/w) as can be seen in Figure 15. In this context it should also be noted that the pilot plant tailing was slightly finer than the expected average meaning that during actual production the curve can be pushed even more to the right and having an even steeper gradient. As also can be seen from Figure 15 the Northland tailing which contain considerably more fines but also more coarse tailing show a more gradual increase of the yield stress.

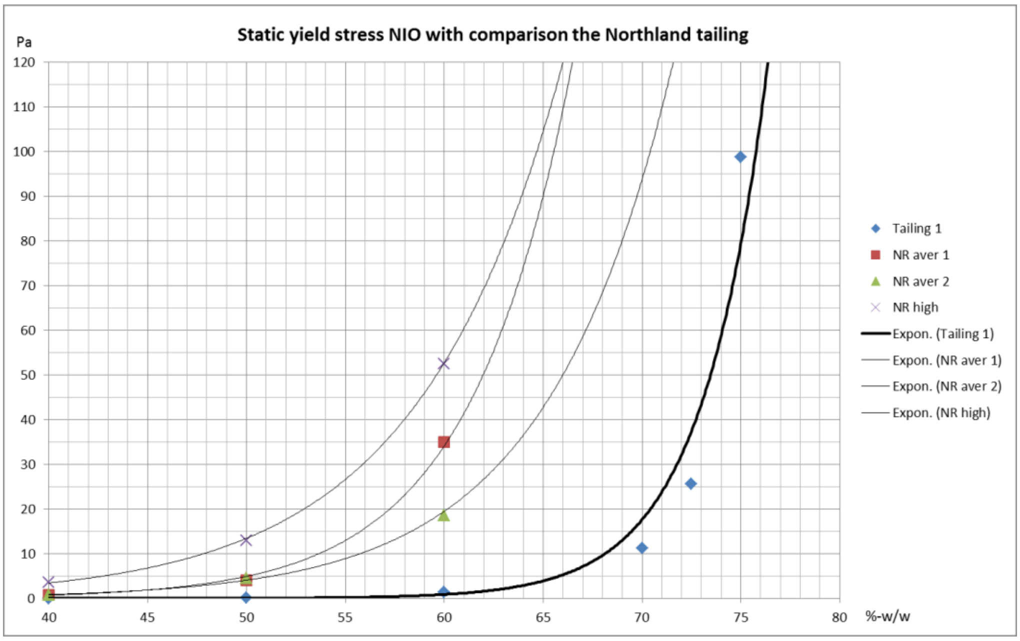

Figure 16. Yield stress plotted against solids content and values for Northland Resources production tailing as comparison.

From an operational point of view the NIO tailing does not show good rheological properties to produce a usable thickened tailing. The fast and steep increase of the yield stress once a thickened tailing is formed is very difficult to manage e.g. in a thickener where only small changes in the solids content can mean either a very easy flowing tailing or a very thick tailing that can take the thickener and pumps out of operation. Further a solid content above 70 %-w/w mean a very high density meaning the use of conventional slurry pumps is less practical due to high wear.

5.5 Physical properties

5.5.1 Scope

Test work of physical test work has mainly been executed by MRM lab, except slurry densities which have been determined by Ramböll. GTK also have done some PSDs on the tailings part of the pilot plant operation.

The test work has included the following test and tailing samples;

- Density; total tailing.
- PSD; total, coarse and fine tailings.
- Saturated permeability; total, coarse and fine tailings.
- Direct shear test for determination of friction angle and cohesion.

5.5.2 Densities

Slurry densities were determined for a range of different solids content using a cylinder with known inner diameter and weight. Results are presented in Figure 17 together with measured reference slurry densities from Northland Resources (based on many hundred measurements). The NIO densities show results as expected at different solids contents, except for the data below 50 %-w/w which likely is due to slightly incorrect result for the density or solids content. From the result above 50 %-w/w the specific gravity have been calculated to 2,83 kg/L (2830 kg/m3) which is in the same order of magnitude as the Northland tailing.

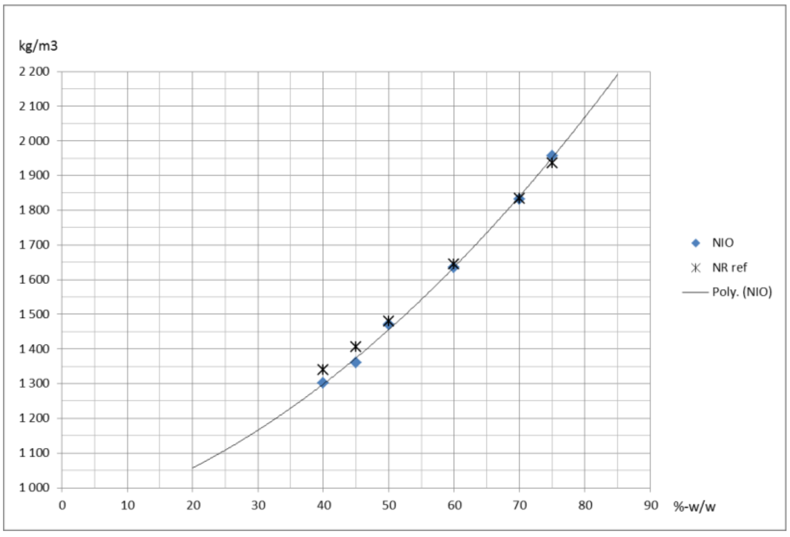

Figure 17. Slurry densities plotted against solids content with Northland Resources data as reference, both having approximately the same specific density.

5.5.3 **Particle size distribution**

5.5.3.1 Total tailings samples

PSD for the total pilot plant tailing are presented in Figure 17. The tailing samples (named "NIO PP tot tail 1 and 2 17-20/11") sent to MRM for the test work have been analysed both by GTK and MRM using wet sieving and by MRM also using sedimentation (pipet method) for fines <30 microns. The sample named "NIO PP tot tail 17-20/11", the curve with the most fines, was takes as a grab sample from the thickener. The samples named "NIO combined…" are the samples taken of the coarse and fine tailings during the main pilot plant operation 19th and 20th Nov for which PSDs was analysed and a combined PSD has been calculated. These PSDs are very useful since the primary grinding of the pulp on these days was around 230-280 microns as compared to the target of 500 microns. The D80 of the calculated PSDs tailing sample, which will have a value a slightly lower than the primary grind, is around 250 microns. Thus in actual operation with the target primary grinding PSD achieved the tailing will have D80 closer to 400 microns. The result for particles less than 100 micron are not expected to change especially much however.

In Figure 18 the PSDs are presented with interpreted fines <30 microns for the samples analysed by GTK based on the results of the MRM result and other data.

From the PSDs in Figure 18 and 19 it can be concluded that tailings samples taken and shipped to MRM for test work was slightly coarser than during the primary pilot plant run. And on the other end the tailings from the runs also contained much finer material as show by the grab sample. This variation also has been seen in other similar pilot plant operation at GTK. All PSDs from the pilot plant operation are slightly finer than expected during actual operation however not unrealistic and can be considered conservative for the planned production.

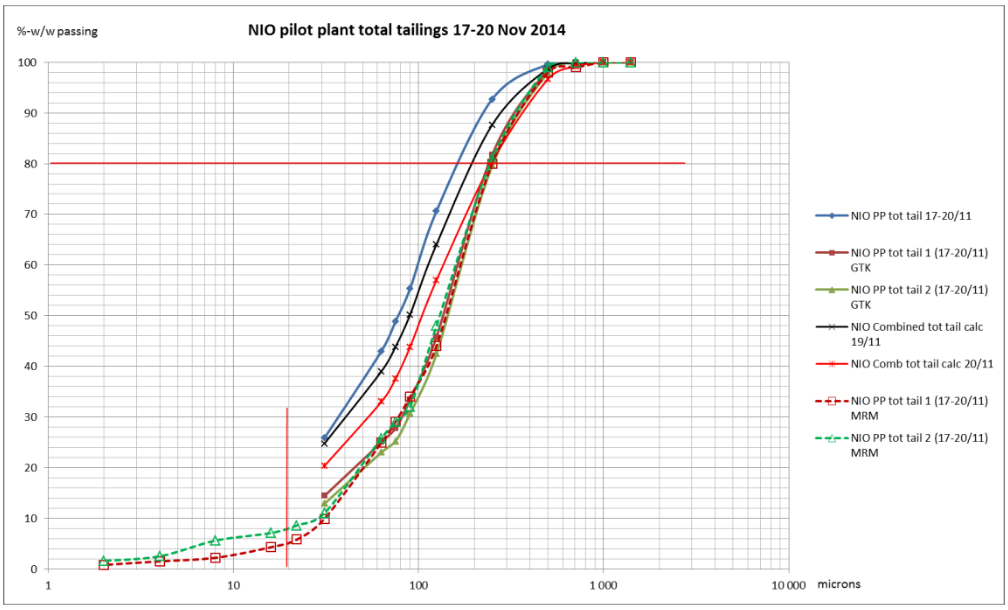

Figure 18. PSDs of total tailing from pilot plant operation.

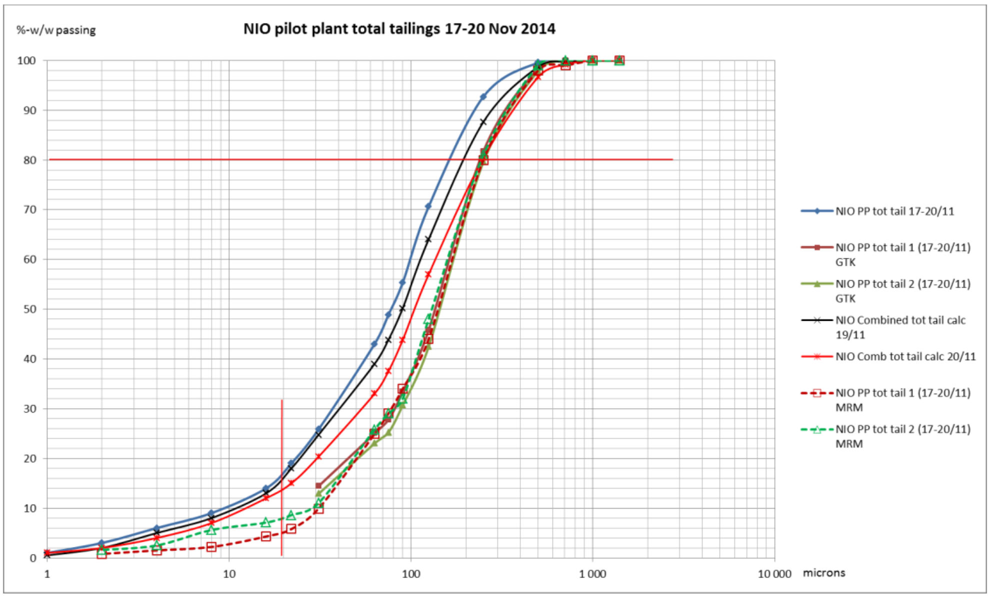

Figure 19. PSDs of total tailing from pilot plant operation with interpreted fines <30 microns for the samples analysed by GTK.

The amount of particles less than 20 microns are around 7% in the MRM samples as can be seen in figure 19. As a rule of thumb at least 20% is needed to form a practical manageable thickened tailing.

Figure 20 present the projected PSDs for the total tailing during production. The finer curve is based on the finest during the pilot plant run in Figure 19, the average on the tailing 1+2 sample with corrected D80 and the coarser on conservative values (high throughput at low grades).

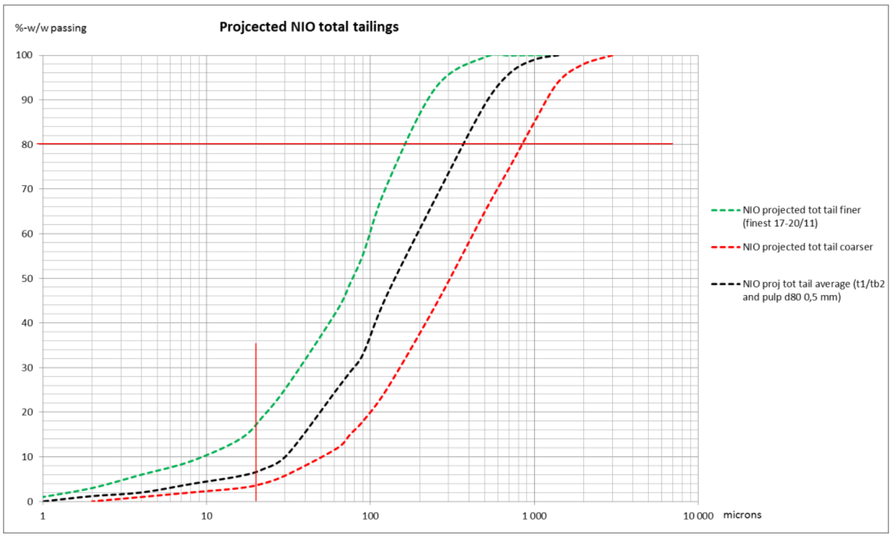

Figure 20. Projected PSDs of the total tailings with average variation finer to coarser.

5.5.3.2 *Coarse and fine tailings samples*

The PSDs of the coarse and fine segregated tailing from the segregation are presented in Figure 21 together with the PSD of the total pilot plant tailing. As can be seen the PSD of the coarse is relatively close to the total tailing which as mentioned above clearly indicate that the total tailing mainly consist of coarser particles which is beneficial for conventional deposition with beach. The fine tailing show a typical curve with large amount of particles finer than 20 and 10 microns.

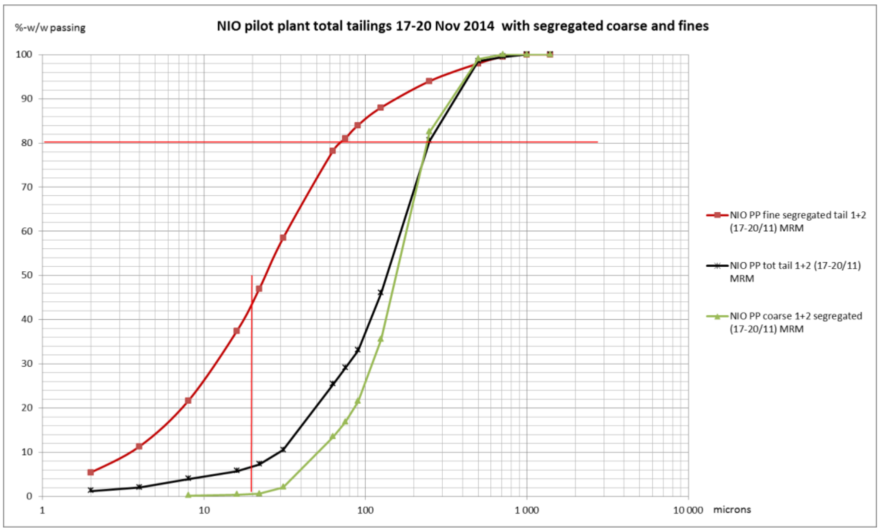

Figure 21. PSDs of the coarse and fine segregated tailing from the segregation test with the PSD of the total pilot plant tailing.

Figure 22 present the projected PSDs of the segregated coarse tailing with some reference PSD and estimated outer limits (dotted black lines). The average coarse (black curve) is based on the PSD of the pilot plant coarse tailing (blue dotted curve) with correction for D80 to around 400 microns same as total tailing. The finer (green) curve is based on the total tailings having the most fines PSD in the pilot plant operation with slightly less fine particles under 100 microns. The coarser curve (red) is based on the projected coarser PSD of the total tailings.

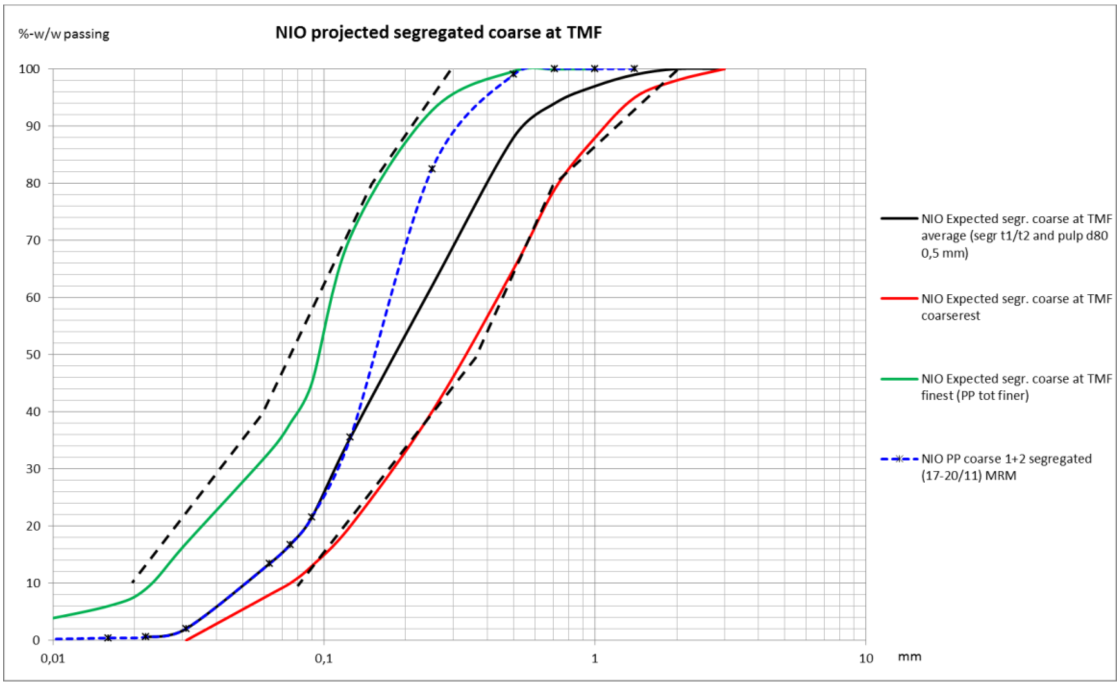

Figure 22. Projected PSDs of the coarse segregated tailing at the TMF.

5.5.4 Saturated permeability

Permeability test has been performed for each of the tailing samples total tailing (1+2), coarser and finer. The samples very packed to the density expected in-field around 1,9 kg/L.

The permeability for the total tailing sample was 1,6x10^{-6} m/s, and for the coarser tailings the permeability it was 4,3x10^{-6} m/s and for the finer tailings is was 1,7x10^{-7} m/s.

In comparison with the previous assumptions made for the coarser tailings in the PEA and permit application, the lab resultant of the saturated permeability according is in the same order of magnitude. For the finer tailings the lab results are about 10 times lower than the earlier assumption made in the PEA. The result from the lab can be interpreted as representing a mean value of the saturated permeability in both x-axis and y-axis. If the difference between the x-axis and the y-axis is about 10 times i.e. 10 times lower in y-axis as often seen in-field, the lab results can be considered as representative.

When evaluating saturated permeability performed in lab the results of the samples with different PSD should always checked and compared to each other. In this case the coarser tailing has the highest permeability, the total tailing slightly higher and finer the lowest of the three samples which verify that the results are relevant.

5.5.5 Shear strength

The direct shear tests have been used to determine the shear strength (friction angle) of the tailing. The test has been performed for the normal stresses 20, 150 and 500 kPa. These three normal stresses will represent different depth in the tailings stack 1 m, 7,5 m and 25 m.

Direct shear strength test indicate (Figure 23) that the finer tailing has the following un-drained internal angel of friction,

- normal stress interval 0-20 kPa at 28,8 degrees
- normal stress interval 20-150 kPa at 15,89 degrees and a cohesion of 5,3 kPa, and
- normal stress interval 150-500 kPa at 16,25 degrees and a cohesion of 4,3 kPa.

Results for the coarser tailing (Figure 22) show the following un-drained internal angel of friction

- normal stress 0-20 kPa at 35 degrees
- normal stress interval 20-150 kPa at 24,4 degrees and a cohesion of 4,9 kPa, and
- normal stress interval 150-500 kPa at 18,5 degrees and a cohesion of 22,86 kPa.

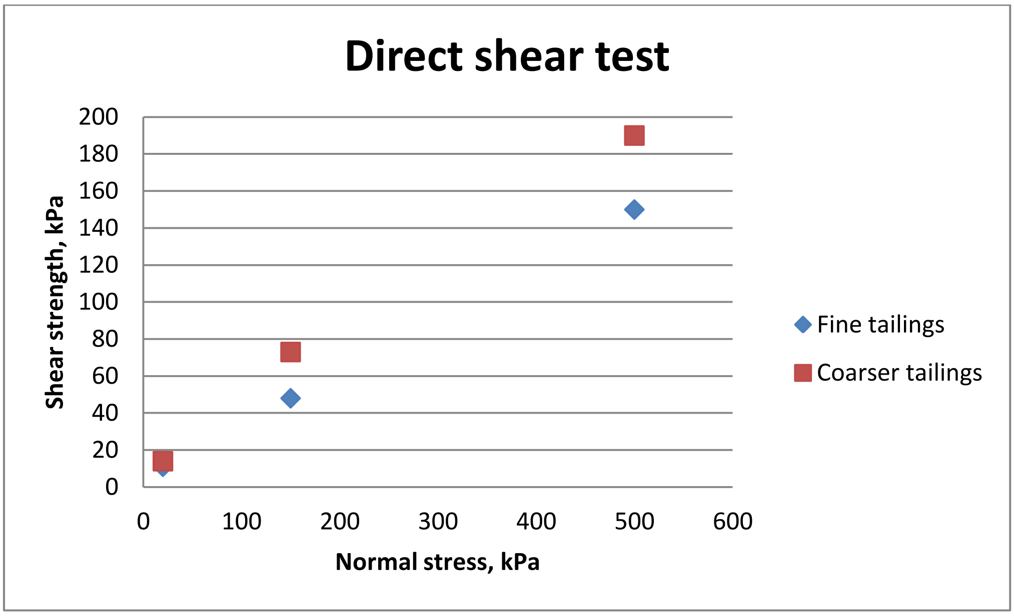

Figure 23. Shear strength plotted against normal stress for coarse and fine tailing.

In comparison with the previous assumptions made in the PEA and permit application the lab results are in the same order of magnitude. The lab results also have a value of the cohesion added to the shear strength. The biggest different is that the lab results show slightly higher shear strength in the shallower upper part of the tailings stack and slightly shear strength in the deeper parts of the tailings stack. The trend is the same for both the coarser and finer tailing. These results are positive comparing to the parameters used for calculating the slope stability in the PEA, with exception for the assessment of liquefaction. For this reason a new assessment of the slope stability have been done for the load case with liquefaction (see section 9.6).

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6. Production and design amounts

The planned amount of ore extracted used part of this study is in total 48 Mton (DMT design document 2015 Feb). This amount will result in around 28 Mton of tailing. The design amount of tailing has been set to +10% meaning 31 Mton.

The final consolidated density of the tailing will differ slightly between the coarse and the fine tailing. Based on the measured non-consolidated slurry densities of the total tailing and experience from other sites the following densities and solids contents have been derived. The coarse tailing will have a density in average 1,95 kg/L (variation 1,9-2,1 kg/L) and solids content 77 %-w/w (variation 72-85 %-w/w). The fine tailing will have a density in average 1,75 kg/L (variation 1,7-1,85 kg/L) and solids content 66 %-w/w (variation 63-70 %-w/w). This results in a combined density in average 1,9 kg/L (variation 1,8-2,0 kg/L) and solids content 73 %-w/w (variation 68-80 %-w/w) based on around 65 % coarse and 35 % fine tailings by mass.

For design the conservative density of 1,8 kg/L have been used. This give a total volume of tailing 15,5 (M)m^3 and 17 (M)m^3 for the design amount.

With an annual planned production of 3 Mton and ramp-up of production the first year, a total of 17 years of production have been used. The annual production of tailing is presented in Figure 24.

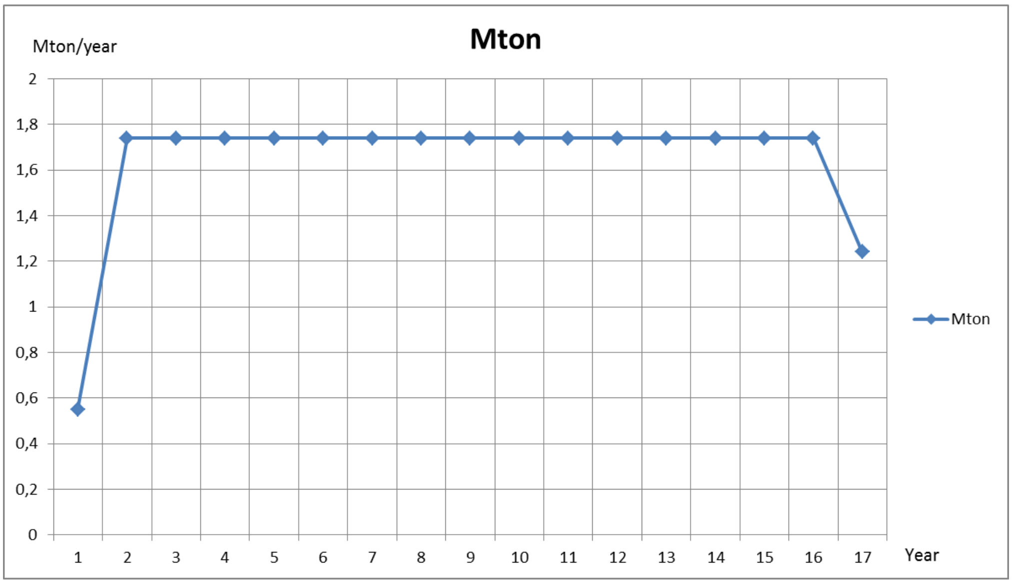


Figure 24. Projected annual production of tailing.

The annual accumulated amounts deposited are presented in Figure 25.

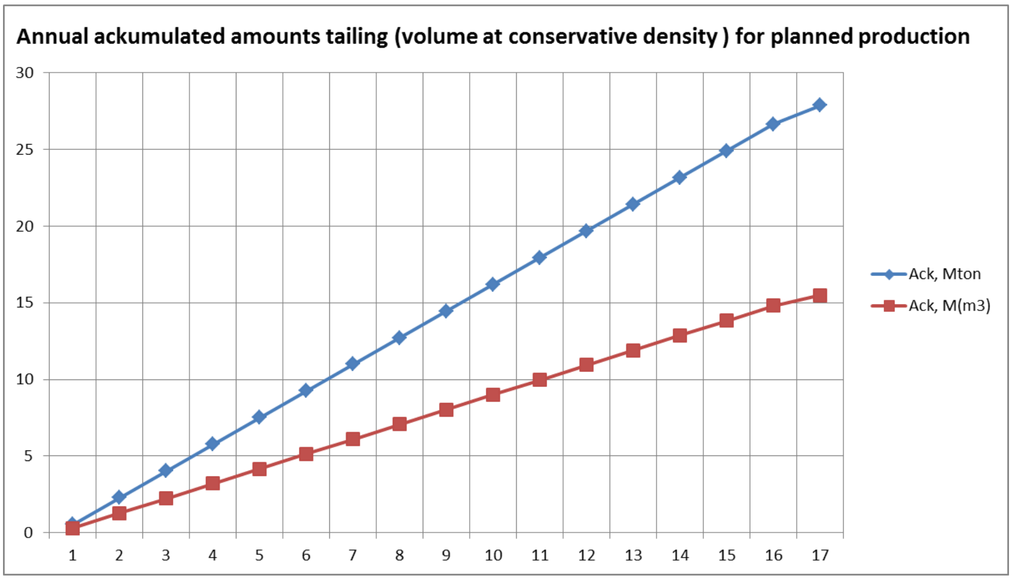


Figure 25. Accumulated annual tonnage and volume of tailing for the planned production amount.

7. Site water balance

7.1 Previous studies and assumptions

Part of the PEA study a very conceptual site water balance of the mining site was prepared that included balances on a yearly and hourly basis. Seasonal and monthly variations were not studied except a very rough estimate of the tailings drainage water frozen during winter and thawed during spring flood. I was also assumed that all drainage water freeze during the four winter months which is a large over estimation.

All tailings drainage water was assumed to become available and returned from the clarification pond to the process plant which is not a correct assumption since around 30% is lock-up as pore water in the tailing.

Water from the Blötberget shaft was assumed to be pumped directly to the canal flowing into Gonäsån, except a minor amount which was pumped to the process plant.

This resulted in an over-all net excess of water mainly due to run-off from the TMFs and clarification pond areas. During the four winter months around 0,8 million cubic meter of water was needed to make up for the frozen tailings drainage water. This water was extracted and pumped from Lake Väsman.

7.2 Updated site water balance

7.2.1 Introduction

This section briefly summarises the basic data and flows, assumptions, method and results for the updated site water balance.

7.2.2 Flows and facilities included

All facilities or parts where water either arises, are consumed, stored or extracted are included in the water balance. Some simplification has been done where the flows are small or does not impact the water balance as a whole.

The facilities and parts included and their main flow characteristic are presented in Figur 26.

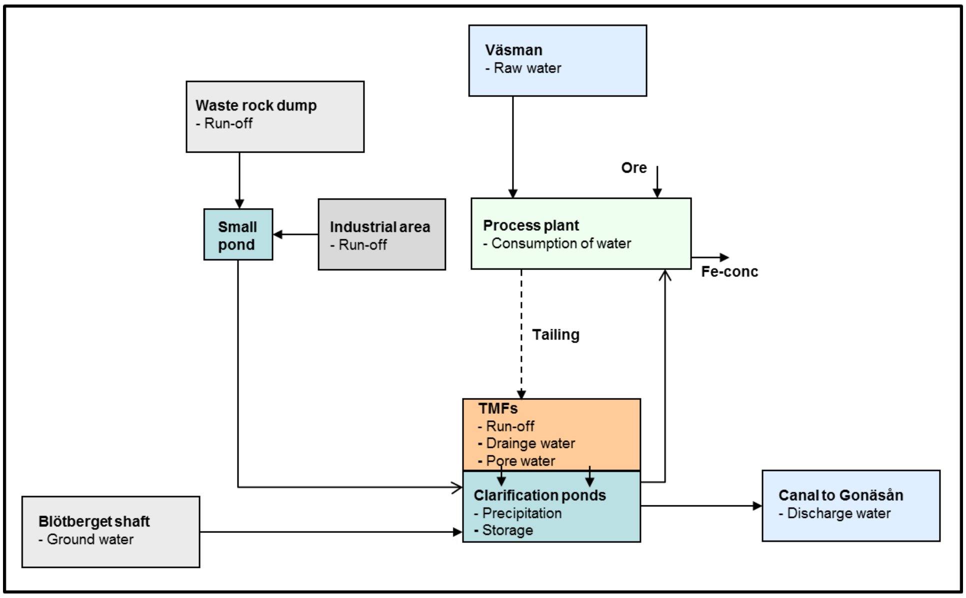


Figure 26. Overview water balance with main facilities/parts and flow of water.

In figure 27 a map of the area and the main water flows are presented.

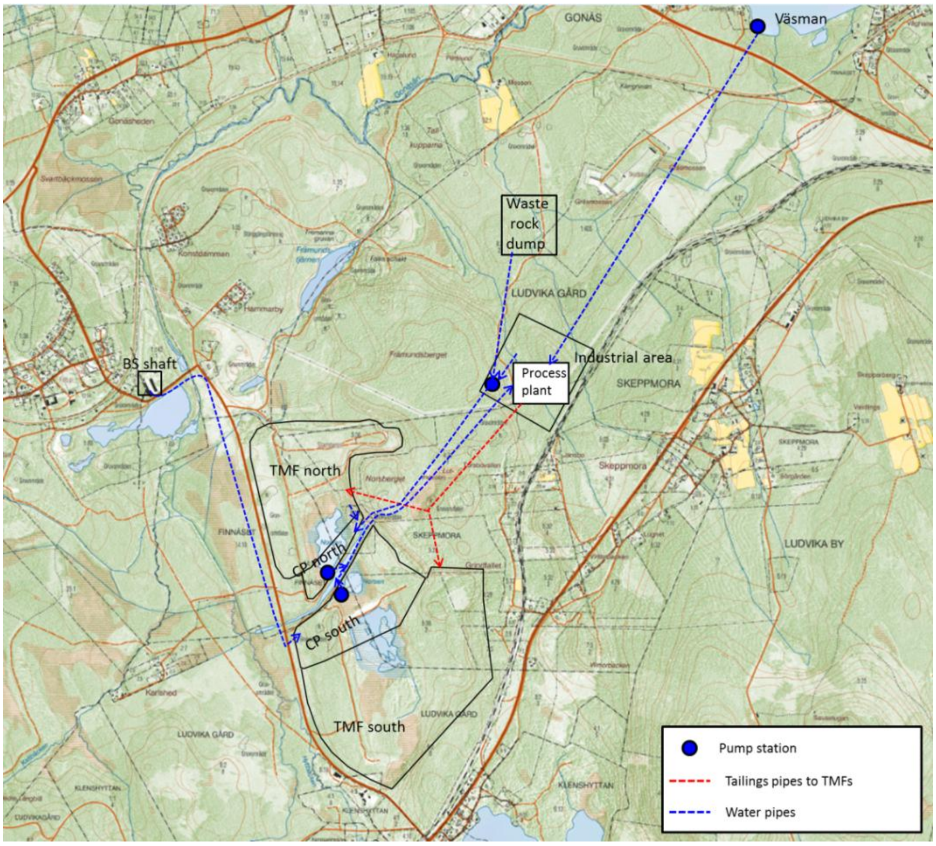


Figure 27. Map of the mining area with main water flows.

Flows not included in the water balance are seepage to and from the dams. Seepage from the dams will be diverted to the canal.

7.2.3 Basic data

7.2.3.1 Temperature, precipitation and evaporation

Average monthly temperatures and precipitation data are based on measurements from local station in Ludvika (SMHI station) for the period 1960-1990.

Evaporation are based on data from other locations in Sweden and corrected to the local site based on temperature differences.

Precipitation for a wet year is assessed to be around 50% higher than the average normal year also taking into account potential climatic changes. Precipitation for a dry year is assessed to be around 60% of the average normal year.

Precipitation and evaporation data are mainly used for the clarification ponds.

Data are presented in table 1.

Table 1. Average temperature, precipitation and evaporation data.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	*Average*
Temperature													
Average	-6,9	-6,6	-2,9	2	8,7	13,4	14,4	12,9	8,6	4,3	-1,1	-5,5	3,4
(mm)													
Precipitation													***Total***
Average	44,2	32,4	32,3	40,4	43,9	62,7	77,4	73,7	70,8	57,2	60,9	46,1	642
Evaporation													
Average	0	0	0	11	46	74	76	69	42	21	0	0	338
Net average	44,2	32,4	32,3	29,8	-2,2	-11,0	1,3	4,8	29,3	36,4	60,9	46,1	304
PET (free water surface)	0	0	0	16,0	69,7	98,9	104,8	95,4	68,6	34,3	0	0	488
Net PET	44,2	32,4	32,3	24,4	-25,8	-36,2	-27,4	-21,7	2,2	22,9	60,9	46,1	154

7.2.3.2 Surface run-off

Surface run-off data are based on SHMI S-Hype model data (1999-2013) for smaller run-off areas nearby to the mining area.

Data for an average monthly run-off together with typical dry year, wet year and wet spring flood are presented in figure 28. As can be seen yearly spring flood normally take place in April.

As with the precipitation data for a wet year is assessed to be around 50% higher than the average normal year and for a dry year around 60% of the average normal year.

Surface run-off data are mainly used for the industrial area and waste rock areas and the TMF with some modifications.

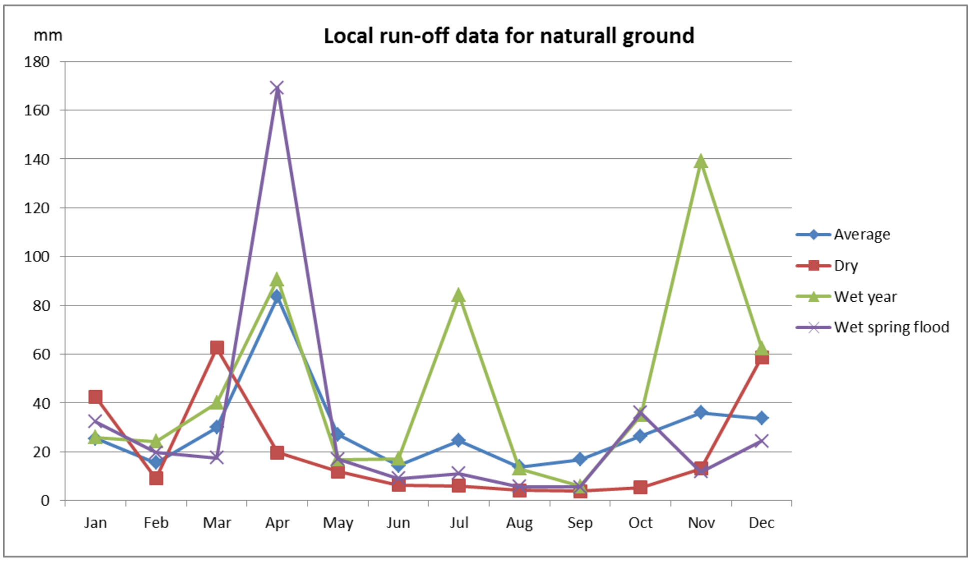


Figure 28. Surface run-off data (SMHI S-Hype for local area).

7.2.3.3 *Areas for precipitation and run-off*

The surface areas used in the water balance for the different facilities are presented in Table 2.

Table 2. Surface area for the facilities in the water balance.

Facility	Area (km2)
TMF north	0,42
TMF north	0,72
Clarification pond north	0,04
Clarification pond north	0,30
Waste rock dump	0,16
Industrial area	0,13

7.2.3.4 *Ground water*

The average flow of ground water pumped from the Blötberget shaft is 144 m^3/h.

7.2.3.5 *Raw water*

Raw water from Väsman can be extracted during four winter months with a maximum flow of 360 m^3/h according to the environmental permit.

7.2.3.6 Process plant water balance

In table 3 the water balance for the process plant is presented at nominal ore throughput with the make-up water needed at the average design solids content of the tailing at 45 %-w/w and the expected variation from 40 to 50 %-w/w. Hourly tonnages of ore, concentrate and tailings are based on the projected mass balance (TSC 2015-02-27).

For the site water balance variations of the throughput higher (at design level) or lower throughput than the nominal including variations in ore grade (impacting the amount of tailing on an hourly or daily basis) does not impact the water balance in the larger scale month to month.

Table 3. Process water balance with make-up water need. Water flows highlighted in blue.

	Unit	Throughput		
		Nominal	*Nominal*	*Nominal*
Ore in				
Solids	t/h	431	431	431
Water	%	3	3	3
	t/h	13	13	13
Fe-Conc out				
Solids	t/h	181	181	181
Part of ore	%	42	42	42
Water	%	6	6	6
	t/h	12	12	12
Tailings				
Solids	t/h	250	250	250
Part of ore	%	58	58	58
Solids	%	40	45	50
Water	%	60	55	50
	t/h	376	306	250
Make-up process water	t/h	374	304	249

7.2.3.7 Tailings characteristic

At deposition the tailing will in average have a solids content of 45 %-w/w with expected variation as presented above. After deposition most of the water will be released as drainage water however not all since some will be locked-up as pore water in the tailing. Around 72% of the water is expected to drain giving final average solids content of 78 %-w/w for the whole tailings stack. The majority of the water will drain from tailing just after deposition and during the first 24 hours when segregation and primary consolidation take place. The remaining will drain during a period of weeks to month when the tailing is impacted e.g. by secondary consolidation, dessication, new layers deposited on top and freeze/thaw.

During the winter months some of the drainage water will freeze in the TMF area, in average 25% based on data from other sites and corrected to the site, and thaw during the spring flood in April. No tailing and drainage water is expected to form permafrost.

7.2.4 Pond volumes

Volumes of the southern and northern clarification pond at different elevations (masl) are presented in figure 29 and 30 together with different operational levels (DL is dam water limit i.e. highest allowed water level). The minimum and normal highest levels are the levels between which the water level can be allowed to vary during normal operation. The minimum level in the ponds has been set based on the level of the intake pipe to the pump stations and with at least half a meter of water above the pipe crest. The normal highest level have been set half a meter below the spillway level not to risk spilling any water during normal operation (also taking into account waves etc.).

The volumes are also presented in table 4 and 5 with the normal available operational buffer volume. Before spring flood the volume in the ponds is lowered slightly to be able to attenuate the peak flows during spring flood resulting in a lower maximum discharge flow.

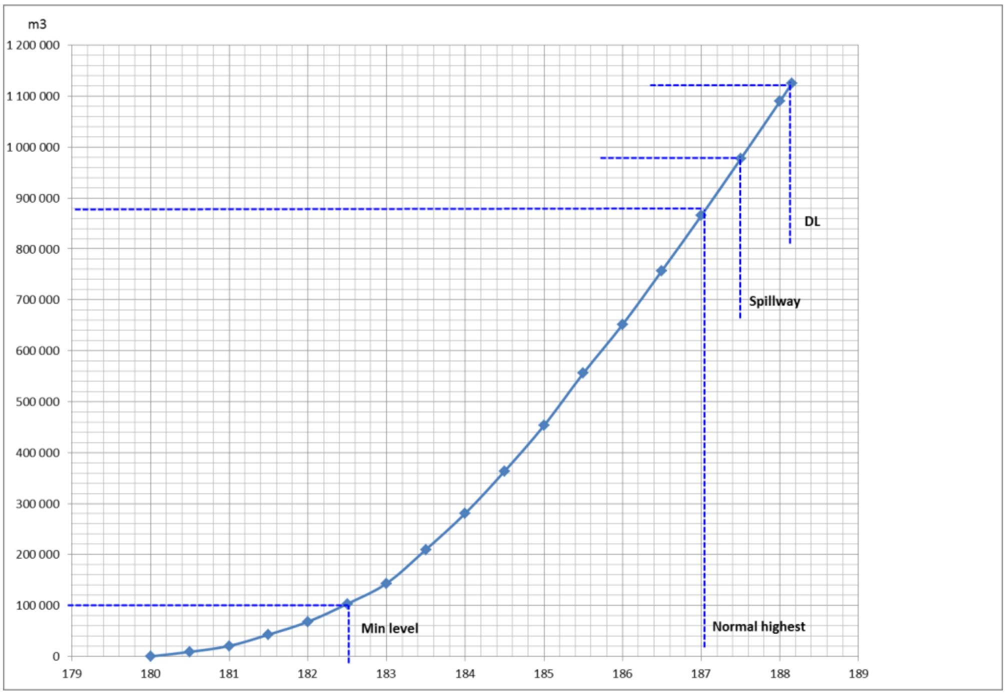


Figure 29. Volumes of water in the south clarification pond at different levels (masl) together with operational levels.

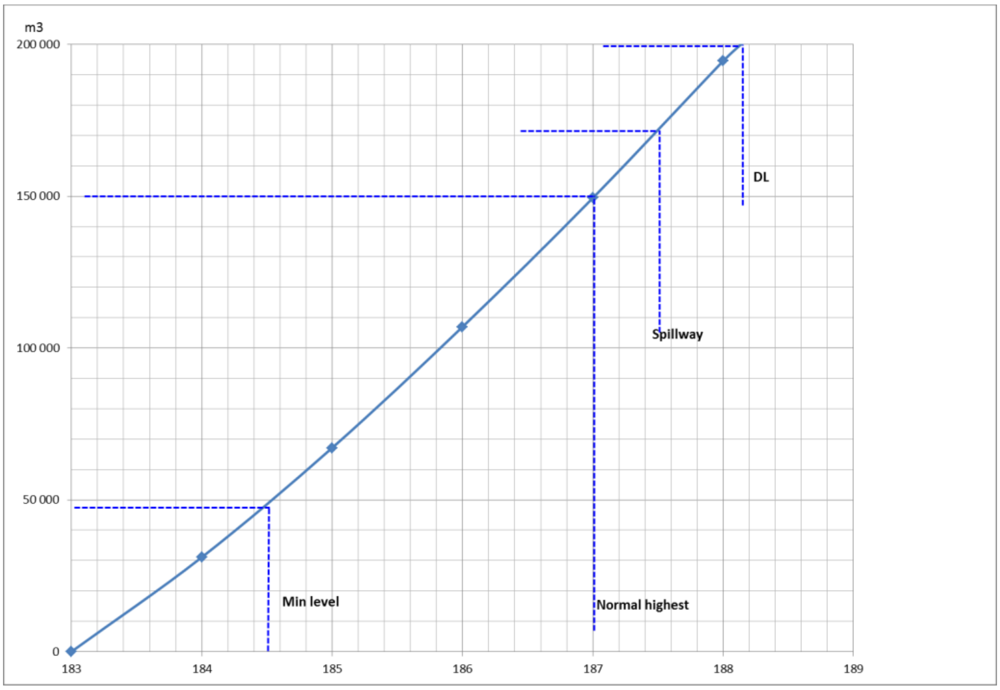


Figure 30. Volumes of water in the north clarification pond at different levels (masl) together with operational levels.

Table 4. Volumes of water in the south clarification pond at different operational levels (masl)

	+Level	Volume (m^3)	Buffer volume (m^3)
Min level	182,5	100 000	
Normal highest	187	880 000	780 000
Spillway	187,5	1 000 000	
Dam limit	188,15	1 120 000	

Table 5. Volumes of water in the north clarification pond at different operational levels (masl)

	+Level	Volume (m^3)	Buffer volume (m^3)
Min level	184,5	46 000	
Normal highest	187	150 000	104 000
Spillway	187,5	167 000	
Dam limit	188,15	200 000	

7.2.5 Water balance method and cases

The site water balance has been prepared on a monthly basis with year to changes primarily when the major changes take place in the operation. The northern clarification pond is first taken into operation beginning of production year one when it is filled up and ramp-up of production assumed to start in the middle of the year. The larger southern pond is taken into operation and filled up at the beginning of year 7 from which on the northern pond is used as long as needed (during rehabilitation of the north TMF).

The water balance includes all the water flows at the site to and from the clarification ponds with seasonal variations of these including need of raw water and discharge of excess water.

Sensitivity analysis has been done for the input data primarily impacting the result of the water balance. This includes dry and wet years of the precipitation and surface run-off and changes of the tailings solids content to 40 %-w/w for the wet year (resulting in the highest flow of drainage water during spring flood) and 50 %-w/w for the dry year (resulting in the highest process make-up water need).

7.2.6 Results and conclusions

Results of the site water balance are presented at year 1-3 (result of years 4-6 is the same as year 3) during ramp-up and use of the northern dam areas only and at year 7-8 (result of years 9-17 is the same as year 8) when the south dams are taken into operation. A special case where only the northern clarification pond is used throughout life of mine is also presented.

Flows to and from the clarification ponds are presented for a normal year in Figure 31.

In figure 32 the different tailings water flows are presented at nominal throughput and 45 %-w/w of solids.

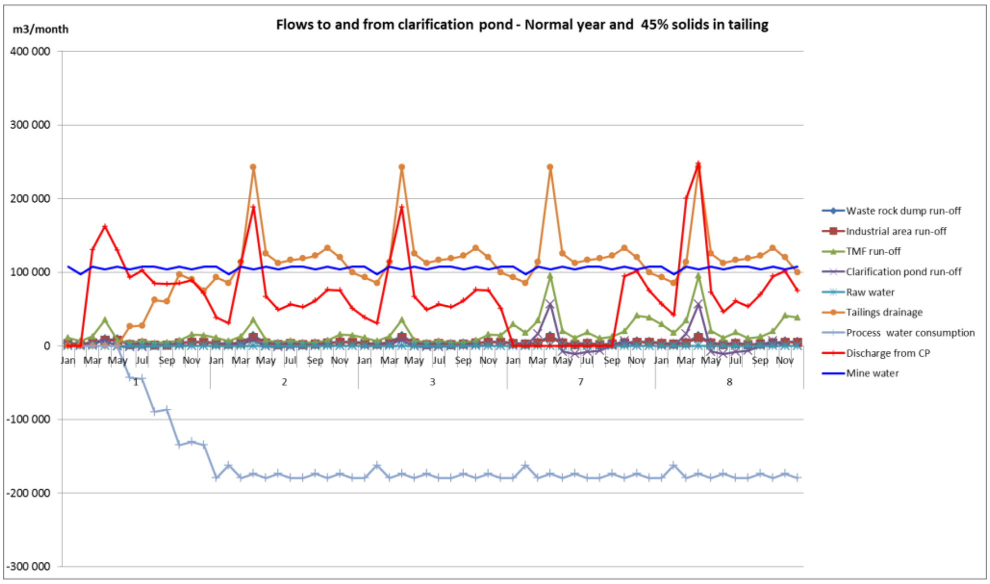


Figure 31. Flows to and from the clarification ponds.

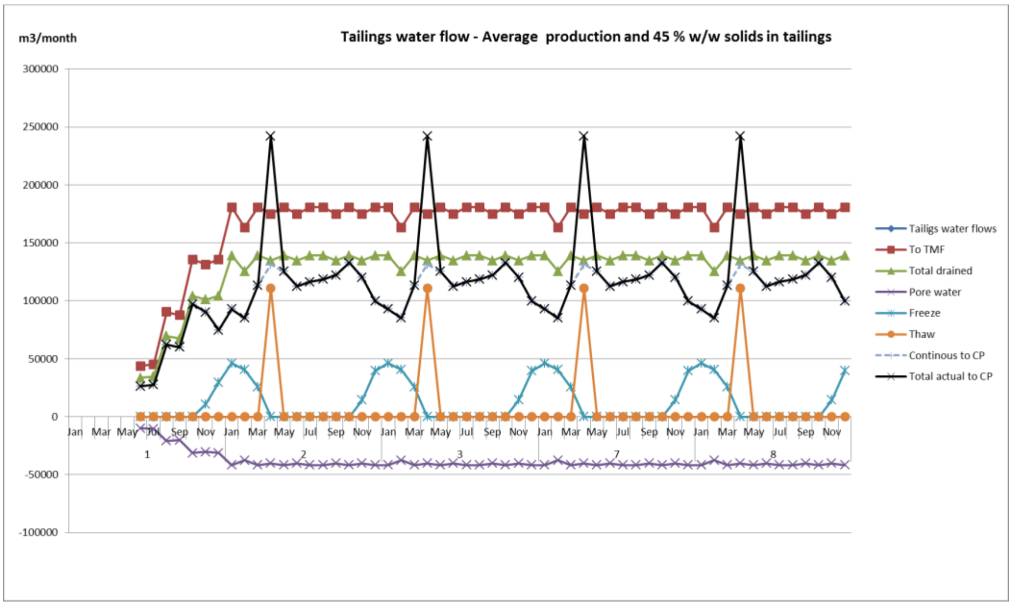


Figure 32. Tailings water flows.

Results of the site water balance for the average normal year with 45 %-w/w solids in the tailings is presented in Figure 33. In Figure 34 result the normal year and using only the northern clarification pond is presented.

Results of the site water balance for the wet year with 40 %-w/w solids in the tailings is presented in Figure 35.

Results of the site water balance for the dry year with 50 %-w/w solids in the tailings is presented in Figure 36.

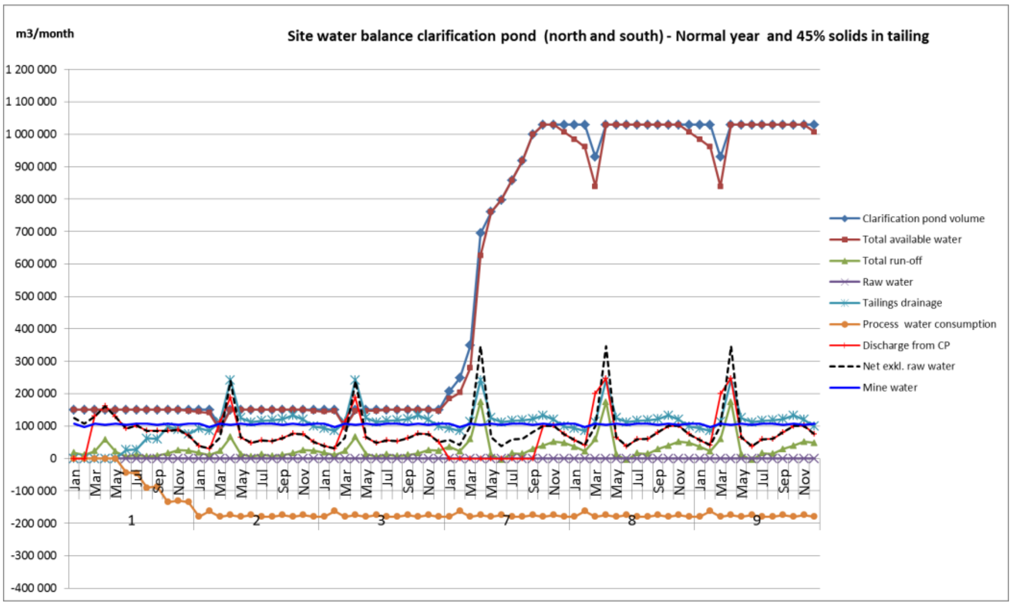


Figure 33. Results site water balance normal year 45 %- w/w of solids in the tailing and use of first the north pond and from year 7 also the south pond.

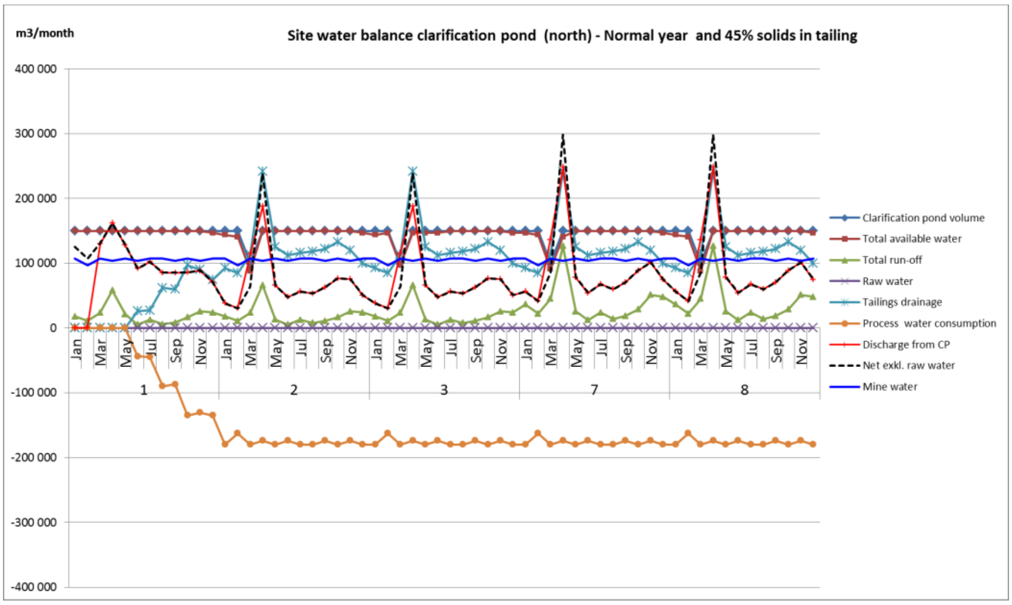


Figure 34. Results site water balance normal year 45 %- w/w of solids in the tailing and use of only the north pond through-out life of mine.

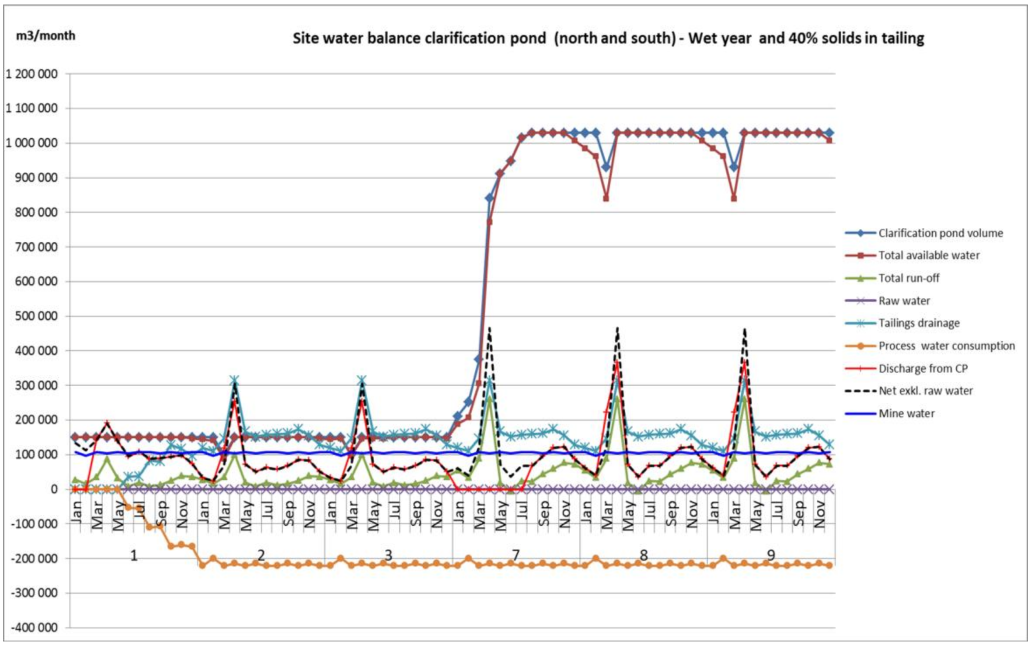


Figure 35. Results site water balance wet year 40 %- w/w of solids in the tailing and use of first the north pond and from year 7 also the south pond.

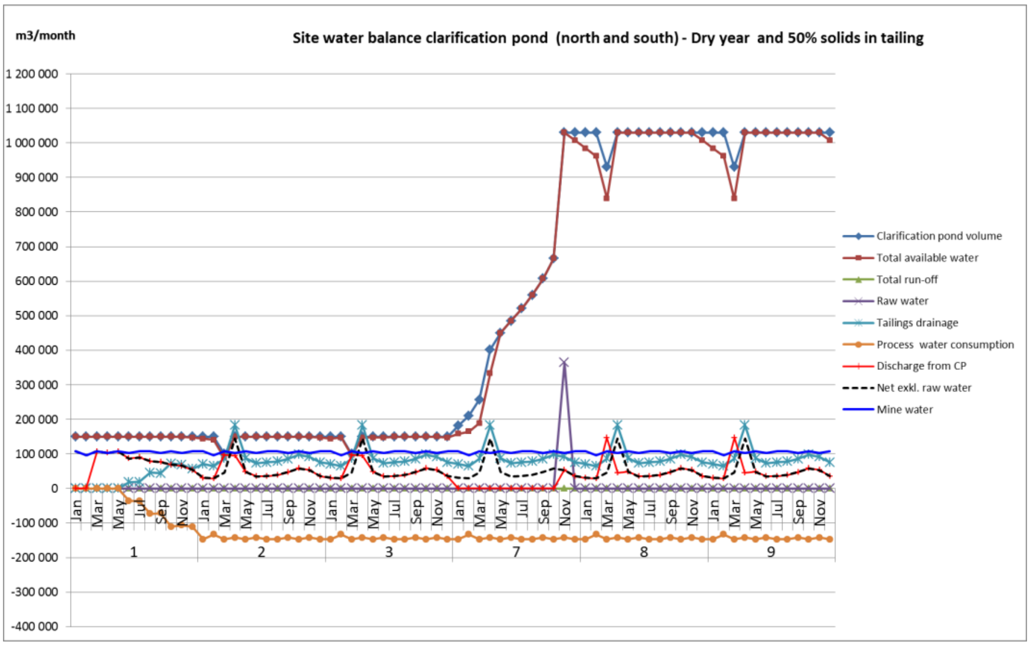


Figure 36. Results site water balance dry year 40 %- w/w of solids in the tailing and use of first the north pond and from year 7 also the south pond.

From the results it can be seen that for all cases the net water is positive all year around meaning a need to discharge water. This mainly because of the ground water from the shaft being pumped to the clarification ponds. For this reason the volume of water in the ponds are basically at or near the normal highest level et all time. This mean that no raw water needs to be pumped from Lake Väsman, except if the pumps ore pipeline from the clarification pond fail or break down. Since water always is needed to run the process it is still wise to have a pipeline from Väsman as back-up.

During spring flood in April around 400 000 m^3 (555 m^3/h in average) is discharge from the pond for the wet year.

As can be seen from figure 33 when the north clarification pond alone is used during life of mine the results of the water balance is basically the same as when the south pond is used; i.e. net excess of water all year round and the volume of the pond staying within the operational levels. This means that from a water supply point of view it would be sufficient only to operate with the north clarification pond and still be able to run the process. However the surface area of the north pond is quite small meaning reduced capacity to treat and remove sediments from the water especially when the larger south TMF is taken into operation. For this reason it is assessed that the south pond still is needed and will give both better buffer volumes and treatment capacity of TMF drainage water.

8. Deposition strategy

8.1.1 Tailings management

The deposition is based on conventional hydraulic deposition with segregation of the tailing. The coarse particles are settled near the spigots of a starter dam forming a beach and the finer particles are settled further away. Normally a very low beach slope is achieved around 0,5%.

The beach is used as part of raising the dam embankment upstream reducing the amount of material needed for support/erosion protection and filters , as compared to raising the embankment up or down stream, for the available dam and pond foot prints. See Figure 37 for a conceptual presentation.

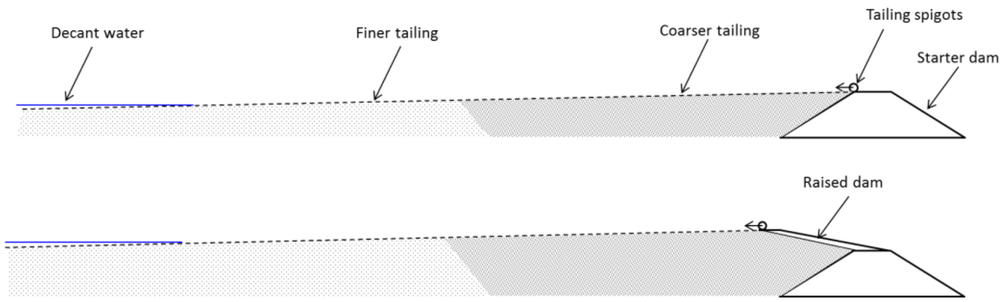


Figure 37. Conceptual figure of hydraulic deposition with segregation of the tailing, including raise of the dam.

Several spigot are used along the dam crest and these are moved gradually upwards as the height of the tailings stack increases and along the dam perimeter to get an even distribution of the tailing.

Challenges with the management of the tailings include to sufficiently segregate the coarse from the fines and get the fines as far as possible from the outer dam embankment since it has inferior stability properties and can increase the phreatic water (due to lower hydraulic conductivity) surface which needs to be as low as possible near the outer dam embankment. Segregation is mainly controlled by:

- the PSD of the tailing (very favourable for NIO tailing),
- the solids content if the tailing (selected conservatively based on the lab test work)
- The flow and speed by which the tailing exits the spigots. This needs to be optimised during operation but normally higher speed results in better segregation.

Berms of coarse tailing can also be constructed closer to the spigots if the flow of tailings and settled coarse material need to be diverted. Dozers are used if needed to distribute the tailing and when raising the outer rock layers.

Another challenge in this case with the NIO TMFs is that spigotting will take place from opposite sides of the perimeter dams at both TMF. This means that potentially fine tailing from one side can flow over and settle close to the other embankment (Figure 38). However fine tailings cannot be allowed as a layer under the coarse tailing this close to the embankment as it can potentially can act as a slip surface and increase the risk of dam failure. Also the decant water (containing fine sediment and increasing the phreatic level) needs to be controlled and prevented from gathering to close to the other embankment. Both these issues are managed by carefully controlling the height of the lift on either side (Figure 39), i.e. moving the spigots well in time before the height one embankment get so high. Berms inside the TMF can also be constructed to stop the flow of fine tailings and water.

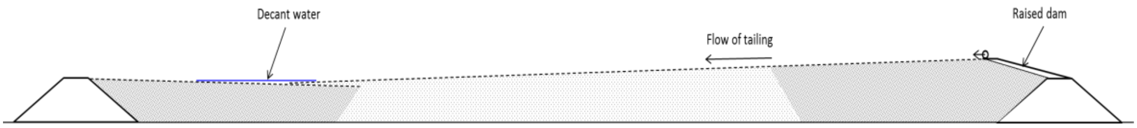


Figure 38. Potential risk with fine tailing from the ongoing spigotting flow to far over on the other side and on the coarse tailing.

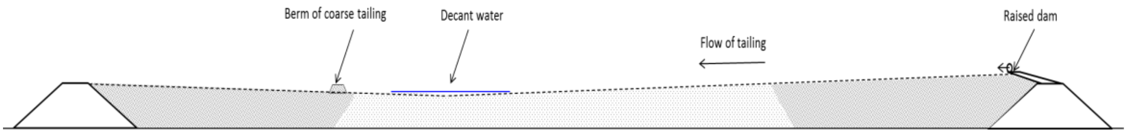


Figure 38. The heigt of the lift on one side controlled to confine the fines to the middle and use of berms if necessary.

8.1.2 Start-up and sequencing of the TMFs

Studying the total volume of tailing both TMFs are needed to contain the volume that will be produced during life of mine within the TMF foot prints. The storage capacity in the north TMF is about 5,5 (M)m^3 and in the south TMF 12,5 (M)m^3. This give a total capacity of 18 (M)m^3. This can be compared to the total tailings for the planned production of 15,5 (M)m^3 and the design amount of 17 (M)m^3 (in both cases using a conservative consolidated density), which consequently mean that the TMFs have a sufficient storage capacity for the life of mine.

There are two alternatives for start-up of operation of the TMFs, either starting with the northern and when it is full continuing with the southern or vice versa. As discussed in section 13 from a Capex cost point of view the best option is to start with the northern TMF which consequently is the preferred and chosen option.

Figure 40 and 41 present the gradual increase of volume and elevation of the tailing year by year in the northern and southern TMF. It will take around 6-7 years until the level in the northern TMF reaches the maximum height of +210, after which the southern TMF is taken into operation beginning of year 7. The southern TMF will be operation for around 10 years.

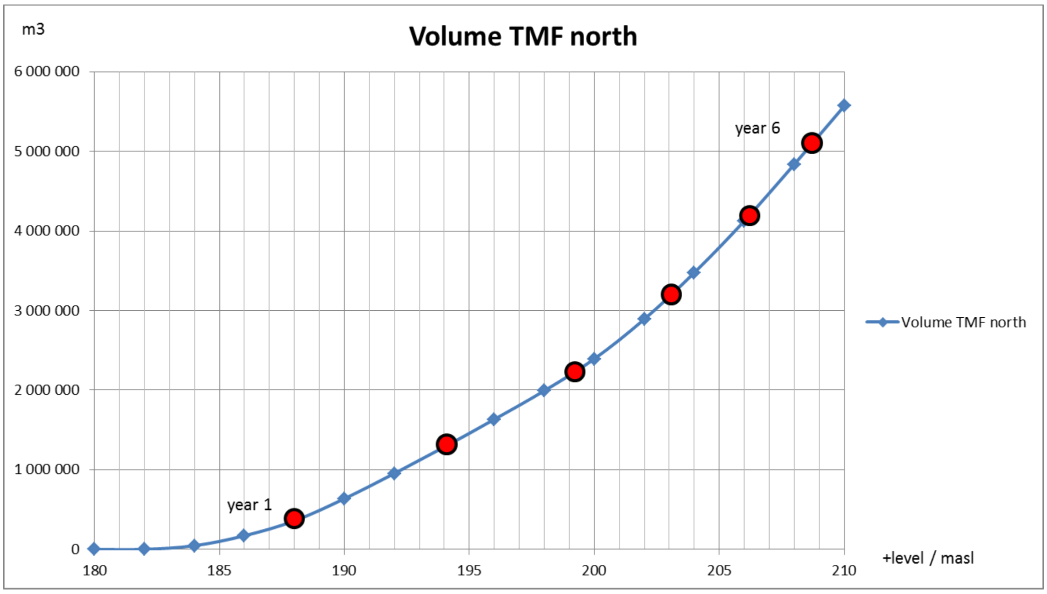


Figure 40. Increase of volume and level year by year in the northern TMF.

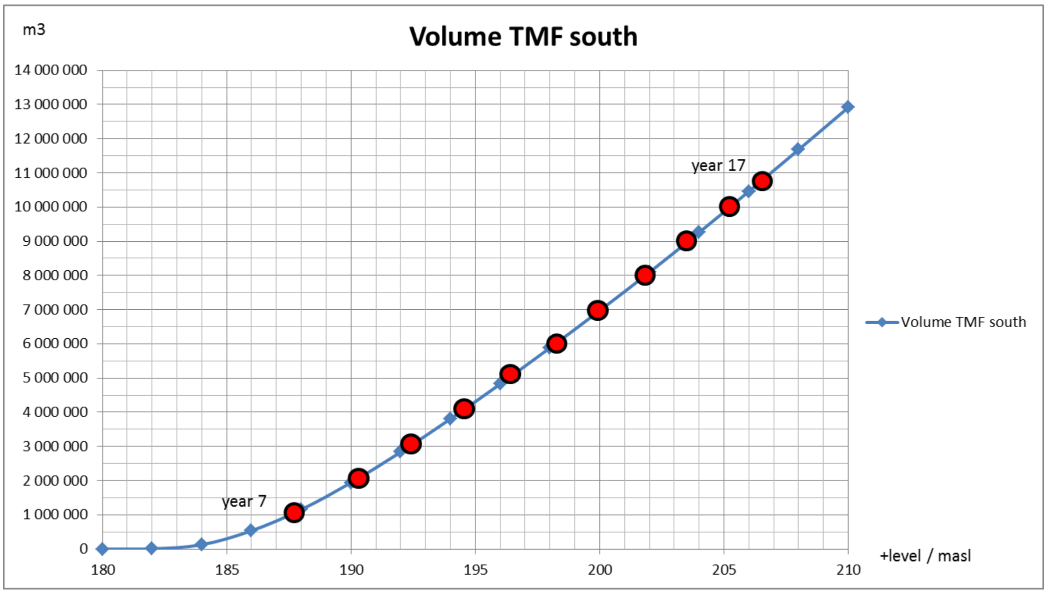


Figure 41. Increase of volume and level year by year in the southern TMF.

9. Dam design

9.1 Site selection

The location of the TMF and clarification pond is more or less the same as in the PEA and what is described in the environmental permit. Since the dams are permitted it mean only small changes of the dam foot print can be made, part of the optimisation in of this study or in future detailed design.

The TMF areas was selected both to continue the deposition on the existing tailings in the northern area, to minimise the land take of virgin ground and make use of the natural elevation to minimise constructed embankments and the size of these, and thus to minimise Capex and Opex. Another reason was the relatively short distance to the process plant. In the northern TMF no dams are needed in the eastern section where the natural elevation forms the containment of the tailing. In the southern TMF no dams are in the same way needed in the north section and in short sections on the eastern side.

The clarification ponds are located in the lowest topographical part of the dam areas next to the canal in a natural depression consisting of small wetland and lake. This way the volume of water in the ponds are maximised with as low constructed embankments as possible. In the north eastern sections of the ponds no dams are needed since the natural elevation forms the conefinement of the water.

9.2 Dam plan layout

9.2.1 Clarification pond

As discussed in section 7.2.6 (water balance) both ponds areas are needed primarily to treat the drainage water from the TMFs containing fine sediments before the water can be discharged or pumped to the process plant or the recipient.

In this study some optimisation and minor changes to the dam layout have been done. The first dam foot print designed part of the PEA study was produced using 5 m contour map while the latest terrain map, also used as a basis for the design work part of the environmental application, have elevation details of less than half a meter.

In the northern clarification pond the south western section of the dam have been moved about 120 meters towards northeast (Figure 42) since this area of the dam did not contribute with any significant volume of water (elevation above normal operational level) and at the same time the length of the dam section has been reduced. In the northern section the natural elevation forms the confinement and no dam is needed. The total length of the dam embankment is around 1100 meter

(Figure 43), of which 450 m is shared with the northern TMF. The foot print inside the pond is 40 000 m^2.

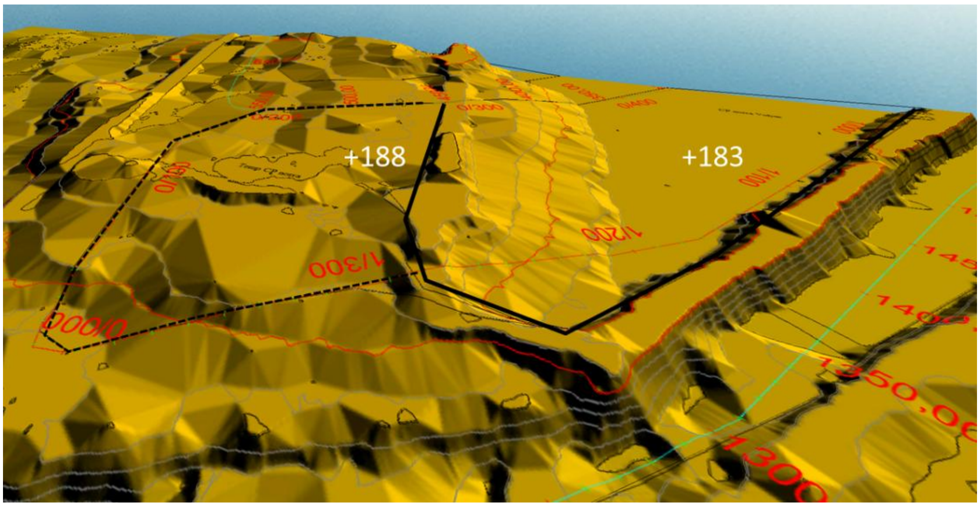


Figure 42. Northern clarification pond. Dotted section removed since not contributing with any volume (normal highest operational level at +187,2).

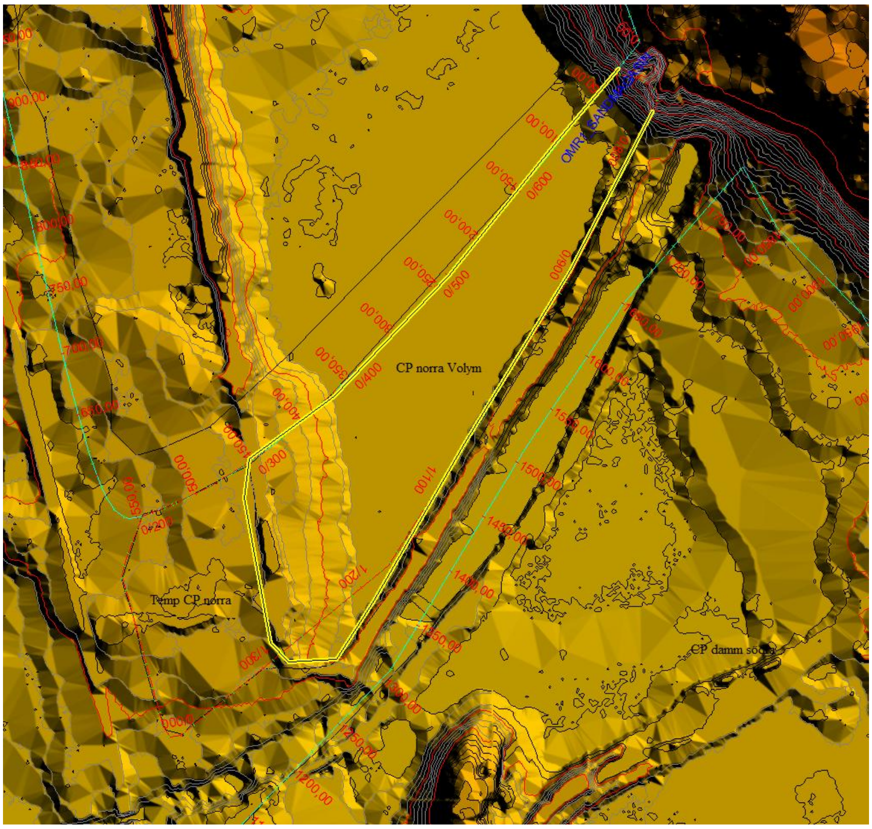


Figure 43. Plan view of northern clarification pond embankment (yellow). Removed section in the south marked with black/red line.

The northwest dam section of the southern clarification pond have been moved slightly more south east since it before was located in the canal (Figure 44). In the northern section the natural elevation forms the confinement and no dam is needed. The total length of the dam embankment is around 1800 meter, of which 850 m is shared with the southern TMF. The foot print inside the pond is 300 000 m^2.

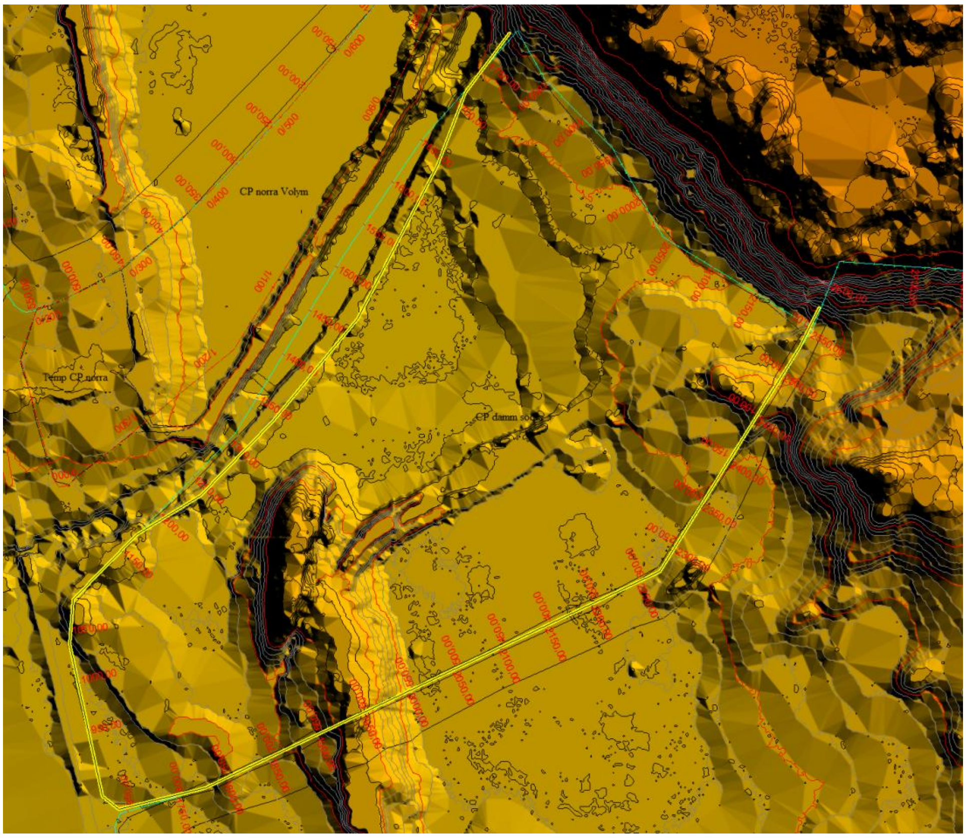


Figure 44. Plan view of southern clarification pond embankment (yellow) with previous location on the western section indicated with green line.

9.2.2 TMF

In this study some optimisation and changes to the dam layout have been done. The first dam foot print part of the PEA study was produced using 5 m contour map while the latest terrain map, also used as a basis for the design work part of the environmental application, have elevation details of less than half a meter.

9.2.2.1 TMF north

In the northern TMF the natural elevation forms the confinement in the eastern section and no dam is needed (Figure 45). The total length of the dam embankment, if the same layout as has been used in the PEA and permit application, is around 1950 meter of which 450 m is shared with the northern clarification pond in the south section. The foot print inside the dams is 420 000 m^2.

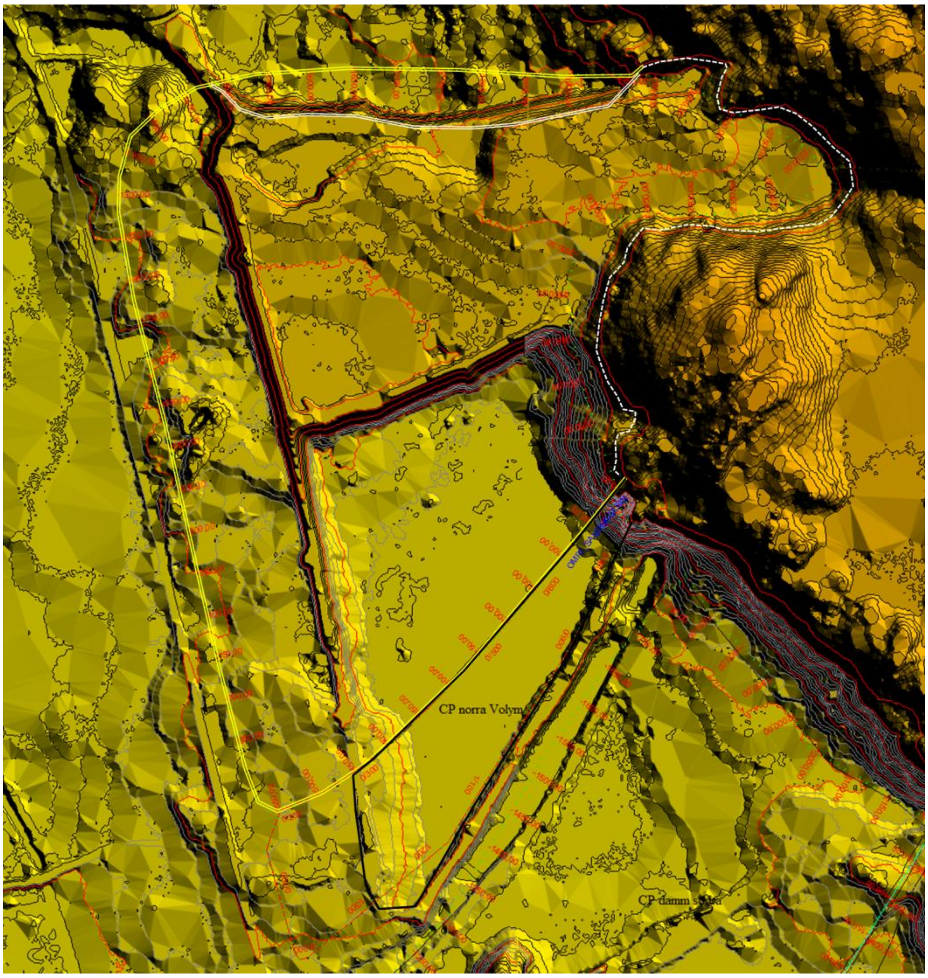


Figure 45. Plan view of northern TMF embankment (yellow) with clarification pond embankment in black. Potential change of dam location in north indicated with with line. White dotted line illustrates the confinement of the tailings by the natural elevation.

An issue with the layout in the northern section of the TMF is the location of the dam behind the old tailings stack. This means that you basically construct a dam behind an already existing "natural" confinement. In figure 46 a 3D-view towards east of the northern part of the TMF is presented with the current dam layout and a suggested change of layout (showed conceptually).

One thing that has to be investigated further is if suitable to construct a starter dam on the old tailing, e.g. considering the PSD, fine layers close to surface etc. The cross section of the dam might in have to be changed slightly.

Further the volume that can be deposited on top of the old tailings stack is relatively low since the surface in average is located at around +202 masl (varying from +198 to +207). This means that the cost of constructing and raising a dam at the current location as compared to the volume of tailing possible to deposit is relatively high.

An additional issue with the current dam layout is management of the tailing and water which is discussed under section 10.

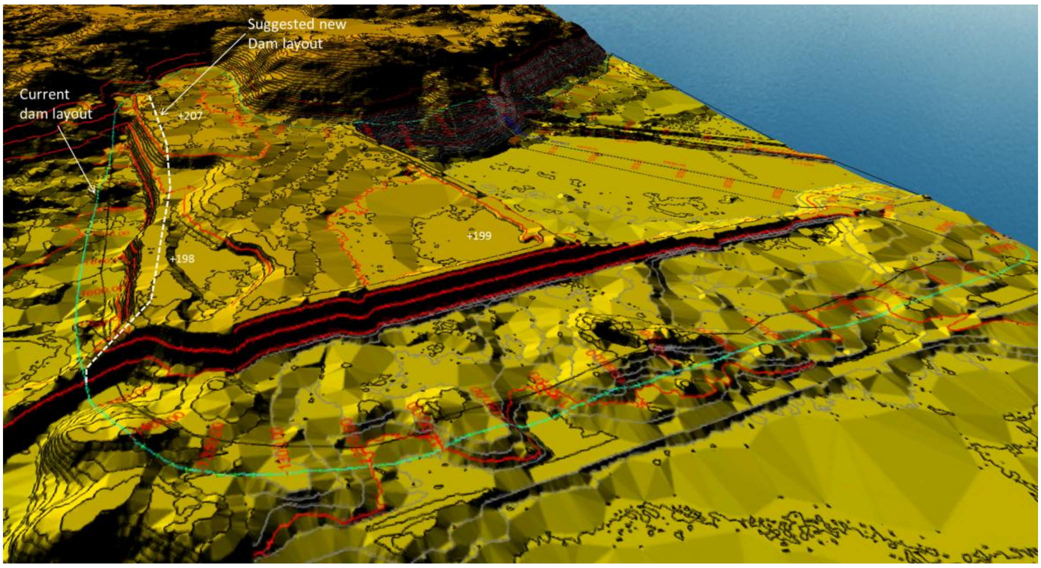


Figure 46. Plan view of northern part of the northern TMF with current dam layout in blue and suggested change in white.

9.2.2.2 *TMF south*

In the southern TMF it has part of this study been assessed that no dams are needed in a few short sections on the eastern side where the natural elevation can be used as confinement (Figure 47). The natural elevation also forms the confinement in the eastern section.

The total length of the dam embankment is around 2900 meter, of which 850 m is shared with the southern clarification pond. The foot print inside the dams is 720 000 m^2.

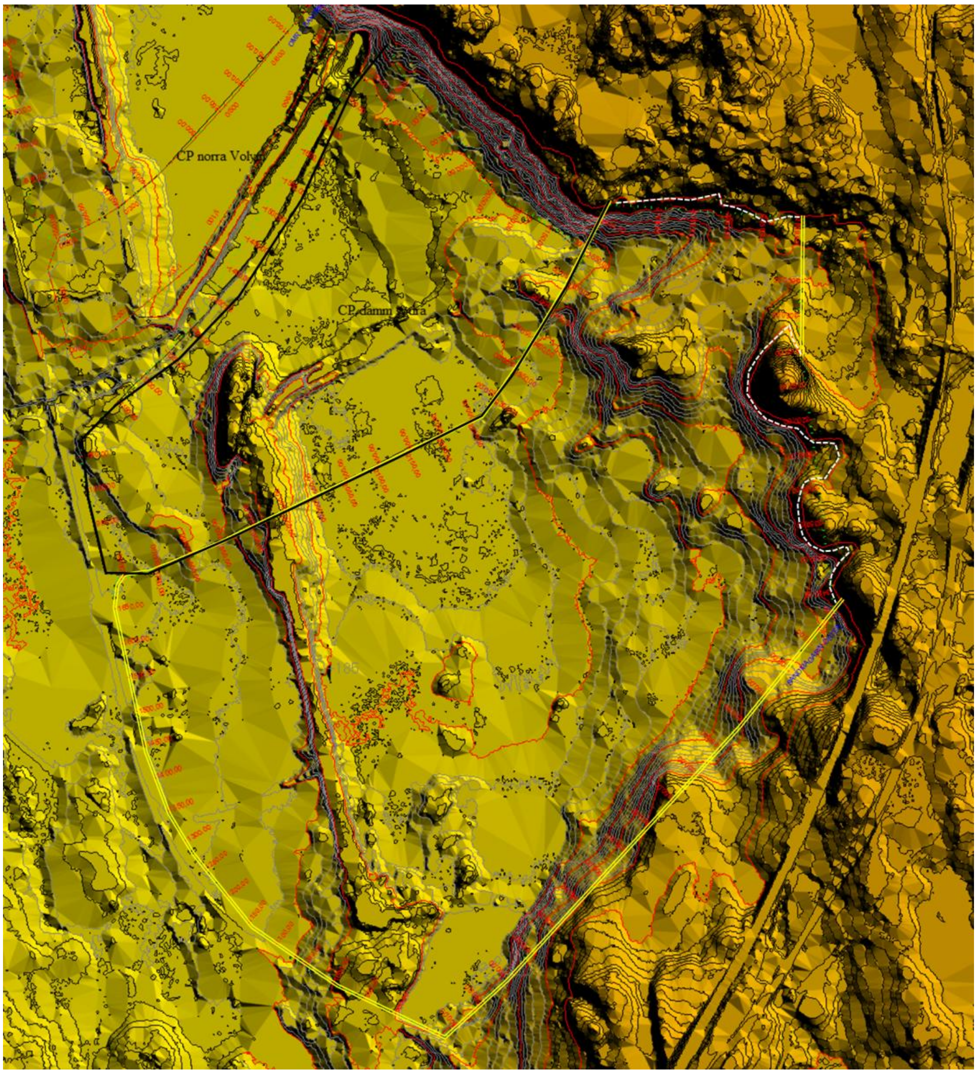


Figure 47. Plan view of southern TMF embankment (yellow) with clarification pond embankment in black. White dotted line illustrates the confinement of the tailings by the natural elevation.

9.3 Embankment Design

Same embankment design will be used for the north and south areas.

9.3.1 Clarification pond

The cross section of the clarification pond embankment consist of a zoned sloping moraine core (crest of the moraine core at +189,15 masl which is 1 m above the water dam limit), with supporting rock fill of blasted rock on the downstream side (Figure 48). The moraine core is surrounded by fine and coarse filter layer to prevent inner erosion of the core. Under the rock fill on the downstream side a horizontal drainage filter blanket is placed to reduce the phreatic surface and pore pressures in the embankment but also to prevent transportation of fines from the sub-structure to the rock fill. On the upstream side supporting rock fill and erosion protection are placed and on the downstream side also a toe rocks fill is placed.

See also the section below for dam in-between TMF and clarification pond.

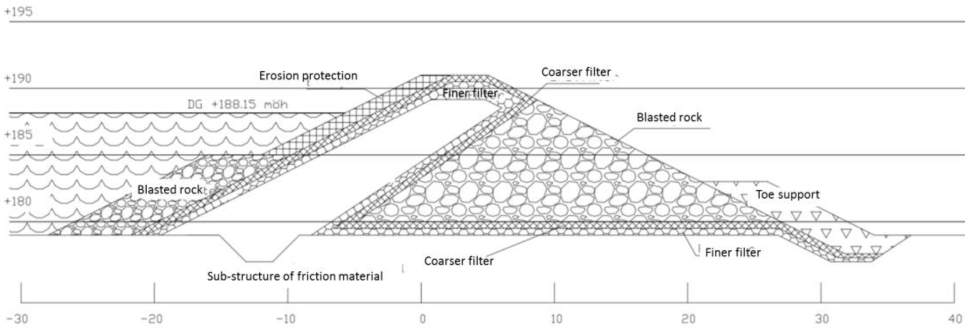


Figure 48. Cross section of the clarification pond dam embankment.

9.3.2 TMF

9.3.2.1 TMF section

The TMF dam embankment is constructed as a drained embankment of rock fill and filters.

The dam is constructed and raised in three stages for the northern TMF. The southern TMF is raised in two stages.

First a starter dam is constructed at the northern TMF up to +186 masl where needed (i.e. natural elevation below +186 masl), consisting of a supporting structure of rock fill (blasted rock) on the downstream side with coarse and fine filter layer on the upstream side (Figure 49) both on the rock fill and as horizontal filter blanket under the tailing. On the filters coarse tailing is placed to protect the filters from erosion during spigotting. The filter layers lower and reduce the gradient of the pore pressure so ensure that the gradient is not steeper than half the friction angle. Spigotting start from the crest of the dam.

During the second stage (first stage for southern TMF) the starter dam and supporting structure with filter layers are raised to full initial height at +190 masl (Figure 50). On the downstream side a supporting toe support of rock is placed. In the sections where the natural elevation is above +186 masl a starter dam also constructed up to +190 masl at this stage.

RAMBOLL

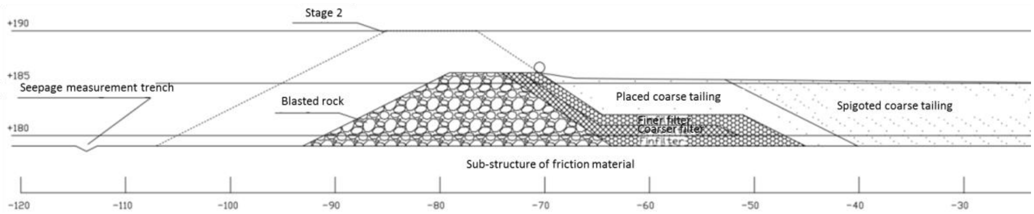


Figure 49. Cross section of the TMF dam embankment stage 1.

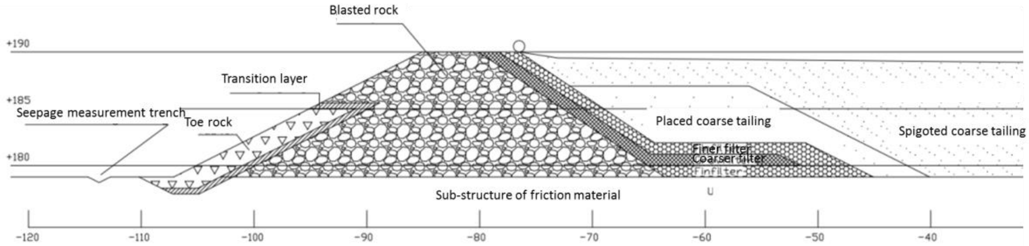


Figure 50. Cross section of the TMF dam embankment stage 2.

At the third stage the dam is gradually raised upstream to the final level of +210 masl using the spigotted coarser tailing as foundation (Figure 51). The outside of the embankment is protected with an erosion protection layer, and up to level +200 masl and filter a layer placed in-between the tailing and the erosion layer. The slope of the embankment from this stage on 1H:6V.

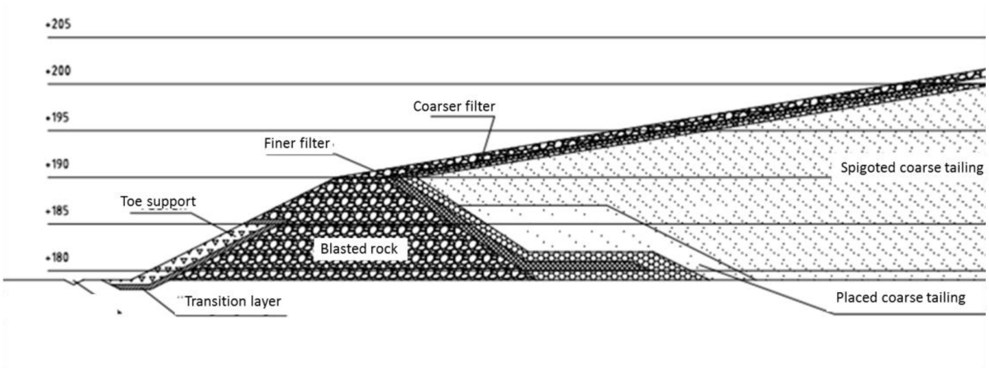


Figure 51. Cross section of the TMF dam embankment stage 3.

9.3.2.2 *Joint dam in-between clarification pond and TMF*

The dam embankment in-between is basically constructed in the same way as the clarification pond dam with a zoned sloping moraine core (up to level +189,15 masl as the clarification pond dam and with at crest at +191,0 masl) and a rock fill supporting structure. The difference is that the horizontal drainage filter blanket has been replaced with an upstream sloping filter (fine and coarse filter layers) on the rock fill and a horizontal filter blanket (fine and coarse filter layers) is used under the tailing to lower the phreatic surface as for the TMF embankment

(Figure 52). Tailing is also placed on the horizontal drainage filter to protect it. The dam is raised upstream in the same way as the TMF embankment.

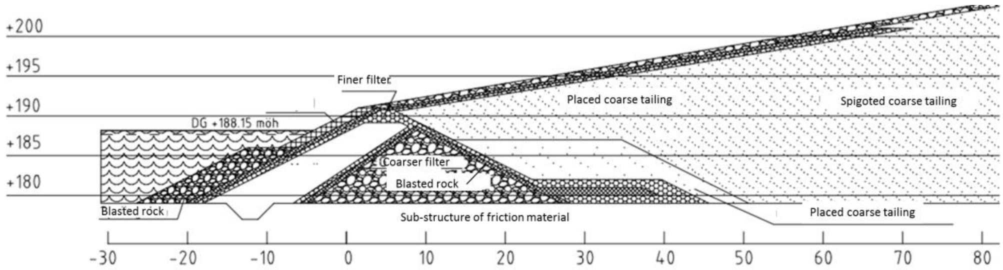


Figure 52. Cross section of dam embankment in-between the TMF and clarification pond.

One aspect that needs to be further studies is if this dam section can be constructed as a non impervious embankment (i.e. a rock fill embankment with filter layers) since water at least initially will be stored on either side of the dam. Also when the dams are raised this concepts might be possible to use since water need to seep to the clarification pond anyway. This concept would make the excavation and of filling in the wetland much easier.

9.3.3 Foundation/Ground Preparation Requirements

The guidelines given in RIDAS and GruvRidas are fairly general in terms of how an embankment founded on soil should be designed. With regard to internal erosion, it must be verified that the hydraulic conductivity of the foundation is similar to that of the impervious element in the embankment. If the embankment is founded on more permeable soil than the core material (e.g. in sections where bands of loose gravel are present), measures must be taken to ensure that the hydraulic conductivity of the foundation and the contact zone between foundation and embankment is reduced to that of the core material (RIDAS, 2012).

The main part of dams can be founded on solid soils without reinforcement.

The sections of both the TMF and the clarification pond dams located in the wet areas next to the canal (see Figure 1 and 2) consist of loose sediments e.g. peat and loosely layered fine moraine. The extent and exact depth of these loose sediments has not been investigated in detail but indications of the depth have been determined in the previous site geo-technical investigation.

Since the dams cannot be founded on these sediments they need to be excavated and removed before the embankments are constructed (see Figure 53). The excavated material is replaced with friction soil material, e.g. moraine of the same type as in the dam core.

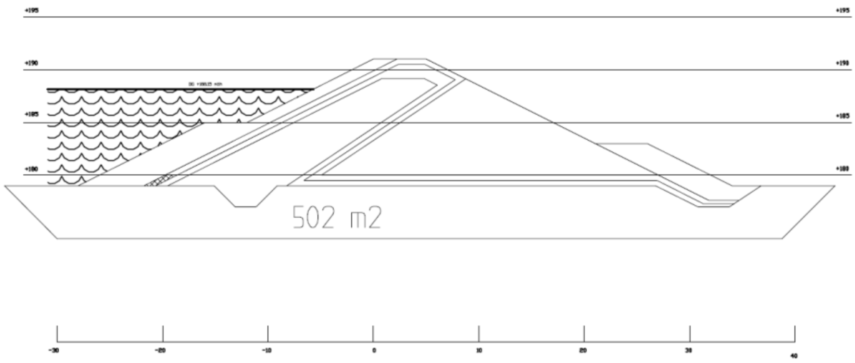


Figure 53. Excavation of loose sediments and re-placemed with moraine.

Further where the dam is founded on moraine the top vegetation layer (about 0,2 m) is removed before the dam is constructed.

9.4 Construction materials and requirements

9.4.1 Borrow Materials

9.4.1.1 Moraine

It is envisaged that the moraine to be used for the construction of the embankments will be borrowed within the inundation areas, most likely the ridge within the southern clarification pond, and also from areas to be identified in the proximity of the dams. The old TMF can also contain suitable material.

Since no trial pits and lab test work of the moraine properties have been undertaken to study if suitable moraine nearby the dams be sourced that fulfil the criteria given in RIDAS for core materials in embankment dams, this is of high priority in future studies. If the moraine need to be taken from sources further away construction costs will increase.

9.4.1.2 Rock material

Rock material for the supporting structure, filter layers and erosion protection will, if possible, be borrowed either from the industrial area or from within the inundation areas (ridge within the southern clarification pond or the north part of the dams). The old TMF can also contain suitable filter material.

When production has started waste rock can likely be used to produce the rock material needed for the dam constructions.

In future studies the availability of rock material, both volumes available from different sources and the quality from these, need to be studied further since they can have a big impact on the costs. If no suitable material is available, material will need to be borrowed from external sources which will increase costs considerably.

9.4.2 Core Material

The core material will be sourced from the glacial till (moraine) material. According to RIDAS (2012), for a material to be used as part of the low permeability core, the maximum amount of fine particles < 0.06 mm is recommended to range between 15 to 40 % of the material passing 20 mm. The maximum amount of material passing 2 mm is recommended to be less than 85 %.

9.4.3 Supporting rock structures, filters and erosion Protection

The availability of suitable filter and similar finer materials in the area may be limited. It may be possible to crush and screen materials from local borrow areas at the dam location or industrial area and from waste rock during operation to create suitable filter and erosion protection materials. This has been used as the base case for this study. The possibility of using local materials as filter material must be investigated at the next stage of design and feasibility study with support from adequate and targeted geotechnical site investigations. If material especially for filters need to be sourced from an external source the cost of the material will increase significantly.

RIDAS provides different filter criteria dependent on the gradation of the base material used for the impervious moraine core in the dam and the coarse tailing. The exact specification of the different filter materials and transition layers have not been assessed part of this study and will be investigated at the next stage of design.

The embankment will be protected by an erosion protection which in the same way needs to follow specific criteria in RIDAS, to avoid that parts of the erosion protection are ripped off with varying water levels and ice formation during winter (RIDAS, 2012), however not as stringently. If the requirement of the erosion layer is set lower it is compensated for by more regular supervision and maintenance.

9.4.4 Placement and Compaction

All imported and excavated materials for the dam embankments will be subject to stringent inspection, sampling and testing to determine the suitability of the materials for construction. Strict quality control will be exercised to ensure that the materials are placed to the required lines and levels, grading, moisture content and density.

9.5 Spillway Design

9.5.1 Introduction

Spillways are constructed with sufficient capacity to evacuate the peak flow induced by storm events to ensure that the highest water level will not be exceeded and overtopping of the dam is avoided at all times. Spillways are constructed both in the TMFs and clarification ponds.

The spillways in the clarification ponds are "emergency spillways" which aren't used during normal operation. The spillways in the TMFs both divert run-off and tailings drainage water to the clarification ponds during normal operation as well as peak flows. The locations of the spillways are conceptually showed in Figure 54.

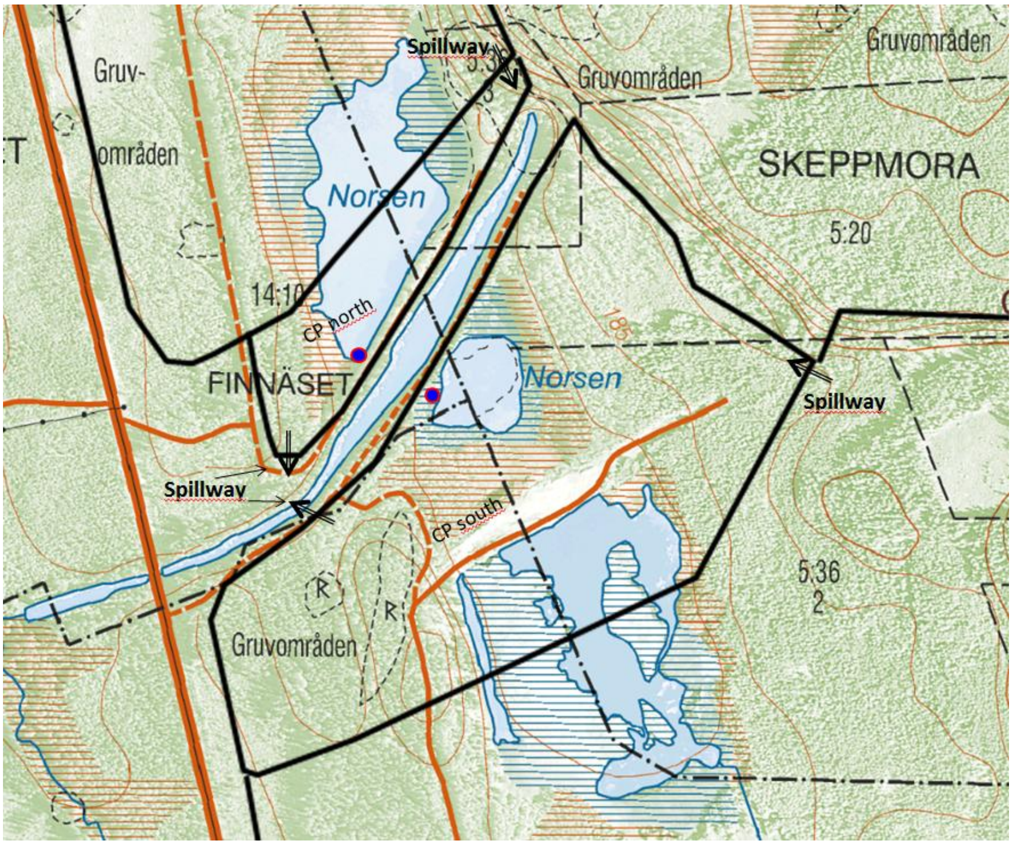


Figure 54. Conceptual location of the spillways on the TMFs and clarification ponds.

The design flow and capacity of the spillways have been assessed and presented part of the environmental permitting process. In this study only a very short description is made of the design data.

9.5.2 **Design**

The design flows are based on the Flood Design Category II (Svensk Energi, 2007), i.e. a rainfall with a return period of 100 years (24-hour value).

9.5.2.1 Clarification pond

The capacity to spill water is highest at the clarification ponds and especially the southern pond, since the capacity of the spillways need to take into account both the surface area of the TMF and clarification pond.

The spillways will be constructed as an open channel cuts in the abutment of the embankments. The spillways are constructed with trapezoid shape (Figure 55) to increase the capacity to spill water. The spillways will be constructed with a combination filter layers, erosion protection layer and concrete threshold.

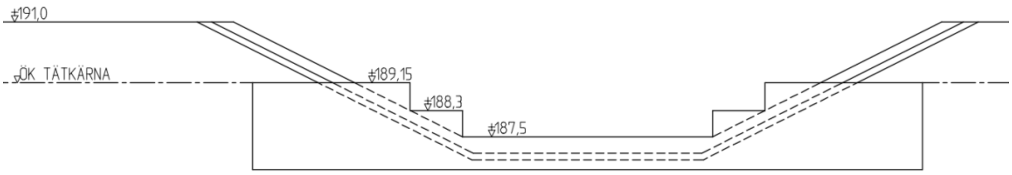


Figure 55. Conceptual drawing of clarification pond spillway.

9.5.2.2 TMF

The spillways from the TMFs to the clarification ponds are constructed as blasted canals in the side of the mountain and with a concrete threshold at the inflow of the spillway (Figure 56). These spillways are moved up as the TMF embankments are raised (Figure 57).

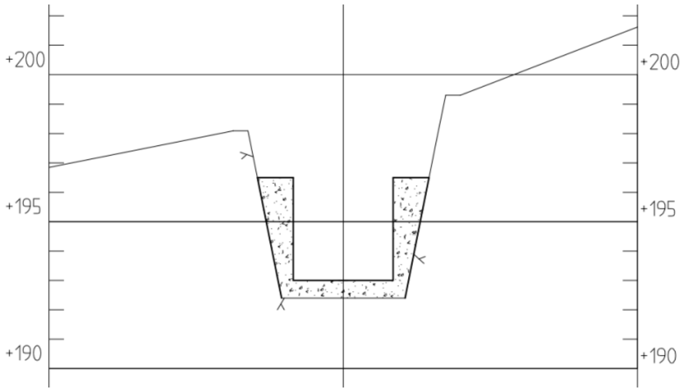


Figure 56. Conceptual drawing of TMF spillway.

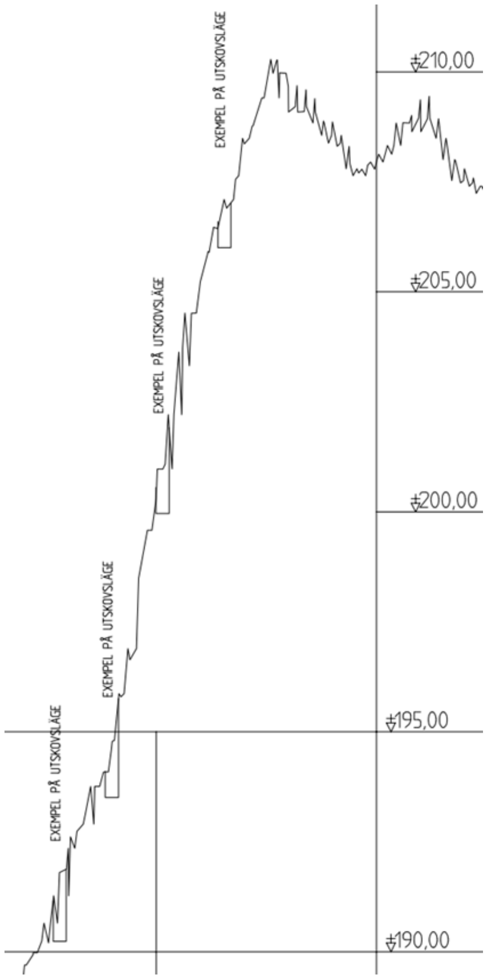


Figure 57. Conceptual illustration of the TMF spillway gradually raised to new higher level as the tailing is deposited.

9.6 Stability Analysis

The stability of the embankment has been analysed using Slope/W, version 7. Methods used are all based on the limit equilibrium formulations used to compute the factor of safety (FoS) of the dam wall. The stability analysis was carried out using effective shear strength parameters. Table 6 summarizes the different load cases and required safety factors according to RIDAS.

Table 6: Required FoS for Different Load Cases according to RIDAS (2012)

Load Case	Description	FoS
1	Completed construction, before filling of magazine	1.5
2	Normal operating conditions, stationary flow through the embankment	1.5
3	Extreme operating conditions	1.3
4	Rapid draw down	1.3

RIDAS/GruvRIDAS has some requirements for the FoS for different set up of the tailings material properties. For drained static load case the FoS required to be 1,5 and for the load case with un-drained shear strength at failure the FoS need to be 1,3. For the load case with liquefaction residual shear strength of the finer tailings is required to have FoS of 1,1.

Based on the results of the lab test on the tailings only the seepage calculations and the load case liquefaction have been recalculated using the new material properties. All other previous assessed cased remain the same.

9.6.1 Material Properties

The material properties used in the stability analysis are presented in Table 7 below.

Table 7: Material Parameters for Stability Analysis

Material Type	Material Parameters				
	Density [kN/m^3]	Angle of Friction [°]	Cohesion Liquefaction [kPa]	Total Cohesion [kPa]	Total Angle of Friction [°]
Talings spigoted (coarser)	19	See chapter 5.5.5 Shear strenght	-		
Tailings spigoted (finer)	19	See chapter 5.5.5 Shear strenght	5		
Coarser tailings worked	19	28	-		
Foundation (till)	21	35	-		
Finer filter	19	34	-	-	-
Coarser filter	19	38	-	-	-
Till (coare)	19	34	-	-	-
Rock fill	18	40	-		
Erosion protection	18	43	-		

The properties detailed above are partly based on results from laboratory campaign and literature.

The embankments will be for the biggest part founded on glacial till, which is basically the same material as in the moraine core. Hence, the calculations were carried out with the same parameters for both layers.

9.6.2 Analyses and Results

9.6.2.1 Clarification pond

A slope stability model was developed for the typical section of the embankment. In the PEA three analyses were carried out: one with stationary conditions with the maximum operating water level (HHW) of 188.15 masl, and one for the case of rapid drawdown (Figure 57) and one for the exceptional load case with dimensioned leakage. The calculated safety factors are summarized in Table 8. The results are the same as in the PEA because no material property has been changed.

Table 8: Results of Stability Analyses

Load Case	Description	FoS	Figure
1	Normal operating conditions with HHW 188,15 masl, stationary flow	1.7	
2	Design leakage 198 masl	1.5	
3	Rapid drawdown to empty pond	1.3	Figure 57

The assessment shows that the stability of the embankment is satisfactory in all load cases.

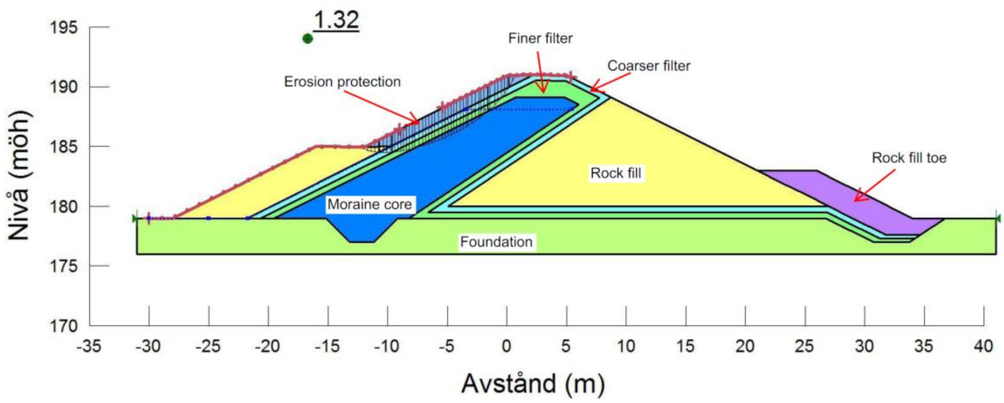


Figure 57 Stability analysis with rapid drawdown to empty pond.

9.6.2.2 *TMF*

A slope stability model was developed for the typical section of the TMF. In the PEA six analyses were carried out:

- one with stationary conditions with dam crest at +186, +190, +195 and +210 masl and tailing just at these levels
- one for the case of long-term perspective with the taling at +210 masl and un-drained conditions and
- one for the exceptional load case with liquefaction.

The calculated safety factors are summarized in Table 9. The results are the same as in the PEA, except the load case with liquefaction (Figure 58) which is slightly lower but still above the requirement of 1,1.

Table 9: Results of Stability Analyses

Load Case	Description	FoS	Requirements in GruvRIDAS and RIDAS	Figure
1	Normal operating conditions with tailings at 186, masl, drained conditions and normal operation	1.7-2.2	1.5	
2	Normal operating conditions with tailings at 190, masl, drained conditions and normal operation	1.6-2.0	1.5	
3	Normal operating conditions with tailings at 195, masl, drained conditions and normal operation	1.6-1.9	1.5	
4	Normal operating conditions with tailings at 210, masl, drained conditions and normal operation	1.6-2.5	1.5	
5	Longtime perspective with tailings at 210, masl, undrained conditions	1.3	1.3	
6	Normal operating conditions with tailings at 210, masl, liquefaction	1.15	1.1	Figure 58

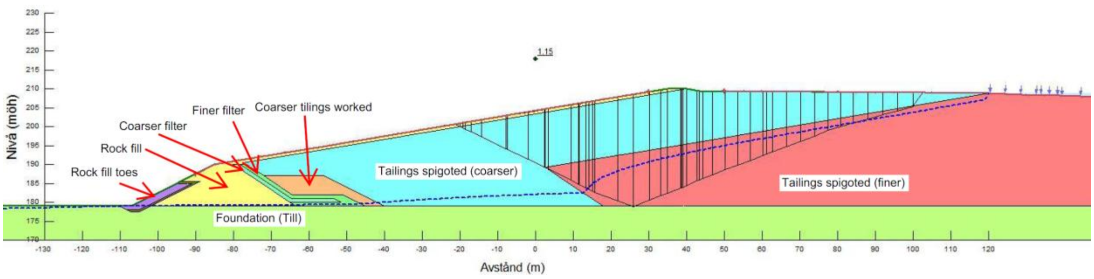


Figure 58. Stability analysis with rapid drawdown to empty pond.

9.6.2.3 *Joint dam in-between clarification pond and TMF*

A slope stability model was developed for the typical section of the embankment. In the PEA four analyses were carried out:

- one with stationary conditions with the maximum operating water level (HHW) of 188.15 masl and a full TMF of tailings level 210 masl,
- one for the case of empty TMF and a CP in maximum operating water level (HHW) of 188.15 masl,
- rapid drawdown in the CP and a full TMF of tailings level 210 masl and
- rapid drawdown in the CP and a full TMF of tailings level 210 masl with liquefaction.

The calculated safety factors are summarized in Table 10. The results are the same as in the PEA, except the load case with liquefaction which is slightly lower but still above the requirement of 1,1.

Table 11: Results of Stability Analyses

Load Case	Description	FoS	Requirements in GruvRIDAS and RIDAS	Result
1	The maximum operating water level (HHW) of 188.15 masl in CP and a full TMF of tailings level 210 masl through embankment	2.1-2.4	1.5	Figure 15
2	A empty TMF and a CP in maximum operating water level	1.7	1.3	Figure 16
3	Rapid drawdown in the CP and a full TMF of tailings level 210 masl with liquefaction	1.6-1.9	1.1	

The analyses show that the stability of the embankment is satisfactory in all load cases.

9.6.2.4 *Seismic assessment*

A seismic assessment of the dam stability was not carried at this stage for any of the dams, as an earthquake load is usually not regarded at the design stage of an embankment dam in Sweden. Earthquake loads are only relevant in terms of long term stability of a TSF upon closure (GruvRIDAS, 2010).

9.7 **Seepage Analysis**

Estimates of seepage through the embankment were analyzed using the SEEP/W (Version 7) software package by GeoStudio. The scenarios introduced in Section 6.4 include a high water level based on maximum design elevation, and a low water level based on a drawdown scenario.

9.7.1 **Material Properties**

The properties detailed above are partly based on results from laboratory campaign and literature, Table 12 summarizes material assumptions.

Table 12: Hydraulic Conductivity Input Parameters

Unit	Saturated Hydraulic Conductivity K	Source and comments
Tailings spigoterad (coarser)	Kx=1x10-5 Ky=1x10-6	Different hydraulic conductivity's in x-way and y-way. Laboratory campaign and experience value from
Tailings spigoted (finer)	Kx=5x10-7 Ky=5x10-8	Different hydraulic conductivity's in x-way and y-way. Laboratory campaign and experience value from
Coarser tailings worked	K=1x10-5	Stirred recalimed tailings, experience value from GruvRIDAS
Foundation (till)	K=1x10-7	Experience value
Finer filter	K=1x10-4	Experience value
Coarser filter	K=1x10-2	Experience value
Till (coarse)	K=1x10-7	Experience value
Rock fill	K=1x10-3	Experience value for Kristallint aggregates
Transition layer	K=1x10-2	Experience value
Toe support rock	K=1x10-1	Experience value
Erosion protection	K=1x10-1	Experience value

9.7.2 **Analysis and Results**

9.7.2.1 Clarification pond

The results of the SEEP/W model based on properties specified in Table 12 are presented in Table 13. Three estimates were generated based on head elevations in the retention pond: a static upper level (188.15 masl), an exceptional case with a damage moraine core there only the rock fill will limits the leakage (design leakage) and a post-drawdown level (180 masl).

Table 13: Seepage results

Seepage scenario	Seepage from face (m3/sec)	
Stationary level	9,5E-06	
Exceptional case with damage moraine core (design leakage)	1,3E-03	
Post drawdown level	Assumed pore pressure line	

The results of the analyses do not include a time-stepped (transient) analysis of a rapid drawdown event, but high and low water steady state calculations. These seepage rates are within expected values for a rock and moraine core embankment.

9.7.2.2 *TMF*

The results of the SEEP/W model based on properties specified in Table 12 are presented in Table 14. Six estimates were generated based on head elevations in the TMF: in different height of the TMF, a static upper level (184,78 masl, 188.78 masl, 193,82 masl and 208,78 msal) for one of each, an exceptional case with a bottom filter is clogged and one where the finer tailings will be liquefaction.

Table 14: Seepage results

Seepage scenario	Seepage from face (m3/sec)	
Stationary level 184,78 masl	3,9E-06	
Stationary level 188,78 masl	1,0E-05	
Stationary level 193,82 masl	3,1E-6	
Stationary level 208,78 masl	3,0E-6	
Exceptional case with a bottom filter is clogged	3,0E-6	

The results of the analyses do not include a time-stepped (transient) analysis of a rapid drawdown event, but high and low water steady state calculations. These seepage rates are within expected values for a rock and moraine core embankment.

9.7.2.3 *Middle dam*

The results of the SEEP/W model based on properties specified in Table 12 are presented in Table 15. Three estimates were generated based on head elevations in the retention pond and the TMF: a static upper level (CP 188.15 masl TMF 209,5 masl), a exceptional case with the bottom filter is clogged in the CP and a post-drawdown level (179 masl) in the CP and a full TMF.

Table 15: Seepage results

Seepage scenario	Seepage from face (m3/sec)	Figure
Stationary level	3,5E-05	Figure 17?
Exceptional case with damage moraine core (design leakage)	3,6E-05	Figure 18?
Post drawdown level in CP and a full TMF	7,7E-06	?

The results of the analyses do not include a time-stepped (transient) analysis of a rapid drawdown event, but high and low water steady state calculations. These seepage rates are within expected values for a rock and moraine core embankment.

10. Water management

10.1 Construction - Drainage and dewatering activities

Where the dam embankments are located in the wet areas (peat area and small lake) drainage and dewatering activities are needed during construction both to be able to excavate the loose sediments and especially to place and compact the different materials. Placement is especially critical for the moraine and fine filter.

The dewatering will likely consist of a combination of measures. Temporary coffer dams/dykes are constructed on either side of the planned dam foot prints. The dykes can likely be placed directly on the peat. Inside/between the dykes pumping wells are placed to divert water and lower the ground water surface. Excavation of the loose sediments are preferably done wintertime.

Placement and compaction of moraine and fine filters are only allowed to be constructed during frost free conditions (late spring to early autumn).

One alternative that need to be investigated further if more cost and time effective is, if the supporting rock structure of the dams can be used as a temporary coffer dams/dykes on one side. This might be possible if the loose sediments can be excavated wintertime without dykes on both side and the back-filled moraine replaced with blasted rock under the rock fill.

It can also be investigated if the horizontal filter blanket under the clarification pond can be replaced with a geotextile which might be possible to place wintertime as compared to the fine filter.

10.2 Production

10.2.1 General management

The clarification ponds are both used to store the water needed for production especially during winter and to attenuation higher flows during spring flood and to treat and remove sediments from the water diverted to the ponds especially from the TMFs but also from other areas.

The capacity to remove the sediments is determined by the surface area of the pond. Fine particles of silt can take several hours to a day to settle to the bottom. The surface areas of the clarification ponds are 40 000 m^2 for the northern pond and 300 000 m^2 for the northern pond.

The northern pond has a lower capacity to treat water with fine sediment which means that preventing sediments from the TMF to flow into the pond with the excess drainage/decant water will be important to reduce the load. Naturally this will also be important for the southern clarification pond to reduce risks. This can be done by keeping the decant pool further away from the spillway and by using the coarser as a filter and canal with low slope to give as low flow velocities as possible (see concept plan view in Figure 59) . The coarse tailing can also be used as a berm to control the level in the decant pond.

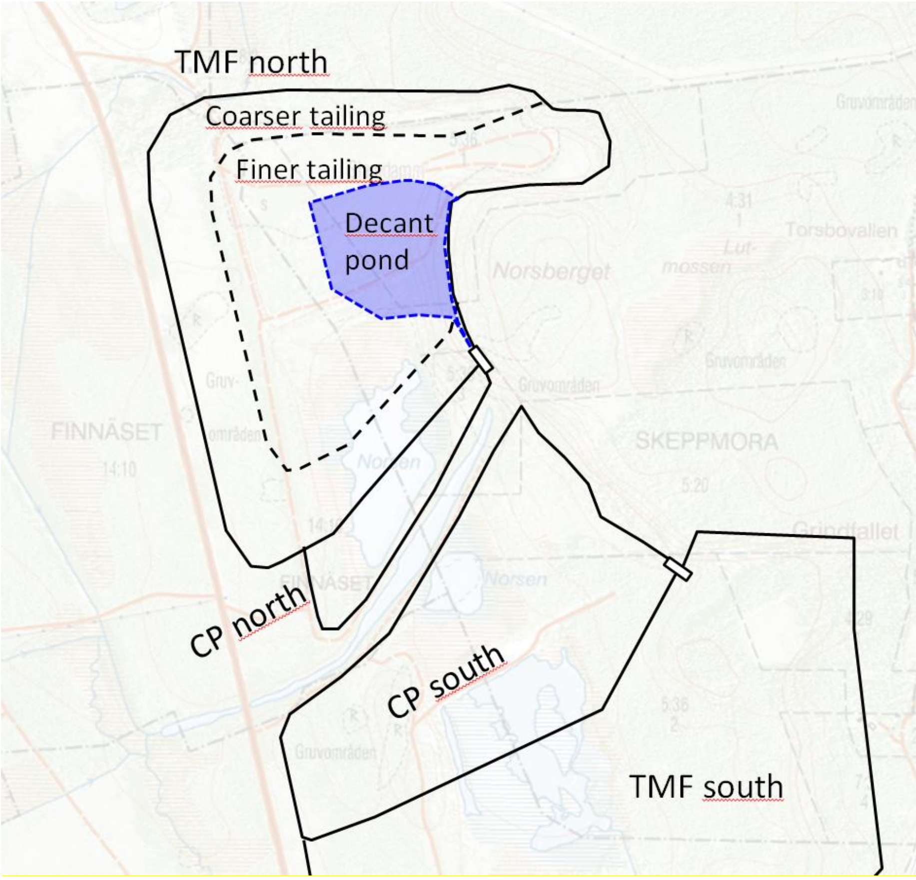


Figure 59. Plan view of northern TMF conceptually illustrating decant pool and run-off to spillway (at late stage of deposition +200 masl).

At the start-up of deposition water will have to be pumped from the low locations inside the TMF areas to the spillway and clarification pond since the spillway level on the dam in-between the TMF and clarification pond cannot be lower than +187,5 masl to be able do dam the water inside the clarification pond.

The water might also have to be pumped or temporary canals constructed to divert water from local low locations towards the spillway.

10.2.2 TMF north

An issue with the northern TMF is the location of the old tailings stack on the inside which is topographically much higher than the northern and western sections of the dam layout.

The deposition strategy with segregation of the coarser material in the tailing settling next to the embankment and spigots and with finer material settling further in (at least 100 m from the dam) mean that the tailing and drainage water must flow perpendicular to embankment for some distance. As can be seen in Figure 60 the northern part of the area has two issues. First water will accumulate and not flow to the clarification pond unless the small ridge in the south western corner is removed and a canal constructed to the pond. Even so the distance from the embankment to the tailings stack is just 30-50 m which means that a mixture of coarse and fine tailing will on the ground where it only should be coarse tailing. Thus making it impossible to raise the dam upstream from a safety and stability point of view. This problem is over come if the dam section is moved to the top of the tailings stack.

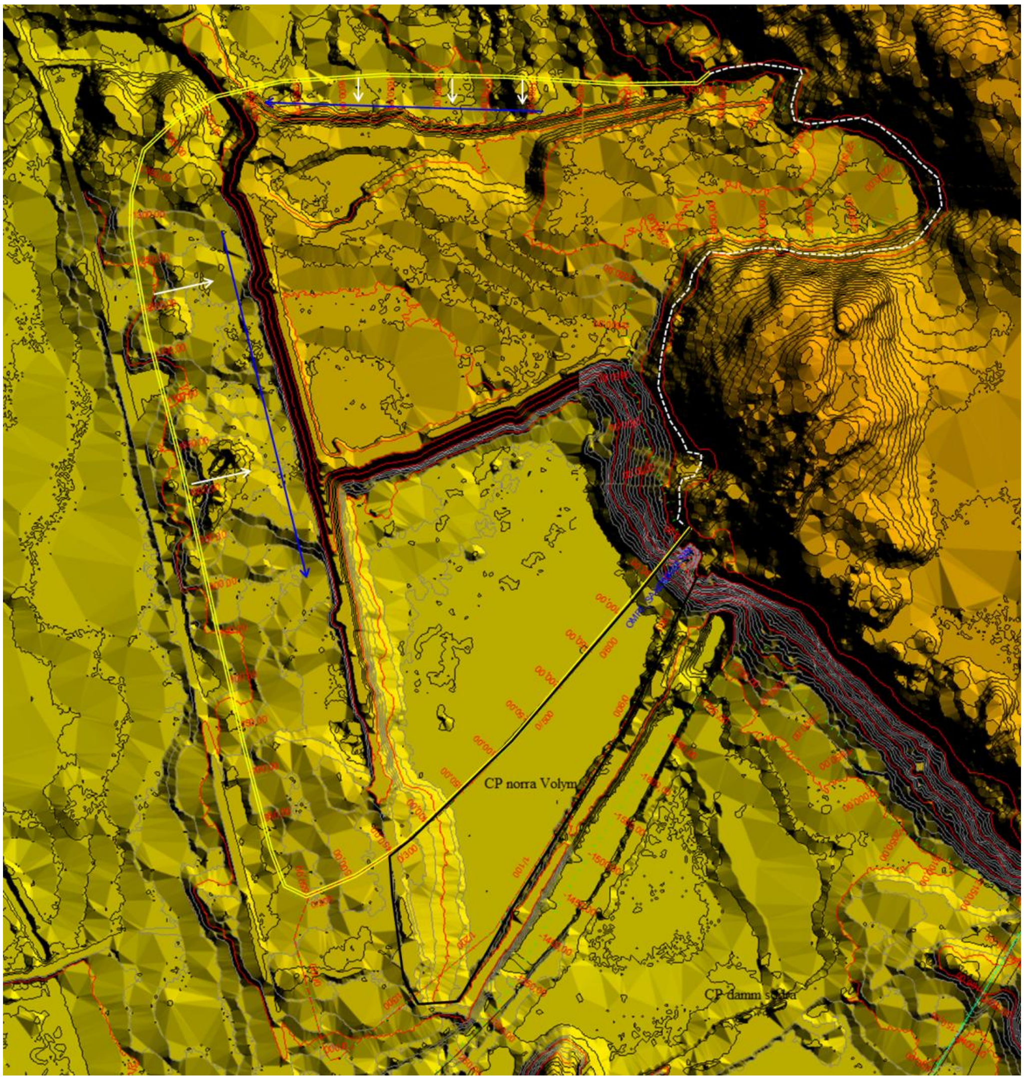


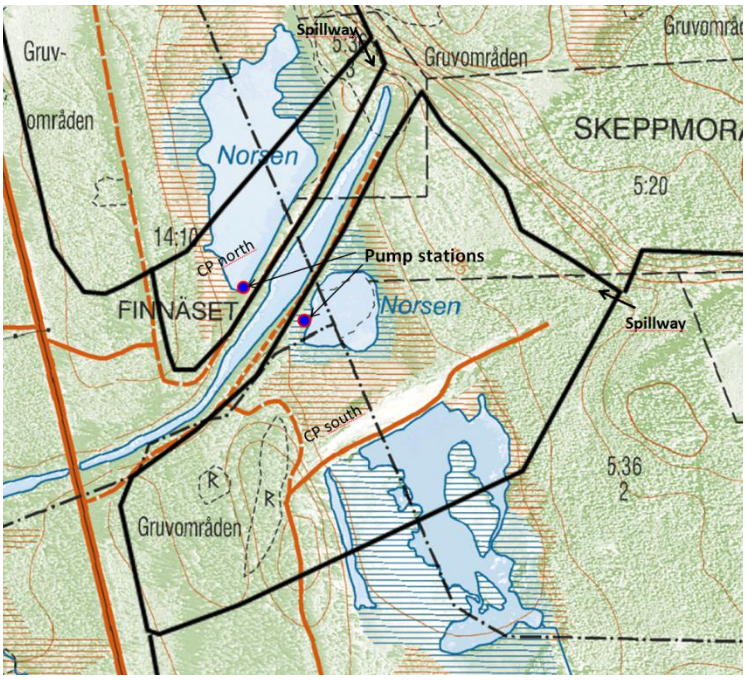
Figure 60. Plan view of northern TMF illustrating how the spigotted tailing with white arrows and fine tailing and drainage water in blue. In the northern section fine tailing and water will accumulate to close to the dam.

Further the distance from the western embankment to the tailing stack is relatively short (100-120 m) which mean that special measured likely will have to be taken to ensure that the finer tailing don't accumulate under the foot print where coarse tailing should and the embankment later will be raised. Drainage ditches very likely have to be constructed at start up to divert the water and finer tailing away from the embankment towards and along the old tailing stack and on to the clarification pond.

10.3 Pump stations

The clarification pond will incorporate pump stations to pump water to the process plant and excess water to the canal. The pump stations will be located in the south western part of the natural depression forming the wet land and small lakes, with the lowest elevation in the area. With the level of the intake pipe to the pump stations situated around half a meter above the existing ground level and the minimum level in the ponds at around a meter above the pipe crest, the practical usable volume in the ponds can be maximised. Reason for the location of the pump station also is to maximise the distance to the spillways from the TMFs to give an as large surface area as possible in-between to improve sedimentation of fine particles in the water.

A map showing the conceptual locations of the planned pumping stations are presented in Figure 61.



Figur 61. Conceptual location of pump stations in the clarification ponds.

11. Consequence Classification

11.1 Introduction

RIDAS and GruvRidas (from here on Ridas) states that dam safety is governed by consequence, i.e. dams should be classified according to the consequences that result from a potential dam failure.

The consequences of a dam failure are evaluated with regard to the probability of:

- Loss of human life or serious injury
- Damage to the environment, public facilities and other economic values

The RIDAS consequence classification system is presented in Table 16. The classification system consists of four different consequence categories, taking the probability of loss of life and damages into account. The classification is divided into four categories, i.e. 1A, 1B, 2, and 3, where 1A corresponds to the most serious category with the highest probability of loss of human life and/or severe damages.

Table 16: Consequence Classification System according to RIDAS (2012)

Consequence classification	Consequence of dam failure, expressed as probability of damage/injury
1A	High probability of loss of many human lives, or High probability of very severe damage to - important local facilities - significant environmental aspects, or enormous financial damage
1B	The probability of serious injury or loss of human life is not insignificant, or Considerable probability of severe damage to - important local facilities - significant environmental aspects or High probability of - enormous financial damage
2	Non-negligible probability of considerable damage to - local facilities - environmental aspects or - financial damage
3	Negligible probability of damage as stated above

Classification is made separately for each of the TMFs since no domino scenario can be expected.

11.4.1.1 *Northern TMF*

A dam breach in the northern section would result in a flow towards Främundstjärnen and on to Gonäsån. No houses are impacted downstream.

A dam breach in the western section would result in a flow over road 611 just west of the TMF, to the lower topographical areas in the west and north towards Gonäsån. Tailing water can also flow into the canal. No houses are impacted downstream.

The northern TMF has been asses as consequence class 2 with low probability of loss of lives. The traffic intensity of the nearby roads is low which mean non-negligible damage. The impact on environment, infrastructure and economical has been assessed to cause non-negligible damage.

11.4.1.2 *Southern TMF*

The southern TMF is located relatively close to the settlements Karlshed (700 m) and smaller industrial area and settlement in Klanshyttan (100 m).

A dam breach in the western section would result in a flow over road 611 just west of the TMF, to the lower topographical areas in the west and north towards Gonäsån. Tailing water can also flow into the canal. The houses mentioned above can be impacted.

The southern TMF has been asses as consequence class 1B due the not insignificant risk of loss of lives. The traffic intensity of the nearby roads is low and thus small impact. The impact on environment, infrastructure and economical damage are small or insignificant. The traffic intensity of the nearby roads is low which mean non-negligible damage. The impact on environment, infrastructure and economical has been assessed to cause non-negligible damage.

Collateral consequences of a dam breach from the TMF into the clarification pond have also been assessed. The consequences and classification is assessed not to increase the consequences thus the classification remain the same.

11.4.2 **Northern clarification pond**

No specific assessment has been for the pond but the flooding and consequences are expected to be similar to that of a breach in the western part of the northern TMF.

The northern clarification pond has been asses as consequence class 2 or 1B due the not insignificant risk of loss of lives.

11.4.3 Southern clarification pond

No specific assessment has been for the pond but the flooding and consequences are expected to be similar to that of a breach in the western part of the southern TMF.

The southern clarification pond has been asses as consequence class 2 or 1B due the not insignificant risk of loss of lives.

11.4.4 Consequences long-term

After the operation end the clarification ponds will be emptied of water and the dam embankment cut-off thus there will be no long term risks.

In the TMFs the phreatic surface will gradually decrease since the embankments are designed to drain the water. When the water level has decreased to a lower level the stability of the tailing and dams will improve thus also the risk of dam breach and potential damage. The consequence classification can likely be lowered to 3.

12. Monitoring and Instrumentation

12.1 Monitoring Program

The purpose of a monitoring program is to obtain information during the life of a dam and to verify that the embankment is behaving as per the design. The monitoring program provides a basis for assessment of changes in performance that could potentially compromise the safety of the construction. If variations affecting the safety of the construction should occur, the monitoring data assists and forms a basis for suitable remedial measures.

The main objectives of the proposed monitoring program are:

- To verify assumed design conditions of the construction
- To verify the status and function of the facility
- To observe changes that may affect the safety of the facility.

Identified variables that are considered to influence the stability if the construction and affect the risk for dam failure include:

- Freeboard of the embankment
- Settlements and displacements of the embankment
- Pore pressures in the embankment
- Seepage through the embankment

12.2 Instrumentation

The requirements for instrumentation according to RIDAS are dependent on the type of dam, construction materials and foundation conditions, as well as the consequence classification. Table 17 presents the basic requirements for an embankment dam with impervious core. Piezometric levels in the dam toe need to be monitored primarily when the dam is founded on soil.

Table 17: Basic Instrumentation and Monitoring Frequency as Recommended by RIDAS

Parameter to be Monitored	Monitoring Frequency
Seepage	Weekly to Monthly
Water level in the embankment	Weekly to Monthly
Water level in the dam toe	Weekly to Monthly
Crest settlements and displacements	Monthly

12.3 Visual Inspections

Visual inspections aim to identify early indications of problems such as settlements, cracking, displacements, sink holes, seepage and erosion. The documentation should include detailed information on the observed defects, e.g.

position, extension and width of cracks, size and depth of sinkholes, and seepage volumes per time unit. The documentation should also contain photographs and sketches to allow for comparison during future inspections.

To facilitate the observation of potential settlements and sinkholes, the crest should be even and the dam toe clear from any kind of vegetation.

12.4 Monitoring of Water Tables

Monitoring of the water table in the embankment should be carried out continuously by water-level gauges in standpipes. The monitoring is performed for safety reasons to avoid overtopping of the embankment. In addition, the information can be used to correlate the water table in the pond with pore pressure levels in the embankment.

12.5 Seepage Measurements

Continuous monitoring of the leakage through the dam will be carried out. The measurements are done by collecting the leakage in trenches along the downstream side of the dam, leading leakage water to Thomson weirs where the flow is measured. The monitoring of the leakage through the dam is a practicable method to observe changes in the flow through the embankments that involve a large volume, while pore pressure measurements are based on spot observations. Additional temperature and conductivity measurements will also be needed in sections where peat occurs. In these areas, the ground water table is approximately at ground surface; therefore, collecting trenches cannot be used for seepage measurements.

12.6 Monitoring of Deformations

Fixed stations, survey beacons and inclinometers can be installed along the crest of the embankment to enable measurement of any vertical and horizontal movements. Ideally, the monitoring of deformations should be performed in the same sections as the pore pressure monitoring. Fixed points for the surveying will be installed outside the area affected by embankment movement. Fixed points as well as measuring points have to be protected from damage and frost action by locating them in insulated and covered well holes. A monthly monitoring frequency is recommended.

13. Cost estimate / Bill of Quantities

13.1 Introduction

In this section the cost estimate for design, construction and operation of the dams are presented for the northern and southern facilities respectively. The estimated costs have been divided in initial investment expenses (Capex), sustained Investment expenses (Sust Capex) and operation expenses (Opex) for each facility.

An important approach part of the planning and start-up of the mining operation by NIO is to reduce the capital investment expenses (Capex) the firsts few years during construction and production and at the same time clarify which costs can be delayed until the process has been started.

Both the clarification ponds and TMFs are large infrastructures and thus relatively costly to construct.

There are two alternatives for start-up of operation of the TMFs, either starting with the northern and when it is full continuing with the southern or vice versa. Since the length of both the dam for the northern clarification pond and TMF (both having smaller storage volumes) are shorter than the southern alternative starting with northern dams mean the lowest Capex. Another advantage is potentially shorter construction time and thus less risk of not have the facilities ready in time when production start.

To even further reduce the Capex the northern TMF will be constructed in two stages. First a lower dam is constructed (Capex costs) where needed to be able to start with the production and deposition, and after this the dam is raised and constructed where the natural elevation is higher, starting sometime during year one, to full initial height from which it can be raised upstream using the spigotted tailing. The cost of the second stage is considered as sustained investment expenses (Sust. Capex).

Since the Southern facilities will be constructed five year later than the Northern facilities, estimated costs is considered as sustained investment expenses (Sust. Capex).

A sensitivity analysis have also been done with estimation of difference in costs when using materials (blasted rock) from existing local quarries nearby as compared to using material (blasted rock) from the industrial area (base case). Material sourced from an external quarry obviously has to be transported a much longer way.

13.2 Assumptions

To estimate the costs several assumptions have been made. These are presented in the sections below along with different sources used for unit prices etc.

13.2.1 Capital investment expenses (Capex – Sust Capex)

- The size of different areas and distances are measured from existing CAD-drawings

- The unit price for cutting and removing trees are received from PEAB Dams Northland Kaunisvaara.
  Approximate 50% of the different dam areas are covered with trees

- The unit price for removing vegetation layer contains transport within a radius of 1 km, according to PEAB Dams Northland Kaunisvaara.
  The estimated area is based on the dam length (3700m and 5000m). The width of the area is 100m and the thickness of the vegetation layer is estimated to 0,2m.

- The unit price for removing vegetation layer contains transport within a radius of 1 km, according to PEAB Dams Northland Kaunisvaara.
  The estimated area is based on the length of drainage and interception ditch (2500m and 3500m). The width of the area is 15m and the thickness of the vegetation layer is estimated to 0,2m.

- The unit price for excavation soil for drainage and interception ditch are received from Maserfrakt 2015-03-12.
  For each meter, 8 m^3 soils have to be excavated.

- The unit price applying erosion protection layer for drainage and interception ditch are received from Maserfrakt 2015-03-12.
  For each meter, 3 m^3 materials have to be applied, if the thickness of the layer is 0,4m.

- Both facilities and both dams are planned to be founded on soft soil. Therefore the soil has to be excavated. The unit price for excavate the soil is received from Maserfrakt 2015-03-12.
  The excavation is estimated to 300 m for all four dams. For each meter, 406 m^3 for one of the dam and 554 m^3 for the other three dams have to be excavated.

- To make the excavation of soft soil possible, temporary coffer dams first has to be applied, one on each side of the excavation area, mentioned above.
  The unit price for applying the coffer dams is received from Maserfrakt 2015-03-12.

The excavation is estimated to 300 m for all four dams. For each meter, 406 m^3 for one of the dam and 554 m^3 for the other three dams have to be excavated.

- Drainage activities have to be made to manage the excavation of soft soil. The costs are estimated from experiences from PEAB Dams Northland Kaunisvaara.

- Where the dams are founded on better soil, a cut-off trench will be excavated. The unit price for excavate the soil is received from Maserfrakt 2015-03-12. The excavation is estimated to 3100 m for both dams respectively. For each meter, 8 m^3 soils have to be excavated.

- To produce different filter material, the source will extracted from bedrock. The unit price for extracting the rock is received from Maserfrakt 2015-03-12.

- To extract the rock, natural soil first has to be removed. The unit price for extracting rock is received from Maserfrakt 2015-03-12. As an assumption an area (100*100 m^2) and the thickness of the soil of 3m.

- The volume of different filter layer are based on evaluation from models in CAD-drawings

- Some measurement facilities will be installed in all dams, such as piezometers, monitoring wells etc. The costs are estimated out of experiences from PEAB Dams Northland Kaunisvaara

- The cost to build a spillway from tailings pond to clarification pond, is based on experiences from PEAB Dams Northland Kaunisvaara.

- To make it possible to build the spillway mentioned above, trees have to be cut end removed. Vegetation and soil layer have to be excavated and finally some rock has to be extracted.
  The unit price for excavating soil and extracting the rock, is received from Maserfrakt 2015-03-12

- A temporary spillway from tailings dam to clarification pond, will be built in an initial phase of the disposal.
  The cost to build this temporary spillway is based on experiences from PEAB Dams Northland Kaunisvaara.

- A spillway from the clarification pond to the canal.
  The costs are estimated out of experiences from PEAB Dams Northland Kaunisvaara

- Costs for detailed geotechnical site investigations are estimated.

- Cost for detailed design and construction drawings are estimated.

13.2.2 **Operation costs (Opex)**

- The disposal strategy is based on a method where the spiggots are moved around to vary the place where tailings are disposed. Therefore the pipes and spiggots will be moved. Estimated time for this is one day work each week for two persons. They also need a tractor for this job.
- This job is assumed to be done by internal workers. The total cost for their salary is estimated to 500 SEK/h.
- The tailings dam will be raised during the operation time. The costs for raising the tailings dam, includes exaction of tailings and applying erosion protection layers on the dam downside slop.
- Since the level of the disposed tailings is raised, the spillway from tailings dam to clarification pond also has to be raised three times. These costs are divided into yearly costs.
- To maintain the dam safety, supervision of the dam will follow recommendations given from GruvRIDAS. These costs are divided in internal and external costs. As internal costs supervision on weekly basis and monitoring instrumentations is estimated to2 days/week for 2 persons. Operating tests and Inspections is estimated to 2,5 days/year for 2 persons. External costs are detailed inspections which will be held once every 6 year and detailed dam safety investigations which will be held once every 18 year. This is based on the assumption that the dams will have "kkl2"
- An important step to reach dam safety status is to collect all data on the dams. Therefore a DTU-manual is assumed to be established. The costs are divided into yearly basis. In addition to the establishments, a yearly update is included.
- A dam safety classification will be established. This classification is based on the consequences if a dam brake occurs. The classification includes a hydraulic model calculation and an evaluation of consequences. The cost of this classifications is divided 50-50 into both facilities.
- The costs for dust mitigation are based on experiences from similar mines. The cost includes dust control on roads, open areas and around crushers/sieves.
- Other various operation costs are also added estimated.

13.3 Result and conclusions

Based on assumption above following Capex and Sust Capex are calculated, see table 18.

Table 18 Compiled capital costs on the dams

Activities	**Capex Northern [kSEK]**	**Sust Capex Northern [kSEK]**	**Sust Capex Southern [kSEK]**
Detailed Geotechnical Investigation	1 000	-	1 500
Detailed Design and Construction	1 000	-	1 500
Total cost	55 454	29 937	119 455
Cut and clean trees	1 077	918	4 122
Excavation drainage ditch	361	570	1 757
Excavation for Dam Foundation	4 197	-	7 009
Construction activities in wet areas	2 680	-	2 680
Extraction of rocks	9 644	7 044	23 801
Processing/Crushing incl trsp < 3km	12 650	10 877	32 870
Placement and Compaction	18 247	9 633	41 221
Instrumentation	600	600	1 200
Detailed Geotechnical Investigation	1 000	-	1 500
Detailed Design and Construction	1 000	-	1 500
Spillway	6 000	295	4 795

Based on assumptions on operating costs each year are calculated, see table 19.

Table 19 Compiled operations costs on the dams

Activities	**Opex Northern [kSEK/year]**	**Opex Sothern [kSEK/year]**
Total cost	12 048	10 828
Tailings disposal	624	624
Raising the dam	1 900	1 444
Processing/Crushing incl trsp < 3km	4 467	3 541
Placement and Compaction	2 893	2 210
Spillway	172	172
Supervision-Internal	882	1 734
Supervision-External	111	103

13.4 Discussions and conclusions

The estimated costs are based on different assumptions, as shown above.

The majority of the Capex for the northern dams consist of cost of constructing the dam for the clarification pond including the dam in between the TMF. The scenario is basically the same for the southern dams>

By assessing costs more carefully, it will be possible to increase the accuracy of the costs of some activities. For instance by looking when these activities will be done, it will be possible to consider different advantages with seasonal conditions. For example excavations in the wet areas are preferably done wintertime.

On the other hand costs related to drainage and dewatering activities of the wet areas during construction of the dams and construction of temporary coffer dams to be able to excavate the loose sediment, and place moraine and dam material under dry conditions, can be more expensive than estimated. Here more planning and investigations is needed

Another example is filter production may decrease. However filter cost can also increase if difficult to produce using the rock material from site.

So far all assumptions on Capital costs, especially for the Northern dams when the production haven't started, are made assuming it is possible to excavate and extract all materials (moraine and rock) from either within the inundation areas of the dams or the industrial area. When production has started waste rock might also be used to produce the rock material needed for the dam constructions, however this has not been assumed. If waste rock can be used the annual Opex will be reduced with around 2 Msek. If not possible the materials needs to be sourced from a local quarry. According to information from Maserfrakt it be possible, however not suitable, to get the material from local quarry in Håksberg. It would add an extra transportation cost of about 70kr/m^3 which would mean an increase of the Capex for the northern dams with around 12 MSek.

Another aspect that needs to be studied further is the dam layout for the northern TMF. As discussed the current dam layout in the northern part is located behind the old tailings stack. Besides from operational issues with the tailings and water management, if the dam layout is moved up on the old tailings stack the sustained Capex for stage 2 and especially Opex will be reduced (around 2 Msek a year during 2-3 years) until the +200 level is reached.

Figure 62 and 63 and presents the different construction stages for the TMFs to give an indication on the amount of dam needed and thus the distribution of cost at each stage.

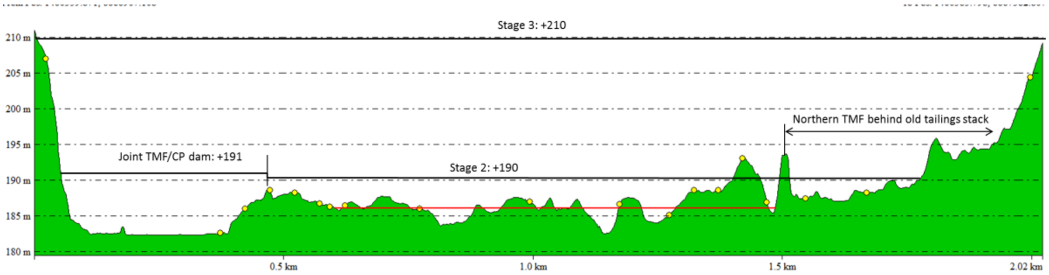
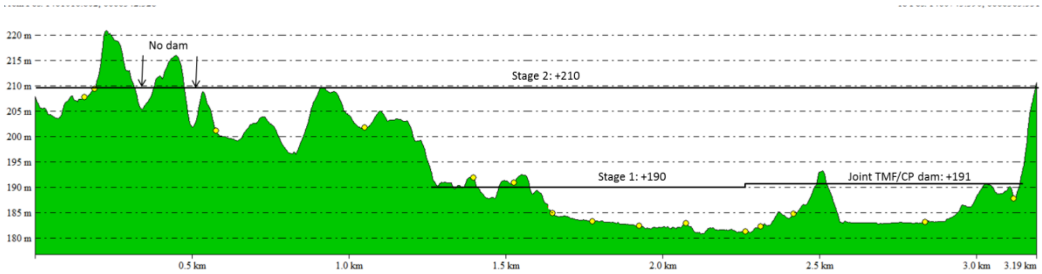


Figure 62. Length profile of the northern TMF current dam (north of old tailings stack)

layout with profile starting from the south eastern corner. Starter dam stage 1 constructed to +186, stage 2 to +190 and stage 3 gradually to +210. Starter dam only needed for short distance as can be seen, but also the dam in-between TMF and clarification pond.



Figur 63. Length profile of the southern TMF with dam profile starting from the south eastern corner. Starter dam stage 1 constructed to +190 and stage 2 gradually to +210. Starter dam only needed for about half the dam length including dam in-between TMF and clarification pond.

14. CONCLUSIONS AND RECOMMENDATIONS

Conclusions and recommendations related to Capex and Opex costs are presented in section 13.4

At the next stage as a basis for detailed design but also construction strategies and more accurate costing estimate, detailed geotechnical site investigations are needed. These will focus mainly on the wetland area but also on sections now categorized as solid grounds (moraine) and the previously deposited tailing and e.g. include ground penetrating radar, sounding (e.g. CPT and other), drilling and borrow pits. Laboratory test work will include e.g. PSDs, permeability tests, direct shear test, and proctor test.

An issue with the layout in the northern section of the TMF is the location of the dam behind the old tailings stack. This means that you basically construct a dam behind an already existing "natural" confinement. One thing that has to be investigated further is if suitable to construct a starter dam on the old tailing, e.g. considering the PSD, fine layers close to surface etc. An additional issue with the current dam layout is management of the tailing and water where the old tailings stack is too close to embankment to get sufficient segregation of the coarse and fine tailings. If raised the embankment would be founded partly on fine sediments which can't be allowed.

Tailings management poses many challenges during operation and needs to be studied and planned more in detail in future. Since spigotting will take place from opposite sides of the perimeter dams at both TMFs is means that potentially fine tailing from one side can flow over and settle close to the other embankment. However fine tailings cannot be allowed as a layer under the coarse tailing this close to the embankment as it can potentially can act as a slip surface and increase the risk of dam failure. Also the decant water (containing fine sediment and increasing the phreatic level) needs to be controlled and prevented from gathering to close to the other embankment. Both these issues are managed by carefully controlling the height of the lift on either side, i.e. moving the spigots well in time before the height one embankment get so high. Berms inside the TMF can also be constructed to stop the flow of fine tailings and water.

Management of the water during the different stages have many challenges both to divert the water from the TMFs to the clarifications ponds considering topographical differences inside the TMF and with a threshold of the spillway located a few meter above the ground at start-up of each of the TMFs (meaning pumping needed to the ponds) and to prevent too much sediments from entering the ponds. In future studies the management of water need to be studied more in detail during the different stages of operation including how the impact on dam minor changes dam locations.

An aspect that needs to be further studies is if the dam section in-between the TMF and clarification pond can be constructed as a non impervious embankment (i.e. a rock fill embankment with filter layers) since water at least initially will be stored on either side of the dam. Also when the dams are raised this concepts might be possible to use since water need to seep to the clarification pond anyway. This concept would make the excavation and of filling in the wetland much easier.

One alternative that need to be investigated further, if more cost and time effective, is whether the supporting rock structure of the dams can be used as a temporary coffer dams/dykes on one side. This might be possible if the loose sediments can be excavated wintertime without dykes on both side and the back-filled moraine replaced with blasted rock under the rock fill.

In future studies the consequence classification needs to be studied more in detail with dam breach simulations where needed.

Appendices

1) Detailed cost estimate Capex and Opex with annual costs

Appendix I

COST ESTIMATE DETAIL

Project		**Blötberget Iron Ore Project**	
		SEK/US$	**8.6232**
		SEK/EURO €	**9.2869**
		SEK/GB £	**12.7441**
		US$/Euro €	**1.0770**

			Pre-production			**Sustaining**																
		Total	**Year -2**	**Year -1**	**Total**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**	**Year 16**	**Total**
		SEK	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**	**SEK**
Underground Mine																						
Combined access/conveyor surface to -304		**110,000,000**	88,000,000	22,000,000	110,000,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Dewatering UG workings(estimate is between 5 & 9SEK/m³ so assume 7.5)		**37,500,000**	30,000,000	7,500,000	37,500,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Rehabilitation of 320, 370 and ramp240-320 incl. rock support		**5,250,000**	-	5,250,000	5,250,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Conveyor tunnel -304 to -420		**22,500,000**	-	22,500,000	22,500,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Decline -320 to -420		**18,750,000**	-	18,750,000	18,750,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Conveyor tunnel -420 to -660		**55,300,000**	-	-	-	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,300,000	-	-	-	-	-	**55,300,000**
Decline -420 to -660		**60,000,000**	-	-	-	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	-	-	-	-	-	-	**60,000,000**
Sandell access, ramp		**20,725,000**	10,362,500	10,362,500	20,725,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Ventilation development - rock excavations		**2,500,000**	-	2,500,000	2,500,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Ventilation raises (diam 4m)		**17,850,000**	-	-	-	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	-	-	-	-	**17,850,000**
Ore passes - vertical (diam 3m)		**31,650,000**	-	-	-	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	-	-	-	-	**31,650,000**
Workshop - rock excavation		**7,995,000**	-	7,995,000	7,995,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Workshop - installations		**5,000,000**	-	5,000,000	5,000,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Crusher - rock excavation per cavern		**14,400,000**	-	3,600,000	3,600,000	-	-	3,600,000	-	-	3,600,000	-	-	3,600,000	-	-	-	-	-	-	-	**10,800,000**
Crusher - haulage drives		**10,000,000**	-	2,500,000	2,500,000	-	-	2,500,000	-	-	2,500,000	-	-	2,500,000	-	-	-	-	-	-	-	**7,500,000**
Auxiliary rock excavations (fuel station, explosives storage etc.)		**6,000,000**	-	6,000,000	6,000,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Conveyor installations at to -420 level (15500SEK/m)		**52,605,000**	-	52,605,000	52,605,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Conveyor installations at to -500, -580 &-660 level (15500SEK/m)		**39,480,000**	-	-	-	-	-	12,600,000	-	-	12,600,000	-	-	14,280,000	-	-	-	-	-	-	-	**39,480,000**
Conveyor sustaining Capex (Replacement drift CV)		**9,847,400**	-	-	-	-	-	-	-	-	9,847,400	-	-	-	-	-	-	-	-	-	-	**9,847,400**
Crusher units (includes 2 refurbishments at 50% of capital cost)		**27,000,000**	-	9,000,000	9,000,000	-	-	9,000,000	-	-	4,500,000	-	-	4,500,000	-	-	-	-	-	-	-	**18,000,000**
Crusher auxiliary (Supporting steel work at 30% cost crusher)		**10,800,000**	-	2,700,000	2,700,000	-	-	2,700,000	-	-	2,700,000	-	-	2,700,000	-	-	-	-	-	-	-	**8,100,000**
Ventilation installation		**40,000,000**	2,820,000	37,180,000	40,000,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Ventilation sustaining		**15,000,000**	-	-	-	-	-	5,000,000	-	-	5,000,000	-	-	5,000,000	-	-	-	-	-	-	-	**15,000,000**
Power distribution		**23,423,674**	-	7,807,891	7,807,891	7,807,891	7,807,891	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**15,615,783**
Power distribution sustaining		**11,360,912**	-	-	-	-	-	4,224,303	-	-	2,933,545	-	-	4,203,064	-	-	-	-	-	-	-	**11,360,912**
Permanent dewatering		**20,000,000**	-	15,000,000	15,000,000	-	-	2,500,000	-	-	2,500,000	-	-	-	-	-	-	-	-	-	-	**5,000,000**
Mobile OPEX production fleet - purchase (in OPEX)		**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Mobile OPEX production fleet - sustaining CAPEX (in OPEX)		**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
Mobile fleet - service units		**11,250,000**	2,812,500	-	2,812,500	-	-	2,812,500	-	-	2,812,500	-	-	2,812,500	-	-	-	-	-	-	-	**8,437,500**
Contingency	15%	**102,928,048**	20,099,250	35,737,559	55,836,809	3,439,934	3,439,934	9,009,270	2,268,750	2,268,750	9,617,767	2,268,750	2,268,750	8,208,085	2,268,750	1,413,750	618,750	-	-	-	-	**47,091,239**
S/T		**789,115,034**	**154,094,250**	**273,987,950**	**428,082,200**	**26,372,825**	**26,372,825**	**69,071,073**	**17,393,750**	**17,393,750**	**73,736,212**	**17,393,750**	**17,393,750**	**62,928,649**	**17,393,750**	**10,838,750**	**4,743,750**	**-**	**-**	**-**	**-**	**361,032,834**
																						-
Surface Minerals Handling																						**-**
Conveyor surface to ROM Silo		**1,550,000**	-	1,550,000	1,550,000	-																**-**
Conveyor surface to Waste Area		**13,500,000**	10,800,000	2,700,000	13,500,000	-																**-**
ROM Silo & Reclaim Feeder		**6,301,745**	-	6,301,745	6,301,745	-																**-**
Product Conveyors Silos and Loading Equipment		**6,942,000**	5,553,600	1,388,400	6,942,000	-																**-**
Rail Outloading Product Conveyor, Silos & Discharge Equipment		**57,521,533**	40,265,073	17,256,460	57,521,533	-																**-**
Contingency	15%	**12,872,292**	8,492,801	4,379,491	12,872,292	-	-	-	-	-	-	-	-	-	-	-	-					**-**
S/T		**98,687,570**	**65,111,474**	**33,576,096**	**98,687,570**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**					**-**
																						-
Rail Terminal																						**-**
Main Switch		**30,000,000**	30,000,000	-	30,000,000	-																**-**
Linjeplats		**40,000,000**	40,000,000	-	40,000,000	-																**-**
Driftsplats		**29,000,000**	-	29,000,000	29,000,000	-																**-**
Contingency	5%	**4,950,000**	3,500,000	1,450,000	4,950,000	-	-	-	-	-	-	-	-	-	-	-	-					**-**
S/T		**103,950,000**	**73,500,000**	**30,450,000**	**103,950,000**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**					**-**
																						-
Port																						**-**
Port facilities		**-**	-	-		-																**-**
Contingency	5%	**-**	-	-		-	-	-	-	-	-	-	-	-	-	-	-					**-**
S/T		**-**	**-**	**-**		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**					**-**
																						-
Industrial Site Civils																						**-**
New Roads & Strengthening Old Roads		**29,500,000**	-	29,500,000	29,500,000	-																**-**
Gravel Areas Industrial site 1&2 & earthworks		**24,680,000**	-	24,680,000	24,680,000	-																**-**
Asphalt Areas around Staff buildings		**2,000,000**	-	2,000,000	2,000,000	-																**-**
Site Clearance Cut & Fill for Industrial site 1&2		**35,478,150**	35,478,150	-	35,478,150	-																**-**
Bridge		**5,250,000**	-	5,250,000	5,250,000	-																**-**
Paintwork and Parking Area identification		**25,000**	12,500	12,500	25,000	-																**-**
Industrial Fencing		**1,150,000**	-	1,150,000	1,150,000	-																**-**
Underground Pipes (Included in Tata report)		**-**	-	-	-	-																**-**
Water Supply		**-**	-	-	-	-																**-**
Sewerage Services		**-**	-	-	-	-																**-**
Contingency	15%	**14,712,473**	5,323,598	9,388,875	14,712,473	-	-	-	-	-	-	-	-	-	-	-	-					**-**
S/T		**112,795,623**	**40,814,248**	**71,981,375**	**112,795,623**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**					**-**
																						-
Industrial Site Buildings																						**-**
Office/Locker-room		**17,800,000**	-	17,800,000	17,800,000	-																**-**
Drill core archive		**7,500,000**	-	7,500,000	7,500,000	-																**-**
Stores		**13,200,000**	-	13,200,000	13,200,000	-																**-**
Process plant		**27,300,000**	8,190,000	19,110,000	27,300,000	-																**-**
General maintenance work shop		**8,100,000**	-	8,100,000	8,100,000	-																**-**
Vehicle workshop		**15,100,000**	-	15,100,000	15,100,000	-																**-**
BS Supply air building		**4,900,000**	-	4,900,000	4,900,000	-																**-**
Contingency	15%	**14,085,000**	1,228,500	12,856,500	14,085,000	-	-	-	-	-	-	-	-	-	-	-	-					**-**
S/T		**107,985,000**	**9,418,500**	**98,566,500**	**107,985,000**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**					**-**
																						-
Surface Electrical																						**-**
Reconstruction of 50 kV power line		**1,700,000**	1,700,000	-	1,700,000	-																**-**
Power Distribution inc HV Substation		**38,570,500**	19,285,250	19,285,250	38,570,500	-																**-**
Contingency	15%	**6,040,575**	3,147,788	2,892,788	6,040,575	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
S/T		**46,311,075**	**24,133,038**	**22,178,038**	**46,311,075**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	0	0	0	0	**-**
																						-

			Pre-production			Sustaining																
		Total	Year -2	Year -1	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Total
		SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK	SEK
Tailings Dams																						-
Northern Facility (detailed Geotech Investigations/Detailed Design)		**2,000,000**	1,000,000	1,000,000	2,000,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Northern Facility		**68,703,160**	22,905,000	22,905,000	45,810,000	11,446,580	11,446,580	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**22,893,160**
Southern Facility (detailed Geotech Investigations/Detailed Design)		**3,000,000**	-	-	-	-	-	1,500,000	1,500,000	-	-	-	-	-	-	-	-	-	-	-	-	**3,000,000**
Southern Facility		**95,653,750**	-	-	-	-	-	-	-	47,826,875	47,826,875	-	-	-	-	-	-	-	-	-	-	**95,653,750**
Tailings Pipeline		**4,742,760**		4,742,760	4,742,760																	
Contingency	15%	**7,171,500**	3,585,750	3,585,750	7,171,500	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S/T		**181,271,170**	**27,490,750**	**32,233,510**	**59,724,260**	**11,446,580**	**11,446,580**	**1,500,000**	**1,500,000**	**47,826,875**	**47,826,875**	**-**	**-**	**-**	**-**	**-**	**-**	0	0	0	0	**121,546,910**
																						-
Process Plant																						-
Major Equipment		**243,691,135**	65,274,411	152,306,959	217,581,370	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814	2,175,814					**26,109,764**
Minor Equipment & Building		**189,295,792**	56,788,738	132,507,055	189,295,792																	-
EPCM (%age of Total Installed)	15%	**61,031,574**	18,309,472	42,722,102	61,031,574																	-
Spares Holding	10%	**21,758,137**		21,758,137	21,758,137																	-
Contingency	10%	**40,687,716**	12,206,315	28,481,401	40,687,716																	-
S/T		**556,464,355**	**152,578,936**	**377,775,654**	**530,354,590**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	**2,175,814**	0	0	0	0	**26,109,764**
Environmental Compliance & Land Acquisition																						
Diversion River Gonäsån		**8,250,000**	8,250,000	-	8,250,000	-																-
Water & Energy Wells		**1,420,000**	-	560,000	560,000	86,000	86,000	86,000	86,000	86,000	86,000	86,000	86,000	86,000	86,000							**860,000**
Energy well rehabilitation		**5,400,000**			-	540,000	540,000	540,000	540,000	540,000	540,000	540,000	540,000	540,000	540,000							**5,400,000**
Water table monitoring		**2,000,000**	-	2,000,000	2,000,000	-																-
Enforcement Right (EP)		**1,016,875**		1,016,875	1,016,875																	-
Flora & Fauna Compensation scheme		**4,500,000**	-	2,000,000	2,000,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000							**2,500,000**
Land Acquisition		**19,959,544**		14,720,106	14,720,106			5,239,438														**5,239,438**
Land Reclamation Fund		**53,400,000**	15,800,000		15,800,000	3,200,000	3,200,000	3,200,000	3,200,000	3,200,000	3,200,000	3,200,000	3,200,000	3,200,000	3,200,000	3,200,000	2,400,000					**37,600,000**
Mine Closure costs		**50,000,000**	-	-	-													50,000,000				**50,000,000**
Contingency	0%	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
S/T		**145,946,419**	**24,050,000**	**20,296,981**	**44,346,981**	**4,076,000**	**4,076,000**	**9,315,438**	**4,076,000**	**4,076,000**	**4,076,000**	**4,076,000**	**4,076,000**	**4,076,000**	**4,076,000**	**3,200,000**	**2,400,000**	50000000	0	0	-	**101,599,438**
																						-
Shipping costs CIF PORT (% of Main Equipment Cost)	6%	**14,621,468**	14,621,468		14,621,468																	-
Logistics within Sweden (% of Main Equipment Cost)	3%	**15,381,411**	15,381,411		15,381,411																	-
Local Taxes (% of Main Equipment Cost)	0%	**-**	-		-																	-
S/T		**30,002,879**	**30,002,879**	**-**	**30,002,879**	**-**																**-**
																						-
Total (including contingency)		**2,172,529,124**	**601,194,075**	**961,046,104**	**1,562,240,179**	**44,071,219**	**44,071,219**	**82,062,325**	**25,145,564**	**71,472,439**	**127,814,900**	**23,645,564**	**23,645,564**	**69,180,462**	**23,645,564**	**16,214,564**	**9,319,564**	**50,000,000**	**-**	**-**	**-**	**610,288,946**
																						-
Total Contingency		**203,447,603**	57,584,001	98,772,363	156,356,364	3,439,934	3,439,934	9,009,270	2,268,750	2,268,750	9,617,767	2,268,750	2,268,750	8,208,085	2,268,750	1,413,750	618,750	-	-	-	-	**47,091,239**
Total (excluding contingency)		**1,969,081,521**	**543,610,074**	**862,273,741**	**1,405,883,814**	**40,631,285**	**40,631,285**	**73,053,054**	**22,876,814**	**69,203,689**	**118,197,134**	**21,376,814**	**21,376,814**	**60,972,378**	**21,376,814**	**14,800,814**	**8,700,814**	**50,000,000**	**-**	**-**	**-**	**563,197,707**
																						-
Grand Total		**2,172,529,124**	**601,194,075**	**961,046,104**	**1,562,240,179**	**44,071,219**	**44,071,219**	**82,062,325**	**25,145,564**	**71,472,439**	**127,814,900**	**23,645,564**	**23,645,564**	**69,180,462**	**23,645,564**	**16,214,564**	**9,319,564**	**50,000,000**	**-**	**-**	**-**	**610,288,946**
																						-
Checksum		*2,172,529,124*	*601,194,075*	*961,046,104*	*1,562,240,179*	*44,071,219*	*44,071,219*	*82,062,325*	*25,145,564*	*71,472,439*	*127,814,900*	*23,645,564*	*23,645,564*	*69,180,462*	*23,645,564*	*16,214,564*	*9,319,564*	*50,000,000*	*-*	*-*	*-*	*610,288,946*
			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
																						-
USD		**251,940,014**	**69,718,211**	**111,448,894**	**181,167,105**	**5,110,773**	**5,110,773**	**9,516,458**	**2,916,036**	**8,288,389**	**14,822,212**	**2,742,087**	**2,742,087**	**8,022,597**	**2,742,087**	**1,880,342**	**1,080,755**	**5,798,312**	**-**	**-**	**-**	**70,772,909**

Pre-production Capex	**1,562,240,179**	
Sustaining Capex	**610,288,946**	
	2,172,529,124	

Pre-production Capex	**181,167,105**	
Sustaining Capex	**70,772,909**	3.26%
	251,940,014	

Project	**Blötberget Iron Ore Project**						
SEK/US$	**8.6232**						
SEK/EURO €	**9.2869**						
SEK/GB £	**12.7441**						
US$/Euro €	**1.0770**						
		Pre-production	**Sustaining**	**Total**	**Pre-production**	**Sustaining**	**Total**
Underground Mine		**SEK**	**SEK**	**SEK**	**US$**	**US$**	**US$**
Combined access/conveyor surface to -304		110,000,000	-	110,000,000	12,756,285	-	12,756,285
Dewatering UG workings(estimate is between 5 & 9SEK/m³ so assume 7.5)		37,500,000	-	37,500,000	4,348,734	-	4,348,734
Rehabilitation of 320, 370 and ramp240-320 incl. rock support		5,250,000	-	5,250,000	608,823	-	608,823
Conveyor tunnel -304 to -420		22,500,000	-	22,500,000	2,609,240	-	2,609,240
Decline -320 to -420		18,750,000	-	18,750,000	2,174,367	-	2,174,367
Conveyor tunnel -420 to -660		-	55,300,000	55,300,000	-	6,412,933	6,412,933
Decline -420 to -660		-	60,000,000	60,000,000	-	6,957,974	6,957,974
Sandell access, ramp		20,725,000	-	20,725,000	2,403,400	-	2,403,400
Ventilation development - rock excavations		2,500,000	-	2,500,000	289,916	-	289,916
Ventilation raises (diam 4m)		-	17,850,000	17,850,000	-	2,069,997	2,069,997
Ore passes - vertical (diam 3m)		-	31,650,000	31,650,000	-	3,670,331	3,670,331
Workshop - rock excavation		7,995,000	-	7,995,000	927,150	-	927,150
Workshop - installations		5,000,000	-	5,000,000	579,831	-	579,831
Crusher - rock excavation per cavern		3,600,000	10,800,000	14,400,000	417,478	1,252,435	1,669,914
Crusher - haulage drives		2,500,000	7,500,000	10,000,000	289,916	869,747	1,159,662
Auxiliary rock excavations (fuel station, explosives storage etc.)		6,000,000	-	6,000,000	695,797	-	695,797
Conveyor installations at to -420 level (15500SEK/m)		52,605,000	-	52,605,000	6,100,404	-	6,100,404
Conveyor installations at to -500, -580 &-660 level (15500SEK/m)		-	39,480,000	39,480,000	-	4,578,347	4,578,347
Conveyor sustaining Capex (Replacement drift CV)		-	9,847,400	9,847,400	-	1,141,966	1,141,966
Crusher units (includes 2 refurbishments at 50% of capital cost)		9,000,000	18,000,000	27,000,000	1,043,696	2,087,392	3,131,088
Crusher auxiliary (Supporting steel work at 30% cost crusher)		2,700,000	8,100,000	10,800,000	313,109	939,326	1,252,435
Ventilation installation		40,000,000	-	40,000,000	4,638,649	-	4,638,649
Ventilation sustaining		-	15,000,000	15,000,000	-	1,739,493	1,739,493
Power distribution		7,807,891	15,615,783	23,423,674	905,452	1,810,903	2,716,355
Power distribution sustaining		-	11,360,912	11,360,912	-	1,317,482	1,317,482
Permanent dewatering		15,000,000	5,000,000	20,000,000	1,739,493	579,831	2,319,325
Mobile OPEX production fleet - purchase (in OPEX)		-	-	-	-	-	-
Mobile OPEX production fleet - sustaining CAPEX (in OPEX)		-	-	-	-	-	-
Mobile fleet - service units		2,812,500	8,437,500	11,250,000	326,155	978,465	1,304,620
Contingency	15%	55,836,809	47,091,239	102,928,048	6,475,184	5,460,994	11,936,178
S/T		**428,082,200**	**361,032,834**	**789,115,034**	**49,643,079**	**41,867,617**	**91,510,696**
		-	-	-	-	-	-
Surface Minerals Handling		-	-	-	-	-	-
Conveyor surface to ROM Silo		1,550,000	-	1,550,000	179,748	-	179,748
Conveyor surface to Waste Area		13,500,000	-	13,500,000	1,565,544	-	1,565,544
ROM Silo & Reclaim Feeder		6,301,745	-	6,301,745	730,790	-	730,790
Product Conveyors Silos and Loading Equipment		6,942,000	-	6,942,000	805,038	-	805,038
Rail Outloading Product Conveyor, Silos & Discharge Equipment		57,521,533	-	57,521,533	6,670,555	-	6,670,555
Contingency	15%	12,872,292	-	12,872,292	1,492,751	-	1,492,751

		Pre-production	Sustaining	Total	Pre-production	Sustaining	Total
Underground Mine		**SEK**	**SEK**	**SEK**	**US$**	**US$**	**US$**
S/T		**98,687,570**	**-**	**98,687,570**	**11,444,426**	**-**	**11,444,426**
Rail Terminal							
Main Switch		30,000,000	-	30,000,000	3,478,987	-	3,478,987
Linjeplats		40,000,000	-	40,000,000	4,638,649	-	4,638,649
Driftsplats		29,000,000	-	29,000,000	3,363,021	-	3,363,021
Contingency	5%	4,950,000	-	4,950,000	574,033	-	574,033
S/T		**103,950,000**	**-**	**103,950,000**	**12,054,690**	**-**	**12,054,690**
Port							
Port facilities		-	-	-	-	-	-
Contingency	5%	-	-	-	-	-	-
S/T		**-**	**-**	**-**	**-**	**-**	**-**
Industrial Site Civils							
New Roads & Strengthening Old Roads		29,500,000	-	29,500,000	3,421,004	-	3,421,004
Gravel Areas Industrial site 1&2 & earthworks		24,680,000	-	24,680,000	2,862,047	-	2,862,047
Asphalt Areas around Staff buildings		2,000,000	-	2,000,000	231,932	-	231,932
Site Clearance Cut & Fill for Industrial site 1&2		35,478,150	-	35,478,150	4,114,267	-	4,114,267
Bridge		5,250,000	-	5,250,000	608,823	-	608,823
Paintwork and Parking Area identification		25,000	-	25,000	2,899	-	2,899
Industrial Fencing		1,150,000	-	1,150,000	133,361	-	133,361
Underground Pipes (Included in Tata report)		-	-	-	-	-	-
Water Supply		-	-	-	-	-	-
Sewerage Services		-	-	-	-	-	-
Contingency	15%	14,712,473	-	14,712,473	1,706,150	-	1,706,150
S/T		**112,795,623**	**-**	**112,795,623**	**13,080,483**	**-**	**13,080,483**
Industrial Site Buildings							
Office/Locker-room		17,800,000	-	17,800,000	2,064,199	-	2,064,199
Drill core archive		7,500,000	-	7,500,000	869,747	-	869,747
Stores		13,200,000	-	13,200,000	1,530,754	-	1,530,754
Process plant		27,300,000	-	27,300,000	3,165,878	-	3,165,878
General maintenance work shop		8,100,000	-	8,100,000	939,326	-	939,326
Vehicle workshop		15,100,000	-	15,100,000	1,751,090	-	1,751,090
BS Supply air building		4,900,000	-	4,900,000	568,235	-	568,235
Contingency	15%	14,085,000	-	14,085,000	1,633,384	-	1,633,384
S/T		**107,985,000**	**-**	**107,985,000**	**12,522,613**	**-**	**12,522,613**
Surface Electrical							
Reconstruction of 50 kV power line		1,700,000	-	1,700,000	197,143	-	197,143
Power Distribution inc HV Substation		38,570,500	-	38,570,500	4,472,875	-	4,472,875
Contingency	15%	6,040,575	-	6,040,575	700,503	-	700,503
S/T		**46,311,075**	**-**	**46,311,075**	**5,370,521**	**-**	**5,370,521**
Tailings Dams							
Northern Facility (detailed Geotech Investigations/Detailed Design)		2,000,000	-	2,000,000	231,932	-	231,932
Northern Facility		45,810,000	22,893,160	68,703,160	5,312,413	2,654,833	7,967,246

		Pre-production	Sustaining	Total	Pre-production	Sustaining	Total	
Underground Mine		**SEK**	**SEK**	**SEK**	**US$**	**US$**	**US$**	
Southern Facility (detailed Geotech Investigations/Detailed Design)		-	3,000,000	3,000,000	-	347,899	347,899	
Southern Facility		-	95,653,750	95,653,750	-	11,092,605	11,092,605	
Tailings Pipeline		4,742,760	-	4,742,760	550,000	-	550,000	
Contingency	15%	7,171,500	-	7,171,500	831,652	-	831,652	
S/T		**59,724,260**	**121,546,910**	**181,271,170**	**6,925,997**	**14,095,337**	**21,021,334**	
Process Plant								
Major Equipment		217,581,370	26,109,764	243,691,135	25,232,091	3,027,851	28,259,942	
Minor Equipment & Building		189,295,792	-	189,295,792	21,951,920	-	21,951,920	
EPCM (%age of Total Installed)	15%	61,031,574	-	61,031,574	7,077,602	-	7,077,602	
Spares Holding	10%	21,758,137	-	21,758,137	2,523,209	-	2,523,209	
Contingency	10%	40,687,716	-	40,687,716	4,718,401	-	4,718,401	
S/T		**530,354,590**	**26,109,764**	**556,464,355**	**61,503,223**	**3,027,851**	**64,531,074**	
Environmental Compliance - Enabling Works								
Diversion River Gonäsån		8,250,000	-	8,250,000	956,721	-	956,721	
Water & Energy Wells		560,000	860,000	1,420,000	64,941	99,731	164,672	
Energy well rehabilitation		-	5,400,000	5,400,000	-	626,218	626,218	
Water table monitoring		2,000,000	-	2,000,000	231,932	-	231,932	
Enforcement Right (EP)		1,016,875	-	1,016,875	117,923	-	117,923	
Flora & Fauna Compensation scheme		2,000,000	2,500,000	4,500,000	231,932	289,916	521,848	
Land Acquisition		14,720,106	5,239,438	19,959,544	1,707,035	607,598	2,314,633	
Land Reclamation Fund		15,800,000	37,600,000	53,400,000	1,832,266	4,360,330	6,192,597	
Mine Closure costs		-	50,000,000	50,000,000	-	5,798,312	5,798,312	
Contingency	0%	-	-	-	-	-	-	
S/T		**44,346,981**	**101,599,438**	**145,946,419**	**5,142,752**	**11,782,104**	**16,924,856**	
Shipping costs CIF PORT (% of Main Equipment Cost)	6%	14,621,468	-	14,621,468	1,695,597	-	1,695,597	
Logistics within Sweden (% of Main Equipment Cost)	3%	15,381,411	-	15,381,411	1,783,724	-	1,783,724	
Local Taxes (% of Main Equipment Cost)	0%	-	-	-	-	-	-	
S/T		**30,002,879**	**-**	**30,002,879**	**3,479,321**	**-**	**3,479,321**	
		-	-	-				
Total (including contingency)		**1,562,240,179**	**610,288,946**	**2,172,529,124**	**181,167,105**	**70,772,909**	**251,940,014**	
Total Contingency		156,356,364	47,091,239	203,447,603	18,132,058	5,460,994	23,593,052	9.4%
Total (excluding contingency)		1,405,883,814	563,197,707	1,969,081,521	163,035,047	65,311,915	228,346,962	
		-	-	-	-	-	-	
Grand Total		1,562,240,179	610,288,946	2,172,529,124	181,167,105	70,772,909	251,940,014	
		-	-	-	-	-	-	
Checksum		*1,562,240,179*	*610,288,946*	*2,172,529,124*	*181,167,105*	*70,772,909*	*251,940,014*	

	Project	**Blötberget Iron Ore Project**																					
		Mining CAPEX estimation																					
		24/03/2015																					
		[US$]	**SEK 8.6232**																				
		EURO €	**SEK 9.2869**																				
		[GB £]	**SEK 12.7441**																				
chksum		**CAPEX item**	*total*	Y-1	Y-2	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
110,000,000	-	Combined access/conveyor surface to -304	*110,000,000*	88,000,000	22,000,000																		
37,500,000	-	Dewatering ug workings(estimate is between 5 & 9SEK/m³ so assume 7.5)	*37,500,000*	30,000,000	7,500,000																		
5,250,000	-	Rehabilitation of 320, 370 and ramp240-320 incl. rock support	*5,250,000*		5,250,000																		
22,500,000	-	Conveyor tunnel -304 to -420	*22,500,000*		22,500,000																		
18,750,000	-	Decline -320 to -420	*18,750,000*		18,750,000																		
55,300,000	-	Conveyor tunnel -420 to -660	*55,300,000*			5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,300,000							
60,000,000	-	Decline -420 to -660	*60,000,000*			6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000	6,000,000								
20,725,000	-	Sandell access, ramp	*20,725,000*	10,362,500	10,362,500																		
2,500,000	-	Ventilation development - rock excavations	*2,500,000*		2,500,000																		
17,850,000	-	Ventilation raises (diam 4m)	*17,850,000*			1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500	1,487,500						
31,650,000	-	Ore passes - vertical (diam 3m)	*31,650,000*			2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500	2,637,500						
7,995,000	-	Workshop - rock excavation	*7,995,000*		7,995,000																		
5,000,000	-	Workshop - installations	*5,000,000*		5,000,000																		
52,605,000	-	Conveyor installations at to -420 level (15500SEK/m)	*52,605,000*		52,605,000																		
39,480,000	-	Conveyor installations at to -500, -580 &-660 level (15500SEK/m)	*39,480,000*					12,600,000			12,600,000			14,280,000									
9,847,400	-	Conveyor sustaining Capex (Replacement drift CV)	*9,847,400*								9,847,400												
14,400,000	-	Crusher - rock excavation per cavern	*14,400,000*		3,600,000			3,600,000			3,600,000			3,600,000									
10,000,000	-	Crusher - haulage drives	*10,000,000*		2,500,000			2,500,000			2,500,000			2,500,000									
27,000,000	-	Crusher units (includes 2 refurbishments at 50% of capital cost)	*27,000,000*		9,000,000			9,000,000			4,500,000			4,500,000									
10,800,000	-	Crusher auxiliary (Supporting steel work at 30% cost crusher)	*10,800,000*		2,700,000			2,700,000			2,700,000			2,700,000									
6,000,000	-	Auxiliary rock excavations (fuel station, explosives storage etc.)	*6,000,000*		6,000,000																		
40,000,000	-	Ventilation installation	*40,000,000*	2,820,000	37,180,000																		
15,000,000	-	Ventilation sustaining	*15,000,000*					5,000,000			5,000,000			5,000,000									
23,423,674	-	Power distribution	*23,423,674*		7,807,891	7,807,891	7,807,891																
11,360,912	-	Power distribution sustaining	*11,360,912*					4,224,303			2,933,545			4,203,064									
20,000,000	-	Permanent dewatering	*20,000,000*		15,000,000			2,500,000			2,500,000												
-	-	Dewatering (See OPEX)																					
-	-	*Mobile OPEX production fleet - purchase (in OPEX)*	-																				
-	-	*Mobile OPEX production fleet - sustaining CAPEX (in OPEX)*	-																				
11,250,000	-	Mobile fleet - service units	*11,250,000*	2,812,500				2,812,500			2,812,500			2,812,500									
686,186,986		(original cost 887,245,000)annual sum:	686,186,986	133,995,000	238,250,391	22,932,891	22,932,891	60,061,803	15,125,000	15,125,000	64,118,445	15,125,000	15,125,000	54,720,564	15,125,000	9,425,000	4,125,000	-	-	-	-	-	-
$5,917,127,618			**$5,917,127,618**	**$1,155,465,684**	**$2,054,480,775**	**$197,754,909**	**$197,754,909**	**$517,924,940**	**$130,425,900**	**$130,425,900**	**$552,906,175**	**$130,425,900**	**$130,425,900**	**$471,866,367**	**$130,425,900**	**$81,273,660**	**$35,570,700**	**$0**	**$0**	**$0**			
				purchase cost	renewal interval																		
			mobile fleet - OPEX units:	[kSEK]	[yrs]	each 5th year:																	
			Jumbo	8,500	5	8,500																	
			Bolter	8,500	5	8,500																	
			Scaler	7,000	5	7,000																	
			Development loader - 10t	4,500	5	4,500																	
			Long-hole rig (4 units)	15,000	5	60,000																	
			Box-cut rig	15,000	7	10,714																	
			production LHD (5 units)	10,000	3	50,000																	
			Trucks - 50t (2 units)	10,000	5	20,000																	
			secondary breakage	3,000	3	3,000																	
			auxiliary fleet	15,000	5	15,000																	
				191,500,000		187,214,286																	
			!! ABOVE INCLUDED IN OPEX - purchase, sustaining, consumables !!																				

	Project	**Blötberget Iron Ore Project**																				
		Surface CAPEX Estimation																				
		24/03/2015																				
		[US$]	**SEK 8.623**			6301745																
		EURO €	**SEK 9.287**																			
		[GB £]	**SEK 12.744**																			
chksum		**CAPEX item**	*total*	Y-1	Y-2	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
1,550,000	-	Conveyor surface to ROM Silo	*1,550,000*		1,550,000																	
13,500,000	-	Conveyor surface to Waste Area	*13,500,000*	10,800,000	2,700,000																	
6,301,745	-	ROM Silo & Reclaim Feeder	*6,301,745*		6,301,745																	
6,942,000	-	Product Conveyors Silos and Loading Equipment	*6,942,000*	5,553,600	1,388,400																	
57,521,533	-	Rail Outloading Product Conveyor, Silos & Discharge Equipment	*57,521,533*	40,265,073	17,256,460																	
			85,815,278		25,744,583																	
		Rail Terminal																				
		Main Switch		30,000,000																		
		Linjeplats		40,000,000																		
		Driftsplats			29,000,000																	
-	-	**Civils**	-			-	-	-	-	-	-	-	-	-	-	-	-					
29,500,000	-	New Roads & Strengthening Old Roads	*29,500,000*		29,500,000																	
24,680,000	-	Gravel Areas Industrial site 1&2 & earthworks	*24,680,000*		24,680,000																	
2,000,000	-	Asphalt Areas around Staff buildings	*2,000,000*		2,000,000																	
35,478,150	-	Site Clearance Cut & Fill for Industrial site 1&2	*35,478,150*	35,478,150																		
5,250,000	-	Bridge	*5,250,000*		5,250,000																	
25,000	-	Paintwork and Parking Area identification	*25,000*	12,500	12,500																	
1,150,000	-	Industrial Fencing	*1,150,000*		1,150,000																	
-	-	Underground Pipes (Included in Tata report)	-			-	-	-	-	-	-	-	-	-	-	-	-					
-	-	**Buildings**	-			-	-	-	-	-	-	-	-	-	-	-	-					
17,800,000	-	Office/Locker-room	*17,800,000*		17,800,000																	
7,500,000	-	Drill core archive	*7,500,000*		7,500,000																	
13,200,000	-	Stores	*13,200,000*		13,200,000																	
27,300,000	-	Process plant	*27,300,000*	8,190,000	19,110,000																	
8,100,000	-	General maintenance work shop	*8,100,000*		8,100,000																	
15,100,000	-	Vehicle workshop	*15,100,000*		15,100,000																	
4,900,000	-	BS Supply air building	*4,900,000*		4,900,000																	
		Surface Electrical																				
1,700,000	-	Reconstruction of 50 kV power line	*1,700,000*	1,700,000																		
-	-				-																	
38,570,500	-	Power Distribution inc HV Substation	*38,570,500*	19,285,250	19,285,250																	
		Tailings Dams																				
2,000,000	-	Northern Facility (detailed Geotech Investigations/Detailed Design)	*2,000,000*	1,000,000	1,000,000																	
68,703,160	-	Northern Facility	*68,703,160*	22,905,000	22,905,000	11,446,580	11,446,580															
3,000,000	-	Southern Facility (detailed Geotech Investigations/Detailed Design)	*3,000,000*					1,500,000	1,500,000													
95,653,750	-	Southern Facility	*95,653,750*		-					47,826,875	47,826,875											
-	-		-																			
-	-		-																			
8,250,000	-	*Diversion River Gonasan*	*8,250,000*	8,250,000																		
86,200,000	-	Mine Closure	*86,200,000*																	86,200,000		
581,875,838		annual sum:	667,691,117	223,439,573	275,433,938	11,446,580	11,446,580	1,500,000	1,500,000	47,826,875	47,826,875	-	-	-	-	-	-	-	-	86,200,000	-	-
$67,477,948		Total US$	**$77,429,622**	**$25,911,445**	**$31,941,036**	**$1,327,417**	**$1,327,417**	**$173,949**	**$173,949**	**$5,546,302**	**$5,546,302**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$9,996,289**	**$0**	**$0**
		93,900,000	939,000.000																			

Y18
-
$0

Blötberget Iron Ore Project
Process Plant CAPEX

Exchange Rates		
SEK/ US$	8.6232	
SEK/EURO	9.2869	
US$/EURO	1.0770	

Pos.	Item	Operating hours	Nominal Throughput	Design Throughput	Imported		Indigenous	Total
		hrs/y	t/h	t/h	EUR	US$	SEK	US$
1	Secondary crusher	4,788	627	690		1,487,500		1,487,500
2-A	HPGR	6,955	950	1045	3,218,646			3,466,375
2-B	HPGR-Set of drives				287,801			309,952
2-C	HPGR-Set of spare rolls				1,374,898			1,480,719
3	Single-deck screen 5mm	6,955	950	1045	232,000			249,856
4	Wet screens 1mm	6,955	636	700	464,000			499,713
5	Rougher (Rg) LIMS	6,955	432	475		292,000		292,000
6	Wet screens 100microns 1	6,955	421	463		1,308,000		1,308,000
7	Regrind Rg LIMS Conc	6,955	210	231		6,723,500		6,723,500
8	Cleaner (Cln) LIMS	6,955	210	231		604,000		604,000
9	Rougher (Rg) spirals	6,955	222	244		522,240		522,240
10	Wet screens 100microns 2	6,955	105	116		436,000		436,000
11	Regrind Rg Spiral Conc	6,955	29	32		3,361,750		3,361,750
12	Hydrosizer (Elutriator)	6,955	75	83		90,400		90,400
13	Cleaner (Cln) spirals	6,955	75	83		522,240		522,240
14	Flotation - conditioning 1	6,955	181	199		111,265		111,265
15	Flotation - conditioning 2	6,955	181	199		72,590		72,590
16	Flotation	6,955	181	199		1,392,000		1,392,000
17	Concentrate thickener	6,955	181	199		35,000		35,000
18	Pressure Filter	6,955	181	199	1,484,400			1,598,649
19	VPA Compressor	6,955	181	199		300,000		300,000
20	Spiral classifier	6,955	192	211	81,432			87,700
21	Tailings thickener	6,955	204	224		280,642		280,642
22	**Total Main Equipment**							**25,232,091**
23	Ancillary equipment cost			12%				3,027,851
24	Erection, piping & cabling cost			15%				3,784,814
25	Civil and Structural Costs			60%				15,139,255
	Total Minor Equipment & buildings etc							**21,951,920**
26	**TOTAL INSTALLED COST OF EQUIPMENT**							**47,184,011**
27	Shipping costs CIF PORT (% of Main Equipment Cost)			0%				0
28	Contingency (% of Total Installed Cost)			10%				4,718,401
29	EPCM (% of Total Installed Cost)			15%				7,077,602
30	Logistics within Sweden (% of Main Equipment Cost)			0%				0
31	Local Taxes (% of Main Equipment Cost)			0%				0
32	Spare Parts (% Main Equipment Costs)			10%				2,523,209
33	**TOTAL INVESTMENT COST**							**61,503,223**
34	Sustenance Capital	1%	of Total Main Equipment					252,321

[SEK]	Pre production -Y1	Production	Total	Remark
Environmental Permit				
Water Table Control				
Water & Energy Wells	560,000	860,000	1,420,000	over 10 years
Potential energy well rehabilitation		5,400,000	5,400,000	over 10 years
Control Programme	2,000,000		2,000,000	
			0	
Enforcement Right (EP)	1,016,875		1,016,875	year 3
			0	
Compensation programme	2,000,000	2,500,000	4,500,000	250k per year
			0	
			0	
			0	
Land Designation			0	
Land acquistition	10,497,544		10,497,544	
			0	
Land acquistition	4,222,563	5,239,438	9,462,000	effective Y3
Total	**20,296,981**	**13,999,438**	**34,296,419**	
			34,296,419	

Wt recovery **45%**

Item/Description	Operating Expenditure $/t_{ore}$	Operating Expenditure $/t_{conc}$
Mining Opex	9.54	21.19
Processing Opex	2.05	4.88
Underground Infrastructure	0.19	0.43
Surface Infrastructure	0.10	0.23
Power (excluding Process)	1.10	2.44
TMF	0.40	0.88
TMF Pipe line & pumps	0.06	0.13
Railway Opex	2.69	5.97
Port handling	1.67	3.71
G&A Costs	1.56	3.48
TOTAL	**19.36**	**43.34**

Project	**Blötberget Iron Ore Project**								
Mining OPEX estimation					8.6232	*SEK/USD*			
24/03/2015									

note: by definition OPEX includes all development on from decline onwards on each production level i.e. footwall drive, prod drives and vent drives. All shafts are CAPEX

SLOS only

8.6232 *SEK/USD*

OPEX

note1: all figures incl maintenance, capex, sust.capex, consumables

note2: contractor costs

		units	unit cost [SEK]	sum LOM	notes	SEK/tonne		Ranges
production drives incl. rock support	[m]	21,807	25,000	545,175,000	*SLOS design March 2015*	11.89	21,807	100 Water
access drives incl. rock support	[m]	4,707	22,500	105,907,500	*SLOS design March 2015*	2.31	4,707	100 Water
foot wall drives incl. rock support	[m]	8,453	22,500	190,192,500	*SLOS design March 2015*	4.15	8,453	
stope production	[ton]	45,870,000			*density: 3.8/m3*			
long-hole drilling	[drilled m]	3,822,500	175	668,937,500	*each ring 2500-5500t, @ 12-15ton/drillm (avg 12), diam 110mm*	14.58		
charging, blasting	[charged m]	1,720,125	150	258,018,750	*powder factor @ 9kg/m = 0.34kg/t i.e. 1.3kg/m3*	5.63		
loading	[ton]	45,870,000	20	917,400,000	*from 12 -> 20 due to remote loading requirements*	20.00		
transport 0-500m	[ton]	-	-	-	*NA when orepasses avalable*			
horizontal haulage	[ton]	50,457,000	1.5	75,685,500	*on the main haulage level, incl waste*	1.65		
crushing	[ton]	50,457,000	-	-	*included in infrastructure costs*			
conveyor transport to surface	[ton]	50,457,000	-	-	*included in infrastructure costs*			
			sub sum:	*2,761,316,750*		60.20		

NIO services		years	annual cost	sum LOM		SEK/tonne
Running cost - power, heating & gasol surface	[annual cost]	15.3	939,000	14,357,310	*1% of Surface Capex*	0.31
Heating UG (gasol)	[annual cost]	15.3	10,000,000	152,900,000	*Power costs in Infrastructure Opex*	3.33
maintenance of ventilation, electrical, water	[annual cost]	15.3	14,500,000	221,705,000	*29 full-time employees (Consumables in infrasturcture Opex)*	4.83
geology, engineering	[annual cost]	15.3	5,500,000	84,095,000	*8 full-time employees + consumables 1.5MSEK*	1.83
overhead	[annual cost]	15.3	3,000,000	45,870,000	*10% of above*	1.00
			sub sum:	*518,927,310*		11.31

	years	annual production	sum LOM				**available tonnes**
Ore production rate	15.3	3,000,000	45,870,000	*revised tonnage estimate March -15 -660 in-situ: 41.66Mt*	in-situ	45,870,000	45,870,000
					ore rec	100%	45,870,000
					dilution	0%	**45,870,000**

		[SEK / ton ore]	[USD / ton ore]	[USD / ton conc]
OPEX (excl NIO)		*60.20*	*6.98*	*15.51*
OPEX (NIO)		*11.31*	*1.31*	*2.92*
OPEX (total)		71.51	8.29	*18.43*
Contingency	15%	10.73	1.24	*2.76*
Mining Total OPEX		82.24	9.54	21.19

	[USD / ton conc]			
OPEX (total)	21.19	*assuming weight recovery 45%*	**Recovery**	**45%**

Notes

Using backfilling to SLOS stopes, can waste handling costs be decreased by range of 26.5SEK/ton waste.

This translates to a cost saving of 2.65SEK/ton ore ($0.31/ton ore)

	SEK/tonne	US$/tonne ore	SEK/tonne	US$/tonne ore
production drives incl. rock support	11.89	1.38	26.41	3.06
access drives incl. rock support	2.31	0.27	5.13	0.60
foot wall drives incl. rock support	4.15	0.48	9.21	1.07
long-hole drilling	14.58	1.69	32.41	3.76
charging, blasting	5.63	0.65	12.50	1.45
loading	20.00	2.32	44.44	5.15
horizontal haulage	1.65	0.19	3.67	0.43
Running cost - power, heating & gasol surface	0.31	0.04	0.70	0.08
Heating UG (gasol)	3.33	0.39	7.41	0.86
maintenance of ventilation, electrical, water	4.83	0.56	10.74	1.25
geology, engineering	1.83	0.21	4.07	0.47
overhead	1.00	0.12	2.22	0.26
contingency	10.73	1.24	23.84	2.76
Total	**82.24**	**9.54**	**182.75**	**21.19**

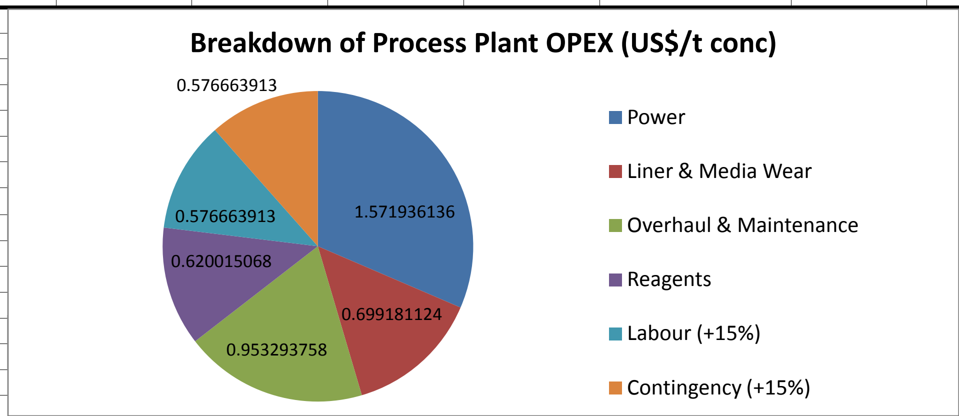
Blötberget Iron Ore Project

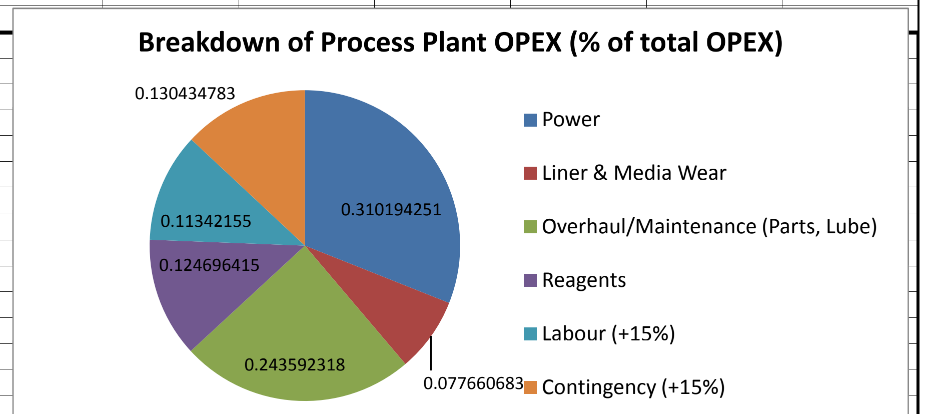
Process Plant OPEX

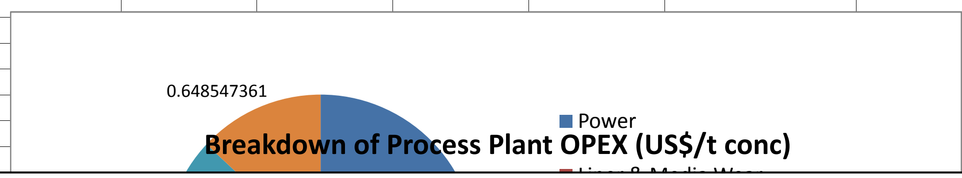
		Recovery		
Plant Feed:	431 tph nominal	45%		2,997,691
Concentrate Production:	181 tph nominal			
Cost of Power:	0.5 SEK/kWh			
Exchange Rate:	8.6232 US$/SEK			
	1.0770 EUR/US$			431.3319531
Media wear :	0.03 kg/kWh	Collector cost:	5.7 US$/kg	
Liner wear:	0.01 kg/kWh	Dispersant cost:	0.4 US$/kg	
Grinding media cost:	0.86 US$/kg	Frother cost:	2 US$/kg	
Liner cost:	37,800 US$/VTM/y	Polymer cost:	4.5 US$/kg	

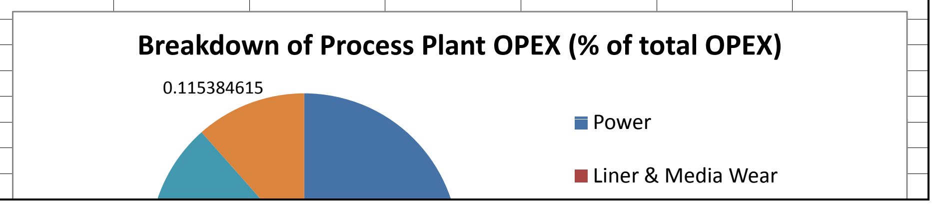
Pos.	Item	Operating hours hrs/y	Nominal Throughput t/h	POWER				LINER & MEDIA WEAR				OVERHAUL&MAINTENANCE (Parts+Lube)		REAGENTS					
				Power Installed kW	Load Factor	Average Power Draw kW	Specific Energy kWh/t	Power kW	Liner Wear rate kg/h	Media Wear rate kg/h	Cost US$/h	Specific Cost US$/t	Cost US$/h	Throughput t/h	Dispersant dosage g/t	Collector dosage g/t	Frother dosage g/t	Polymer dosage g/t	Cost US$/h
1	Secondary crusher	4,788	627	355	0.85	208	0.57				8.48	0.056	34.82						
2	HPGR	6,955	950	1,700		1,283	1.35				52.03	0.015	14.09						
3	Single-deck screen 5mm	6,955	950	44	0.85	37	0.05					0.004	3.74						
4	Wet screens 1mm	6,955	636	110	0.85	94	0.17					0.012	7.48						
5	Rougher (Rg) LIMS	6,955	431	30	0.85	26	0.07					0.005	2.10						
6	Wet screens 100microns 1	6,955	421	11.4	0.85	10	0.03					0.045	18.81						
7	Regrind Rg LIMS Conc	6,955	210	2,240		1,453	6.92	1453		43.6	48.4								
8	Cleaner (Cln) LIMS	6,955	210	60	0.85	51	0.29					0.021	4.34						
9	Rougher (Rg) spirals	6,955	222	0	0.85	0	0.00					0.017	3.75						
10	Wet screens 100microns 2	6,955	105	3.8	0.85	3.2	0.04					0.059	6.27						
11	Regrind Rg Spiral Conc	6,955	29	1,120		320	11.00	320		9.6	13.7								
12	Hydrosizer (Elutriator)	6,955	75	0	0.85	0	0.00					0.009	0.65						
13	Cleaner (Cln) spirals	6,955	75	0	0.85	0	0.00					0.050	3.75						
14	Flotation - conditioning 1	6,955	181	7.5	0.85	6.4	0.04					0.004	0.80	181	500				36.2
15	Flotation - conditioning 2	6,955	181	5.5	0.85	4.7	0.03					0.003	0.52	181		50			51.6
16	Flotation	6,955	181	148	0.85	126	0.82					0.035	6.26	181			10		3.6
17	Concentrate thickener	6,955	181	10	0.85	8.5	0.06					0.001	0.25	181				20	16.3
18	Pressure Filter	6,955	181	200	0.30	60	1.11					0.311	56.22						
19	VPA Compressor	6,955	181	900	0.50	450	4.97					0.012	2.16						
20	Spiral classifier	6,955	192	7.5	0.85	6.4	0.04					0.009	1.70						
21	Tailings thickener	6,955	204	10	0.85	8.5	0.05					0.010	2.02	204				5	4.6
	Pumps and other	10%		696	0.85	592													
	Total			**7,659**		**4,746**					122.6		169.7						112.2

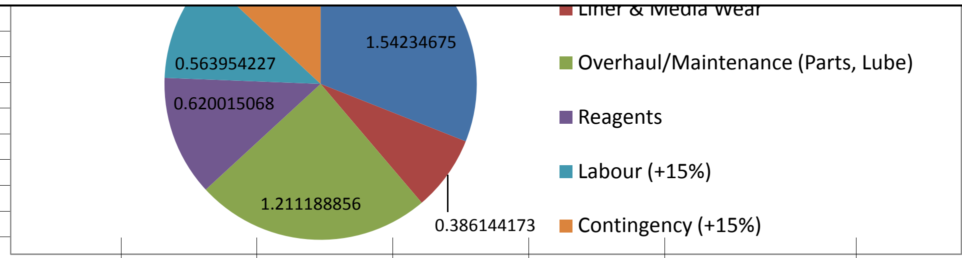
Process Plant OPEX - Breakdown	US$/t ore	US$/t conc	% of total
Power	0.64	1.52	31%
Liner & Media Wear	0.28	0.68	14%
Overhaul & Maintenance	0.39	0.94	19%
Reagents	0.26	0.62	13%
Labour (+15%)	0.24	0.56	12%
Contingency (+15%)	0.24	0.56	12%
Total	**2.05**	**4.88**	**100%**

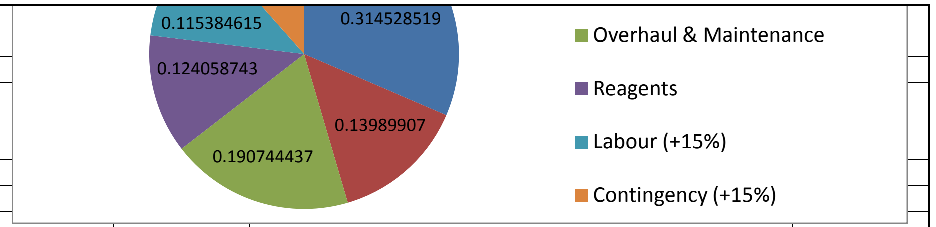












VTM1			
Power, $/y	586,053		
Media, $/y	261,245		
Liner, $/y	75,600	US$/h	US$/t
	922,898	132.7	0.63

	Project	**Blötberget Iron Ore Project**																						
	Infrastructure OPEX estimation																							
	24/03/2015	SEK																						
	[US$]	**SEK 8.623**																						
	EURO €	**SEK 9.287**																						
	[GB £]	**SEK 12.744**																						
	LOMP ROM Tonnes	**45,870,000**																						
	Wt recovery	**45%**																						
Total	**Opex item**		*Factor*	*LOM Total*	Y-1	Y-2	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
9,000,000	Dewatering UG workings	*20,000,000*	*3%*	*9,000,000*			600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000	600,000			
30,000,000	Ventilation	*40,000,000*	*5.0%*	*30,000,000*			2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000	2,000,000			
1,125,000	Workshop Maintenance	*5,000,000*	*1.5%*	*1,125,000*			75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000	75,000			
20,719,125	UG Conveyor installations	*92,085,000*	*1.5%*	*20,719,125*			1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275	1,381,275			
8,505,000	UG Crusher installations	*37,800,000*	*1.5%*	*8,505,000*			567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000	567,000			
2,475,000	Services/ 100mm diam Pipes	*11,000,000*	*1.5%*	*2,475,000*			165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000			
5,270,327	Electrical Power distribution	*23,423,674*	*1.5%*	*5,270,327*			351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355	351,355			
	Underground Total			***77,094,452***			***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	***5,139,630***	-	-	-
21,127,500	Surface Infrastructure	*93,900,000*	*1.5%*	*21,127,500*			1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500	1,408,500			
19,308,438	Rail Outloading Product Conveyor, Silos & Discharge Equipment	*85,815,278*	*1.5%*	*19,308,438*			1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229	1,287,229			
	Surface Total			***40,435,938***			***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	***2,695,729***	-	-	-
72,290,971	*TMF North Dams*	*72,290,971*		*72,290,971*			10,148,267	10,148,267	10,148,267	10,148,267	10,148,267	10,148,267	1,266,819	1,266,819	1,266,819	1,266,819	1,266,819	1,266,819	1,266,819	1,266,819	1,266,819			
84,454,587	*TMF South Dams*	*84,454,587*		*84,454,587*			-	-	-	-	-	-	9,383,843	9,383,843	9,383,843	9,383,843	9,383,843	9,383,843	9,383,843	9,383,843	9,383,843	-	-	
	TMF Total			***156,745,558***			***10,148,267***	***10,148,267***	***10,148,267***	***10,148,267***	***10,148,267***	***10,148,267***	***10,650,662***	***10,650,662***	***10,650,662***	***10,650,662***	***10,650,662***	***10,650,662***	***10,650,662***	***10,650,662***	***10,650,662***	-	-	-
	Power																							
433,935,195	Power Consumption (0.57 SEK/kwhr)	*433,935,195*		*433,935,195*	9,811,138	12,263,923	25,357,966	25,357,966	26,829,637	26,829,637	26,829,637	28,278,988	28,278,988	28,278,988	29,020,479	29,020,479	29,020,479	29,020,479	29,020,479	25,357,966	25,357,966			
				708,211,143																				
699,211,143				708,211,143	9,811,138	12,263,923	43,341,592	43,341,592	44,813,263	44,813,263	44,813,263	46,262,614	46,765,009	46,765,009	47,506,500	47,506,500	47,506,500	47,506,500	47,506,500	43,843,987	43,843,987	-	-	-
$81,084,881	**Total US$**			**82,128,577**	**$1,137,761**	**$1,422,201**	**$5,026,161**	**$5,026,161**	**$5,196,825**	**$5,196,825**	**$5,196,825**	**$5,364,901**	**$5,423,162**	**$5,423,162**	**$5,509,150**	**$5,509,150**	**$5,509,150**	**$5,509,150**	**$5,509,150**	**$5,084,422**	**$5,084,422**	**$0**	**$0**	**$0**

	LOMP Cost (SEK)	**SEK/t ore**	**SEK/t conc**	**US$/t ore**	**US$/t conc**
Underground Infrastructure	77,094,452	1.68	3.73	0.19	0.43
Surface Infrastructure	40,435,938	0.88	1.96	0.10	0.23
Power (excluding Process Plant)	433,935,195	9.46	21.02	1.10	2.44
TMF	156,745,558	3.42	7.59	0.40	0.88
Railway		23.18	51.50	2.69	5.97
Port		14.40	32.00	1.67	3.71
LOMP Average Total Opex	**708,211,143**	**40.99**	**91.09**	**4.75**	**10.56**

[US$]	SEK 8.623	
EURO €	SEK 9.287	
[GB £]	SEK 12.744	
LOMP ROM Tonnes	SEK 1.077	
Wt recovery	45%	
ROM	3,000,000	tpa

OH costs

Positions	No	staff cost	office costs	year cost	
CEO	1	2,700,000	50%	4,050,000	
CFO	3	3,240,000	50%	4,860,000	
HR	2	1,620,000	50%	2,430,000	
Production	1	1,080,000	50%	1,620,000	
Procurement	3	3,240,000	50%	4,860,000	
Sales& Logistics	3	3,240,000	50%	4,860,000	
Geology	2	1,800,000	50%		Included in Miing Opex
HSE	3	3,240,000	50%	4,860,000	
Construction& Development	2	2,160,000	50%	3,240,000	
Total	**20**	**22,320,000**		**30,780,000**	

Administation costs	yearly	
Employer organisation	300,000	
IT support	1,000,000	
IT license fees	1,500,000	
land lease	400,000	
Security	1,500,000	
board fees	2,000,000	
Insurances	3,000,000	
Mineral royalties		0,2% on FOB price
	9,700,000	

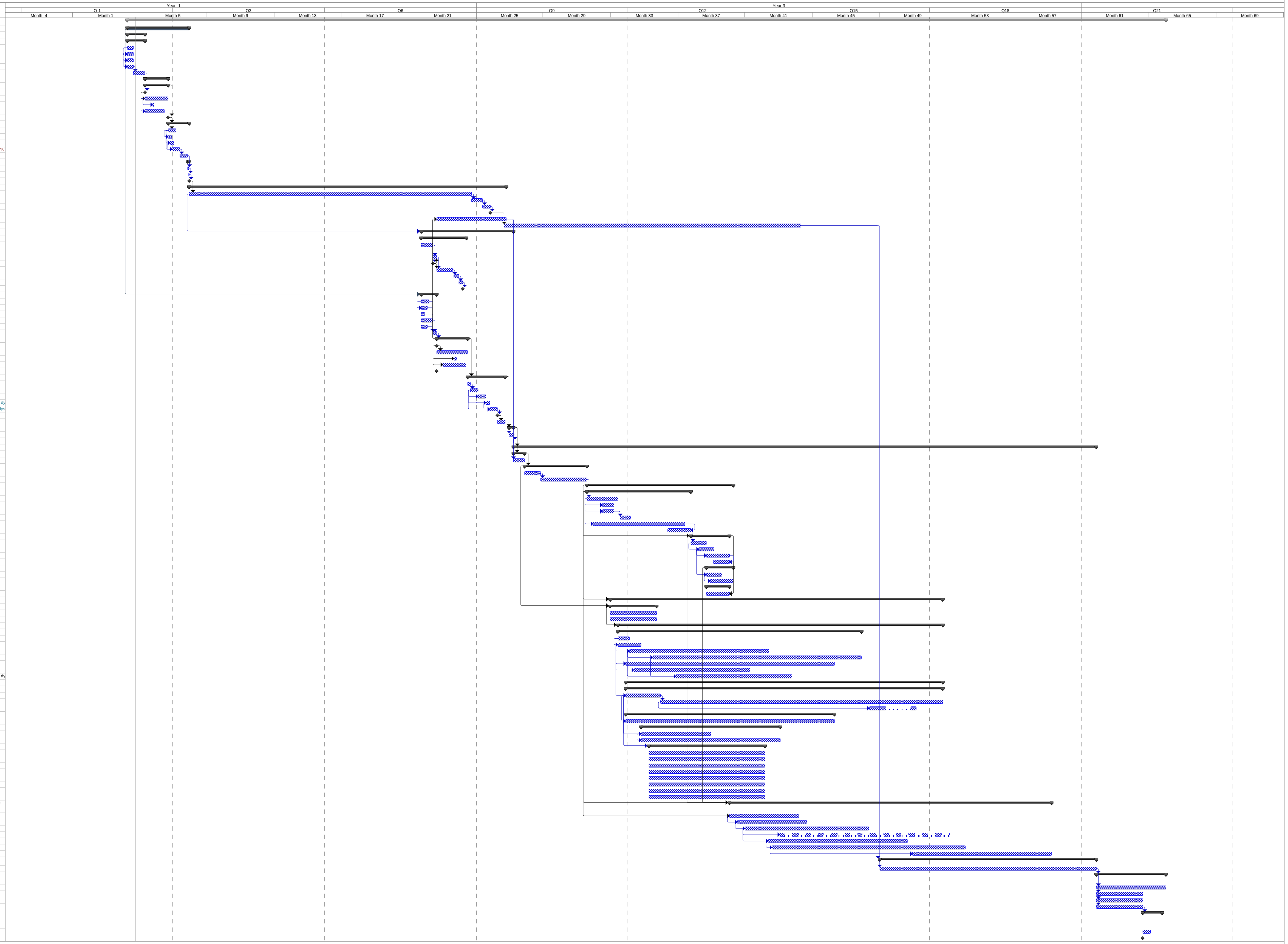
	SEK/t ore	SEK/t conc	US$/t ore	US$/t conc
OH Costs	10.26	22.80	1.19	2.64
Admin Costs	3.23	7.19	0.37	0.83
	13.49	29.99	1.56	3.48

Appendix J

PROJECT EXECUTION SCHEDULE

Nordic Iron Ore Blötberget Mine
Project Development Schedule

ID	Task Name	Duration	Start	Finish	Predecessors
0	**Project Schedule _Initial Draft PS_V1**	**1342 dys**	**Fri 10/04/15**	**Mon 01/06/20**	
1	**Phase 1 - Award of Definitive Feasibility Study**	**80 dys**	**Fri 10/04/15**	**Thu 30/07/15**	
2	**Preperation of ToR & RfP - DFS**	**22 dys**	**Fri 10/04/15**	**Mon 11/05/15**	
3	**Request for Proposal(RfP)- Preparation (O/E)**	**22 dys**	**Fri 10/04/15**	**Mon 11/05/15**	
4	Instructions to Consultants	7 dys	Fri 10/04/15	Mon 20/04/15	
5	Scope of Services	7 dys	Fri 10/04/15	Mon 20/04/15	4SS
6	Conditions of Contract	7 dys	Fri 10/04/15	Mon 20/04/15	5SS
7	Proposal Formats	7 dys	Fri 10/04/15	Mon 20/04/15	6SS
8	Client Approval	15 dys	Tue 21/04/15	Mon 11/05/15	7
9	**Supervision of DFS Tender Process (O/E)**	**30 dys**	**Mon 11/05/15**	**Mon 22/06/15**	
10	**Proposal Period**	**30 dys**	**Mon 11/05/15**	**Mon 22/06/15**	
11	Issue RfP for DFS	0 dys	Mon 11/05/15	Mon 11/05/15	8
12	Proposal Period	30 dys	Tue 12/05/15	Mon 22/06/15	11SS
13	Bidders Meeting	2 dys	Tue 26/05/15	Wed 27/05/15	12SS+10 dys
14	Managing and Responding to Clarifications	25 dys	Tue 12/05/15	Mon 15/06/15	12SS
15	**Proposal Submission**	**0 dys**	**Mon 22/06/15**	**Mon 22/06/15**	**10**
16	**Evaluation of Proposals in response to ITT (O/E)**	**28 dys**	**Tue 23/06/15**	**Thu 30/07/15**	**15**
17	Technical Evaluation	10 dys	Tue 23/06/15	Mon 06/07/15	15
18	Contractual/Commercial Evaluation	5 dys	Tue 23/06/15	Mon 29/06/15	17SS
19	Financial Evaluation	5 dys	Fri 26/06/15	Thu 02/07/15	18SS,17SS+3 dys
20	Evaluation Report Preparation & Submission	10 dys	Tue 30/06/15	Mon 13/07/15	17SS+5 dys,18SS+2 dys,
21	Client Approval	10 dys	Tue 14/07/15	Mon 27/07/15	20
22	**Engage Consultant for DFS study**	**3 dys**	**Tue 28/07/15**	**Thu 30/07/15**	
23	Negotiation of Consultant Contract	2 dys	Tue 28/07/15	Wed 29/07/15	21
24	Award of Contract and Signing	1 dy	Thu 30/07/15	Thu 30/07/15	23
25	**Contract Effective**	**0 dys**	**Thu 30/07/15**	**Thu 30/07/15**	**24**
26	**Phase 2- DFS Study Period**	**410 dys**	**Fri 31/07/15**	**Thu 23/02/17**	
27	DFS Study	365 dys	Fri 31/07/15	Thu 22/12/16	25
28	DFS Draft Report Submission, Review and Presentation	**14 dys**	**Fri 23/12/16**	**Wed 11/01/17**	**27**
29	Clinet Review and Comments	10 dys	Thu 12/01/17	Wed 25/01/17	28
30	DFS Final Report and Submission	**7 dys**	**Thu 26/01/17**	**Fri 03/02/17**	**29**
31	**Gonasan River Diversion**	**90 dys**	**Fri 21/10/16**	**Thu 23/02/17**	**49SS**
32	**Mine De-Watering**	**385 dys**	**Mon 20/02/17**	**Fri 10/08/18**	**30FS+10 dys**
33	**Phase 3 - Engagement of Mine Development Contractor**	**119 dys**	**Fri 23/09/16**	**Wed 08/03/17**	**27SS+300 dys**
34	**Prequalification of Bidders**	**58 dys**	**Fri 23/09/16**	**Tue 13/12/16**	
35	Prepare Schedule of Requirements, Criteria and Prequalification Documents	15 dys	Fri 23/09/16	Thu 13/10/16	
36	Clients Approval of Prequalification Documents	5 dys	Fri 14/10/16	Thu 20/10/16	37,35
37	Publish Notice for Prequalification	0 dys	Thu 13/10/16	Thu 13/10/16	35
38	Period for Prequualification	21 dys	Fri 21/10/16	Fri 18/11/16	36,37
39	Evaluation of Submissions	7 dys	Mon 21/11/16	Tue 29/11/16	38
40	O/E Report & Reccommendations for Shortlisted bidders	5 dys	Wed 30/11/16	Tue 06/12/16	39
41	Client's Approval of pre-quaified bidders	5 dys	Wed 07/12/16	Tue 13/12/16	40
42	**Prepare Invitations to Tender Documents**	**20 dys**	**Fri 23/09/16**	**Thu 20/10/16**	**1SS+350 dys**
43	Insructions to Tenderers	10 dys	Fri 23/09/16	Thu 06/10/16	
44	Conditions of Contract	7 dys	Fri 23/09/16	Mon 03/10/16	43SS
45	Schedule of Requirements	5 dys	Fri 23/09/16	Thu 29/09/16	
46	Scope of Work	15 dys	Fri 23/09/16	Thu 13/10/16	
47	Tender Formats	7 dys	Fri 23/09/16	Mon 03/10/16	
48	Client Approval of ITT	5 dys	Fri 14/10/16	Thu 20/10/16	43,44,45,46,47
49	**Tendering Process**	**40 dys**	**Thu 20/10/16**	**Thu 15/12/16**	**48**
50	**Issue Tender Documents to Qualified Tenderers**	**0 dys**	**Thu 20/10/16**	**Thu 20/10/16**	
51	Tender Period	40 dys	Fri 21/10/16	Thu 15/12/16	50
52	Tenderers Site Visit & Conference	5 dys	Mon 21/11/16	Fri 25/11/16	50SS+21 dys
53	Clarification Period	30 dys	Tue 01/11/16	Mon 12/12/16	50SS+7 dys
54	Tender Submission Deadline	0 dys	Thu 20/10/16	Thu 20/10/16	
55	**Tender Evaluation**	**48 dys**	**Fri 16/12/16**	**Tue 21/02/17**	**49**
56	Preliminary Examination of Tenders for Compliance	3 dys	Fri 16/12/16	Tue 20/12/16	
57	Technical Evaluation	10 dys	Wed 21/12/16	Tue 03/01/17	56
58	Commercial/Contractual Evaluation	10 dys	Wed 04/01/17	Tue 17/01/17	57SS+10 dys
59	Financial Evaluation	5 dys	Wed 18/01/17	Tue 24/01/17	57SS+15 dys,58SS+10 dy
60	Tender Evaluation Report Preparation	10 dys	Wed 25/01/17	Tue 07/02/17	57SS+15 dys,58SS+5 dys
61	**Tender Evaluation Report Submission**	**0 dys**	**Tue 07/02/17**	**Tue 07/02/17**	**60**
62	Client Review and Approval	10 dys	Wed 08/02/17	Tue 21/02/17	61
63	**Contract Award and Effectiveness**	**6 dys**	**Wed 01/03/17**	**Wed 08/03/17**	**55**
64	Contract Negotiation	5 dys	Wed 01/03/17	Tue 07/03/17	62FS+5 dys
65	**Contract Award**	**1 dy**	**Wed 08/03/17**	**Wed 08/03/17**	**64**
66	**Mine Development Phase (MD Contractor)**	**753 dys**	**Thu 09/03/17**	**Mon 27/01/20**	**63**
67	**Project Kick Off Period**	**14 dys**	**Thu 09/03/17**	**Tue 28/03/17**	**63**
68	DFS data review	14 dys	Thu 09/03/17	Tue 28/03/17	31
69	**Contractor Mobilisation**	**81 dys**	**Wed 29/03/17**	**Wed 19/07/17**	**67**
70	Site Establishment	21 dys	Wed 29/03/17	Wed 26/04/17	
71	Preliminary Works (access roads etc)	60 dys	Thu 27/04/17	Wed 19/07/17	70
72	**Mine Design & Engineering**	**189 dys**	**Thu 20/07/17**	**Tue 10/04/18**	
73	**Mine**	**134 dys**	**Thu 20/07/17**	**Tue 23/01/18**	
74	Mine Planning	40 dys	Thu 20/07/17	Wed 13/09/17	71
75	Development Systems	15 dys	Thu 17/08/17	Wed 06/09/17	74SS+20 dys
76	Mining Systems	15 dys	Thu 17/08/17	Wed 06/09/17	74SS+20 dys
77	Mine Services (Power, Water, Air)	15 dys	Mon 18/09/17	Fri 06/10/17	76FS+7 dys
78	Mine Design & Engineering	120 dys	Mon 31/07/17	Fri 12/01/18	74SS+7 dys
79	Equipment & Systems Specifications	30 dys	Wed 13/12/17	Tue 23/01/18	78FF+7 dys
80	**Mine Surface**	**50 dys**	**Wed 24/01/18**	**Tue 03/04/18**	**73SS+30 dys**
81	Surface facilities	20 dys	Wed 24/01/18	Tue 20/02/18	79
82	Surface layout	20 dys	Wed 07/02/18	Tue 06/03/18	81SS+10 dys
83	Surface Design & Engineering	30 dys	Wed 21/02/18	Tue 03/04/18	82SS+10 dys
84	Equipment, Facilities & Systems Specifications	21 dys	Tue 06/03/18	Tue 03/04/18	83FF
85	**Interfaces**	**35 dys**	**Wed 21/02/18**	**Tue 10/04/18**	
86	Underground - Surface	20 dys	Wed 21/02/18	Tue 20/03/18	82SS+10 dys
87	Surface - Access Points	30 dys	Wed 28/02/18	Tue 10/04/18	86SS+5 dys
88	**Environmental**	**30 dys**	**Wed 21/02/18**	**Tue 03/04/18**	
89	Environmental Management Plans	30 dys	Wed 21/02/18	Tue 03/04/18	80FF
90	**Construction & Implementation Phase**	**430 dys**	**Thu 31/08/17**	**Wed 24/04/19**	**72SS+30 dys**
91	**Mobilisation**	**60 dys**	**Thu 31/08/17**	**Wed 22/11/17**	**69SS+15 dys**
92	Personnel Deployment	60 dys	Thu 31/08/17	Wed 22/11/17	
93	Construction Equipment Deployment	60 dys	Thu 31/08/17	Wed 22/11/17	
94	**Contrstruction**	**420 dys**	**Thu 14/09/17**	**Wed 24/04/19**	**91SS+10 dys**
95	**Surface**	**315 dys**	**Thu 14/09/17**	**Wed 28/11/18**	
96	Site Access	15 dys	Thu 14/09/17	Wed 04/10/17	
97	Site Clearance	30 dys	Thu 14/09/17	Wed 25/10/17	96SS
98	Foundation Works	180 dys	Thu 05/10/17	Wed 13/06/18	97SS+15 dys
99	Buiding & Plant Construction	270 dys	Thu 16/11/17	Wed 28/11/18	98SS+30 dys
100	Site Road Construction	270 dys	Thu 28/09/17	Wed 10/10/18	97SS+10 dys
101	Site Drainage construction	150 dys	Fri 13/10/17	Thu 10/05/18	97SS+21 dys
102	Mechanical Services/Electrical Services	150 dys	Thu 28/12/17	Wed 25/07/18	98SS+30 dys,99SS+30 dy
103	**Underground - Drift access**	**410 dys**	**Thu 28/09/17**	**Wed 24/04/19**	
104	**Drift No 1**	**410 dys**	**Thu 28/09/17**	**Wed 24/04/19**	
105	Drift No 1 Portal Construction	45 dys	Thu 28/09/17	Wed 29/11/17	97SS+10 dys
106	**Drift Driveage**	365 dys	Thu 30/11/17	Wed 24/04/19	105
107	Junction Construction	30 dys	Thu 13/12/18	Thu 07/03/19	106SS+270 dys
108	**Underground Development**	**270 dys**	**Thu 28/09/17**	**Wed 10/10/18**	
109	Underground Roadway Development	270 dys	Thu 28/09/17	Wed 10/10/18	105SS
110	**Mining Area(s) Development**	**180 dys**	**Thu 26/10/17**	**Wed 04/07/18**	
111	Establish working areas	90 dys	Thu 26/10/17	Wed 28/02/18	105SS+20 dys
112	Ore Production	180 dys	Thu 26/10/17	Wed 04/07/18	111SS
113	**Mine Services Installation**	**150 dys**	**Thu 09/11/17**	**Wed 06/06/18**	**105SS+30 dys**
114	Ventilation	150 dys	Thu 09/11/17	Wed 06/06/18	
115	Power	150 dys	Thu 09/11/17	Wed 06/06/18	
116	Water	150 dys	Thu 09/11/17	Wed 06/06/18	
117	Air	150 dys	Thu 09/11/17	Wed 06/06/18	
118	Pumping/Drainage	150 dys	Thu 09/11/17	Wed 06/06/18	
119	Monitoring & Communication Systems	150 dys	Thu 09/11/17	Wed 06/06/18	
120	Iron Ore Processing	150 dys	Thu 09/11/17	Wed 06/06/18	
121	Material & Personnel Transport Systems	150 dys	Thu 09/11/17	Wed 06/06/18	
122	**Equipment & Plant Procurement, Delivery, Installation & Commissioning**	**417 dys**	**Wed 04/04/18**	**Thu 07/11/19**	**73SS+30 dys,80SS+30 dys,85SS+30 dys**
123	Specification approval	90 dys	Wed 04/04/18	Tue 07/08/18	73SS+30 dys
124	Compatibility/Interface approval	90 dys	Wed 18/04/18	Tue 21/08/18	123SS+10 dys
125	Procurement/Manufacturing Period	160 dys	Wed 02/05/18	Tue 11/12/18	124SS+10 dys
126	Works Inspections & Tests	90 dys	Wed 04/07/18	Tue 07/05/19	125SS+45 dys
127	Shipment/Delivery to Site	180 dys	Wed 13/06/18	Tue 19/02/19	125SS+30 dys
128	Installation	250 dys	Wed 20/06/18	Tue 04/06/19	127SS+5 dys
129	Commissioning	180 dys	Fri 01/03/19	Thu 07/11/19	128SS+30 dys
130	**Iron Ore pre-production**	**280 dys**	**Tue 01/01/19**	**Mon 27/01/20**	**32**
131	Performance Tests	280 dys	Tue 01/01/19	Mon 27/01/20	32
132	**O/E - Monitoring and Supervision of operations during defects liability period.**	**90 dys**	**Tue 28/01/20**	**Mon 01/06/20**	**131**
133	Compliance Monitoring(OE)	90 dys	Tue 28/01/20	Mon 01/06/20	**131**
134	Daily Activity Reports (OE)	60 dys	Tue 28/01/20	Mon 20/04/20	**131**
135	Daily Event Log(OE)	60 dys	Tue 28/01/20	Mon 20/04/20	**131**
136	Daily Control Logs(OE)	60 dys	Tue 28/01/20	Mon 20/04/20	**131**
137	**Preparation and Handover of Project completion reports, Project correspondence and As-Built drawings**	**25 dys**	**Tue 21/04/20**	**Mon 25/05/20**	**136**
138	As-built drawing report(OE)	10 dys	Tue 21/04/20	Mon 04/05/20	
139	**Project Completion Report(OE)**	**25 dys**	**Tue 21/04/20**	**Mon 25/05/20**	



Project: Project Schedule _Initial Draft
Date: Fri 24/04/15

Task | Milestone | Summary | Rolled Up Task | Rolled Up Milestone | Rolled Up Progress | Split | External Tasks | Project Summary | Group By Summary | Inactive Task | Inactive Task | Inactive Milestone | Inactive Summary | Manual Task | Duration-only | Manual Summary Rollup | Manual Summary | Start-only | Finish-only | Progress | Deadline

Appendix K

RISK REGISTER

Risk Register

KEY to scoring

Risk Matrix

Severity					
5	O	R	R	R	R
4	Y	O	O	R	R
3	Y	Y	Y	Y	Y
2	G	G	Y	Y	Y
1	G	G	G	Y	Y
	1	2	3	4	5

Likelihood

Impact Ratings

Likelihood of Occurance (L)

1. Improbable
2. Unlikely 1-10%
3. Possible 10-50%
4. Likely 50-90%
5. Almost Certain 90%+

Severity of Occurance (S)

Health, Safety and Environment	Reputation	Commercial*
1 Negligible	Deviation from Accepted Standards	1. <USD 100k
2 Minor	Delayed delivery (weeks)	2. USD 100k - USD 1M
3 Major	Major Delay to Project Delivery (months)	3. USD 1M - USD 10M
4 Severe	Severe Delay to Project Delivery (Years)	4. USD 10M - USD 100M
5 Extreme	Inbility to deliver Project	5. >USD 100M
		*Cost or Affect

The risk matrix considers hazards that are associated with the project. It lists the consequences that could arisein the event that the hazard was realised and then allocates a score, which is designed to rate the risk.

All risks other than green should have a mitigation action. An orange or red risk MUST be mitigated before the completion of the study.

Risk #	Area	Activity	Hazard/Risk	Effects	Constraints	Responsible	Assigned To	Pre RCM Likelihood	Pre RCM Severity	Pre Control Risk	RCM	Post RCM Likelihood	Post RCM Severity	Residual Hazards	RECM approved
1	Project wide	Operation	Subsidence - due to mining	Water table; surface infrastructure; local resident relations	Understanding of effect of mining on subsidence	NIO / mining design consultant	Hans Thorshag	2	5	4					
2	Project wide	Operation	Subsidence - due to dewatering	Water table; surface infrastructure; local resident relations	Understanding of effect of dewatering on subsidence	NIO / hydro consultant	Hans Thorshag	2	5	4					
3	Project wide	Operation	Health and safety - poor record	Internal duty to maintain; cost (commitment); changes incur additional costs	Communication; ethics	NIO	Hans Thorshag	3	3	2					
4	Project wide	Operation	Community - Loss of social licence	Lost relations between mine-local population	Communication; ethics	NIO	Hans Thorshag	3	3	2					
16	Project wide	FS	Building permit - delays due to appeals	Delay to DFS	Knowledge of building design; time; cost	NIO / consultants	Hans Thorsdag	2	2	1					
16	Project wide	FS	A definitive list of the required permits, who is to apply for them and on what time-scale is not available. Potential operational risk if the necessary permits are not obtained	Delay to DFS	Knowledge of building design; time; cost	NIO / consultants	Hans Thorsdag	2	3	2					
5	Project wide	Operation	Mining boom - competition	Access to services and personnel; cost inflation		-	-	3	2	2	Competitive salaries				
5	Project wide	Operation	Corporate policy for land acquisition and details of landowner consultation are not yet available. Potential delay if policy and practice do not meet international guidelines.	Access to services and personnel; cost inflation		NIO	Christer Lindqvist	3	2	2	Consultation to complete				
6	Project wide	Operation	European regulations and restrictions - changes	Changes in operations		-	-	3	3	2					
7	Project wide	Selling	Market - lack of buyers for the product	Revenue	Market	NIO	Paul Marsden	2	4	3					
8	Project wide	Selling	Product quality - low quality	Lack of market buyers; FOB price; revenue	Product quality	-	-	2	5	4					
9	Project wide	Selling	Fe selling price - low	Revenue; mine operation	Price	-	-	3	4	3					
10	Environment	H&S	Restrictions - sound, dust (transportation and mining), seismic	Community	Cost; changes in regulations	NIO		3	3	2					
11	Project wide	Operation	Weather & ACTS OF GOD	Cost; time	God	God	God	2	4	3					
12	Infrastructure	Operation	Lack of sufficient bunkerage	Loss of production/revenue	Mining plan	NIO		4	3	2					
13	Ventilation	Operation	Inadequate ventilation	High CO levels	Mining system	NIO		3	3	2					
37	Infrastructure	FS	Rail - too low capacity, upgrading required	Production rate; OPEX; revenue	Railway capacity	NIO	Michael Malmqvist	2	3	2					
14	Transportation	Operation	Single entry transport system	Loss of production; equipment damage; provision of supplies	Mining system	NIO		3	3	2					
15	Engineering	Operation	Single line components	loss of production	Mining system	NIO		3	3	2					
16	Logistics	Operation	Travelling time	Loss of production	Mining system	NIO		3	2	2					
14	Project wide	Constructi	Financing delays	Project delay; procurement and construction delay	Construction	NIO/Lenders		3	3	2					
15	Project wide	Constructi	Delay in tendering process due to procedures or market forces	Project delay; procurement and construction delay	Construction	NIO/Market		3	3	2					
42	Shafts	FS	Shaft stability &/or blockage	Mine design not practicable	Shaft survey	NIO	Hans Thorsdag	3	4	3					
16	Construction	Operation	Dangers in working in pumped out shaft; wall collapse	Cost; time ; construction	Operations	NIO		3	3	2					
40	Project wide	FS	Procurement contracts and long-lead items - delays	Delays to DFS	Cost; time; types	NIO	Paul Marsden	2	3	2					
14	Transportation	Operation	Proposed road use and development have nopt been fully assessed for environmental risks. Need to consider safety, road capacity, air emissions and noise levels associated with proposed traffic for the development	Loss of production; equipment damage; provision of supplies	Mining system	NIO	Hans Thorsdag	3	3	2	Hydrogeological investigation required				
2	Project wide	Operation	Geotechnical - potential risk of roof failure	Working environment	Understanding of effect of dewatering on subsidence	NIO / hydro consultant	Hans Thorshag	2	3	2	Additional data acquisition required				
2	Project wide	Operation	Hydrogeology - potential risk of flooding of UG mine workings	Water table;	Understanding of effect of dewatering on subsidence	NIO / hydro consultant	Hans Thorshag	2	3	2	Additional data acquisition required				
37	Infrastructure	FS	Proposed transmission routes from the national grid. Potential sensitive habitats or populations on the selected route	Production rate; OPEX; revenue	Railway capacity	NIO	Hans Thorsdag	2	3	2					
10	Environment	H&S	Water and effluent management plans are still to confirm. Potential for releases of pollutants from mine water to surface or groundwater.	Community	Cost; changes in regulations	NIO	Hans Thorsdag	3	3	2	Provide water balance for the mining operations, including details of the use of water treatment, water storage. Characterise potential pollutant disharges to surface and groundwater.				
10	Environment	H&S	Still unclear when the river diversion will be undertaken and is necessary for the development of the project. same to assess as part of the EIA. Potential direct and indirect effects on local population, agriculture and ecosystems as a result of river works.	Community	Cost; changes in regulations	NIO	Hans Thorsdag	3	3	2	Confirm when the diversion will be undertaken and whether this has been assessed as part of the EIA.				
41	Closure	FS	Closure costs underestimated	Increased costs (reduced revenue)	Lack of knowledge	NIO / consultants	Hans Thorsdag	3	2	2					

Risk Register

KEY to scoring

Risk Matrix

Severity					
5	O	R	R	R	R
4	Y	O	O	R	R
3	Y	Y	Y	Y	Y
2	G	G	Y	Y	Y
1	G	G	G	Y	Y
	1	2	3	4	5

Likelihood

Impact Ratings

Likelihood of Occurance (L)

1. Improbable
2. Unlikely 1-10%
3. Possible 10-50%
4. Likely 50-90%
5. Almost Certain 90%+

Severity of Occurance (S)

Health, Safety and Environment	**Reputation**	**Commercial***
1 Negligible	Deviation from Accepted Standards	1. <USD 100k
2 Minor	Delayed delivery (weeks)	2. USD 100k - USD 1M
3 Major	Major Delay to Project Delivery (months)	3. USD 1M - USD 10M
4 Severe	Severe Delay to Project Delivery (Years)	4. USD 10M - USD 100M
5 Extreme	Inbility to deliver Project	5. >USD 100M
		*Cost or Affect

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